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G. Guiot
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Binder 1

Materials Technology I

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Chapter 1

Raw Materials Supply for Cement and Aggregate Industry

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Raw Material Supply to Cement and Aggregate Industry

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1. RAW MATERIAL SUPPLY TO CEMENT AND AGGREGATE INDUSTRY

The basis for all business the in cement and aggregate industry is adequate supplies of raw material. This raw material is found in nature in form of rock formations. In order to find and secure sufficient reserves of such rock formations exploration work is conducted. The actual production is gained from open cast mining of a raw material deposit (any volume of rock can represent a raw material deposit) in most cases. Picture 1 depicts a typical quarry for a cement factory. It becomes very evident that the meaning of a quarry is to dig a hole into the landscape. Considering the fact that for the production of clinker or aggregates certain qualities of rock are needed, it is of paramount interest to the plant to know beforehand what type of rock and what quality will be encountered within the mountain, behind the quarry faces.

Picture 1 The Quarry, source of raw materials

Picture 1: The Quarry, source of raw materials

This need for information before mining starts is the classical problem of exploration.

Two main aspects are to be considered:

- Geometry of the raw material deposit, that means geological boundaries like interfaces of formations, faults and also topography
- Quality of the rocks in terms of chemical and mineralogical composition, physical characteristics like hardness, abrasiveness, pozzolanic activity.

Obviously this task of exploration is not a simple one. Our means of acquiring data is limited compared to the large size of rock volumes to be investigated. Exploration drillings provide precise but also very spotty information on the rock volumes. In order to fill the gaps

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between drillholes and also in order to interpret results of drillings, we need a model of the raw material deposit. This model is developed with the help of the natural science GEOLOGY. Within this science, all aspects with respect to rock formation, deformation and transformation are studied and general rules are established. These concepts of geology are absolutely instrumental for the interpretation of any raw material deposit.

The application of geology is, however, also an ambiguous matter. There are hardly two geologists who are of the same opinion on a given raw material deposit. This fact creates confusion especially amongst engineers, who are used to well-defined systems, expressed in precise figures. Exactly these precise figures can never be provided by the geologists, because of the inherent ambiguities of the geology. Why is that so?

Geology is a relatively young natural science. It is part of nature and hence very tricky. The development of this science is not as advanced as, for example, chemistry or physics. The reason for this lies in the complexity of the subject. It represents a combination of every conceivable combination of physical, chemical and biological processes at all scales.

Many phenomenons observed in nature are not understood and related processes are not controlled by scientists. A good example of this fact is the inability to predict earthquakes.

A further reason lies in the fact that geology deals with very different dimensions, from the very tiny atoms and molecules to finally the whole globe. Especially the large dimensions are difficult to overcome.

Additionally geology deals with a very long time span. It is an ambitious task to reconstruct the history of a given rock because of the scarce information. Normally such a history has to be pieced together the geologist is obliged to use his imagination to fill in gaps in the information.

Nevertheless, the geological concepts are the only help in the interpretation of raw material deposits. In order to arrive at the best possible interpretation, it is important to ask the most adequate person for an opinion. Considering the many different specialists which exist in the field of geology (see table 1), the choice is not always easy. Very often different specialists contribute to the erection of a geological model for a raw material deposit. As for instance in the field of medicine, it is important to select the correct specialist for the purpose in mind (nobody would select as gynaecologist for an appendix operation).

Because of the difficulties described, the results of geological studies are often qualitative. For the design of a cement or aggregate plant, however, the engineers require quantitative results. Also from this side emerges a problem of communication between geologists and engineers. This is expressed in the fact that qualitative statements of the geologists are discarded by engineers, and that they the base the design of equipment on shaky but apparently precise figures. The result of this practice can be very costly. It is the aim of the following lessons to improve on this special communication between geologists and engineers.

- The geologists have to learn that quantification is an absolute must in our industry
- The engineers have to learn that no raw material deposit is homogenous and no deposit can be characterized by precise figures alone.

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Table 1

Geological Specialities
Stratigraphy/Biostratigraphy
Igneous Geology
Volcanology
Sedimentology
Structural Geology/Tectonics
Paleontology/Micropaleontology
Palynology
Geochemistry
Hydrology
Geophysics
Oil Geology
Engineering Geology
Paleogeography

TABLE 2

Type of raw materials used in the cement and aggregate industry	
limestone	clinker/lime production aggregates
claystone/shale	clinker production expanded clay
iron-ore	corrective material clinker production
bauxite	corrective material clinker production
laterite	corrective material clinker production
kaolinite	white cement production
gypsum	cement additive
pozzolana	cement additive
sand/gravel	aggregates
crushed stone (granite, diorite, basalt etc.)	aggregates

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Chapter 2

Theoretical Concepts of Geology

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Theoretical concepts of Geology

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1. THEORETICAL CONCEPTS OF GEOLOGY

1.1 General remarks

The following concepts of geology represent a very small selection of available concepts. They should illustrate how the gaps between isolated data points are filled. In a way they represent the essence of the "imagination" a geologist uses to produce a coherent geological model of a raw material deposit. These concepts are derived from observations made on the actual geological process. This ongoing process, although slow in time, can be studied and described and the results can be compared to ancient "documents" in the form of rocks.

There is also much circumstantial information used in the construction of these concepts. By using the existing examples of rock formation, one assumes that the conditions for these processes have not changed with time. This is not entirely true, since certain changes to these general conditions (like for example the polarity and the strength of the earth's magnetic field) have occurred many times in the past. Still, the so-called actualistic model is best suited to explain the features of ancient rocks.

1.2 The structure of the earth

There is enough scientific evidence to assume that the earth consists of a series of zones (Fig. 1) which differ distinctly from on another other in their chemical and physical characteristics. The earth's center is a solid core of nickel and iron, surrounded by a zone of liquid material (liquid core"). The mantle lying between the core and the crust is divided into two sections: the mantle as such and the "upper" mantle. Both are chemically characterized by the abundantly present sulphur-oxygen compounds combined with heavy metals. The crust itself can be divided into two portions, the oceanic crust (Silicon, Magnesium and Iron as main elements) and the continental crust (Silicon and Alumina).

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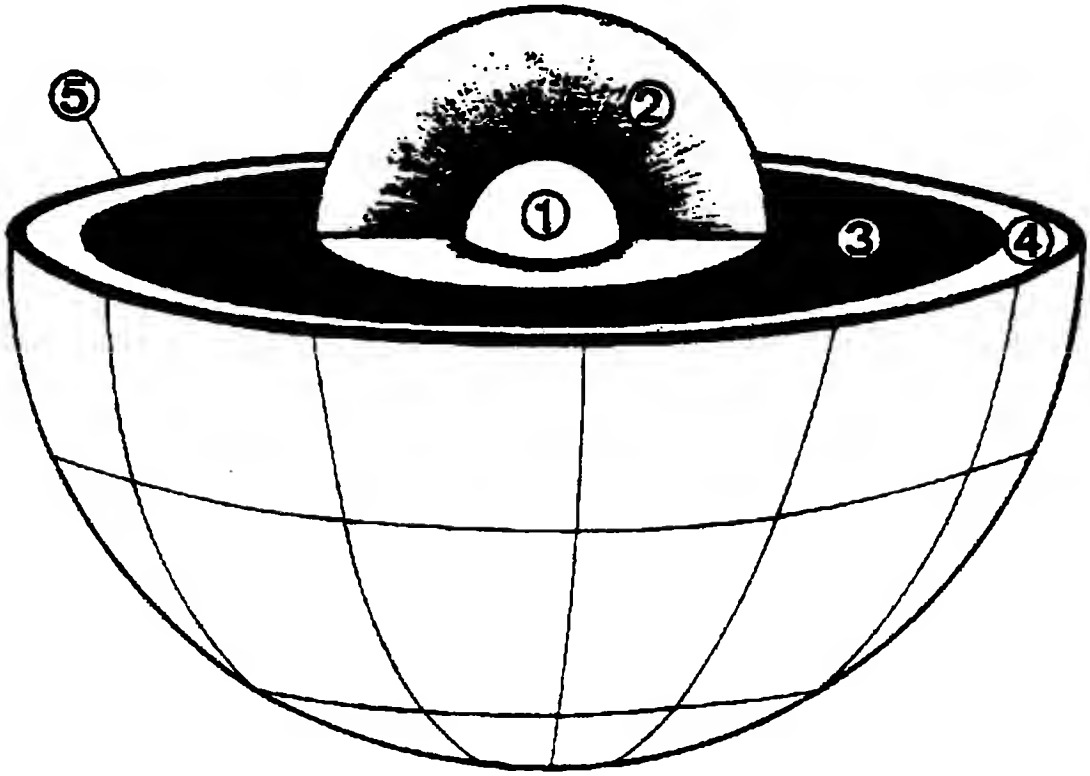
Fig.1 Structure of the Earth



Legend:

① solid core

② liquid core

③ mantle

④ upper mantle

⑤ crust

Compared with the other structural elements of the earth, the crust is a very thin layer, of an average thickness of only 30 - 40 km (Fig. 2).

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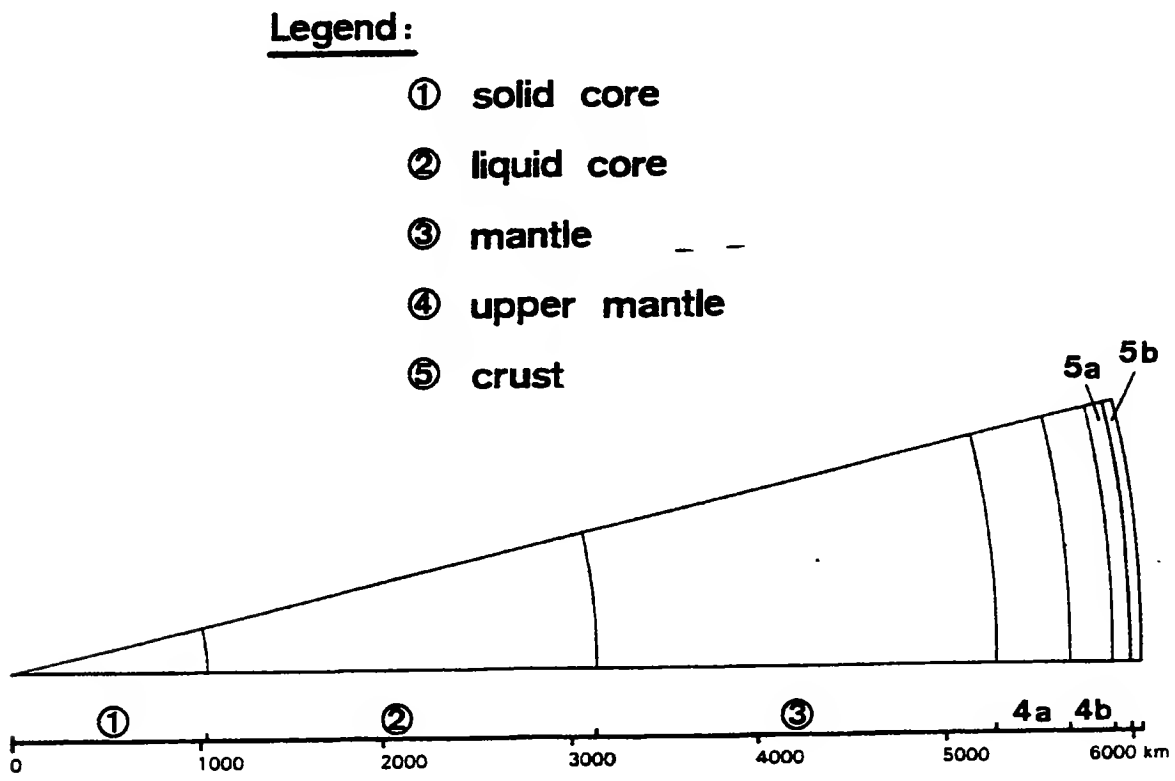
Fig.2 Structure of the Earth



The crust and the upper mantle together form the lithosphere, which forms a solid plate of rock of about 100 km thickness.

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1.2.1 Composition of the Lithosphere

Only approx. 10 - 15 km of the lithosphere has been sufficiently investigated to permit characterization. It is astonishing that the predominant components in that portion of the lithosphere, which are accessible for industrial processing, are, oxygen (50 %) and silicon (25 %, Tables 1 and 3). The remaining 25 % are formed by eight other elements and a mere 0,8 % by the remaining 82 naturally occurring elements, many of which are technically and industrially important (Table 2).

TABLE 1: Composition of the lithosphere

O	46,6 %	⎫
Si	27,7 %	⎪
Al	8,1 %	⎬ 91,0 %
Fe	5,0 %	⎪
Ca	3,6 %	⎭
Na	2,8 %	
K	2,6 %	
Mg	2,1 %	

TABLE 2: Composition of the lithosphere: minor and trace constituents

10^{-1} %	Ti, H, Mn, P
10^{-2} %	F, S, C, Cl, Rb, Sr, Ba, Zr, Cr, V, Zn
10^{-3} %	Ni, Cu, Li, N, Sn, Co, Pb, Th
10^{-4} %	As, B, Mo, Br, W, U
10^{-5} %	Sb, Bi, Ag
10^{-6} %	Hg, A
10^{-7} %	Au, Pt, He

TABLE 3: Chemical composition of the earth, the lithosphere and of Portland cement

		Earth	Lithosphere	Portland cement
Oxygen	O	22,0	46,6	37,0
Silicon	Si	11,0	27,7	9,5
Aluminium	Al	0,6	8,1	3,2
Iron	Fe	50,0	5,0	2,0
Calcium	Ca	1,0	3,6	45,3
Magnesium	Mg	9,0	2,1	1,2
Potassium	K		2,6	0,5
Sodium	Na		2,8"	0,1
Hydrogen	H		0,9	
Nickel	Ni	6,0		
Titanium	Ti		0,6	

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1.2.2 Global Plate Tectonics

Based mainly on results of geophysical research done during the last 30 years (Geomagnetics, Paleomagnetics, Gravity, Seismic), an old hypothesis was confirmed beyond doubt: Continents are constantly changing their positions relative to one another, i.e. they move on the globe. Such movements were put forward in the thirties by A. WEGENER, based on the excellent fit of the coast lines of North/South America on one side, and Europe/Africa on the other side of the Atlantic Ocean. It has been found that the entire globe is actually covered by so-called "plates" that comprise the crust and the solidly crystallized part of the upper mantle (fig. 2a) (the "lithosphere"). Solid parts of this lithosphere move constantly (continental drift), and the so-called "plate boundaries" accommodate this shift.

Three types of plate boundaries are recognized:
distensive, compressive and lateral

- The distensive boundaries are those sites, where new lithosphere is created, nearly exclusively along the so-called mid-ocean ridges. Their relatively mobile liquid masses from the molten part of the upper mantle are welling up to the sea floor and crystallise to form a solid "crust". The rising magma itself segregates because of changes in pressure and temperature, as a result of which different rocks are formed at different depths. The lateral rate of accretion has been measured to be in the order of 2 to 10 cm per year.
- The compressive plate boundaries are essential for destroying parts of the crust (otherwise, the globe would expand). This is effected by thrusting slabs of oceanic crust + mantle down into the liquid part of the mantle.
  These slabs potentially reach a depth of 600 - 700 km, and they are the reason for the deep-seated earthquakes. At surface, they are marked by deep-sea trenches, volcanic arcs and mountain chains (fig. 2b).
- The lateral boundaries are expressed as large wrench faults, normally a system of near vertical faults which can accommodate lateral movements of the plates. These form the links between the other two types of plate boundaries.

Plate boundaries are naturally marked by many geological activities like earthquakes, mountain forming, rapid sedimentation, volcanism etc. The major features of these plates are well established, and a globally compatible pattern of their movements over the last 200 - 300 mio years has been worked out. In detail, however, there are still many problems pending, e.g. the problem of "microplates" or the development of "marginal basins". The descriptions thus far concerned exclusively the kinematics of the plates. The dynamics, or "the driving force", is not completely clear and still heavily contested amongst geologists. The simplest model consists of a "convection cell" mechanism. The liquid part of the mantle cools and sinks in certain areas, whereas in other places the hot molten material rises to the lithosphere (fig. 2 c). Calculations of the thermodynamics involved show that even violent convection is possible, provided the assumption of viscosities, temperatures etc. is realistic. The global plate tectonics model for the first time in the history of the science of geology explains the large geological features in a satisfactory way. Doubtless with the increasing amounts of reliable data available, the model will be modified in the future.

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Fig.2a **Major Plate Boundaries of the Globe**

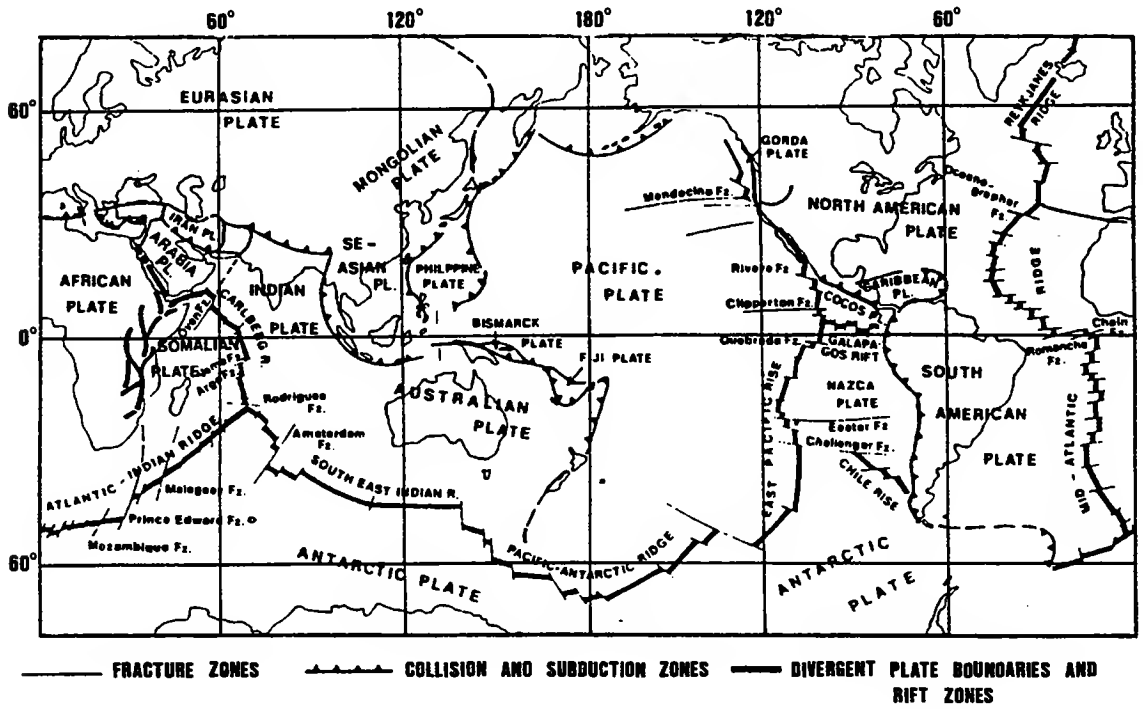


Fig.2b **Compressive Plate Boundary**

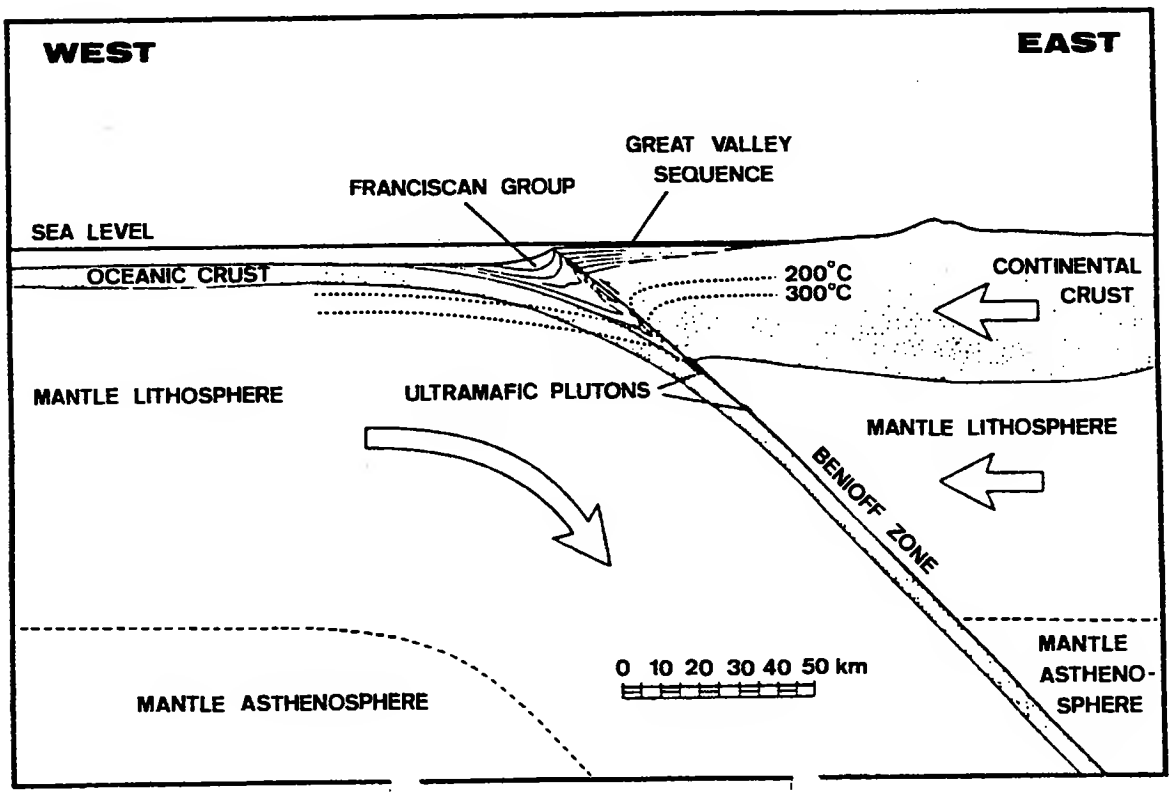
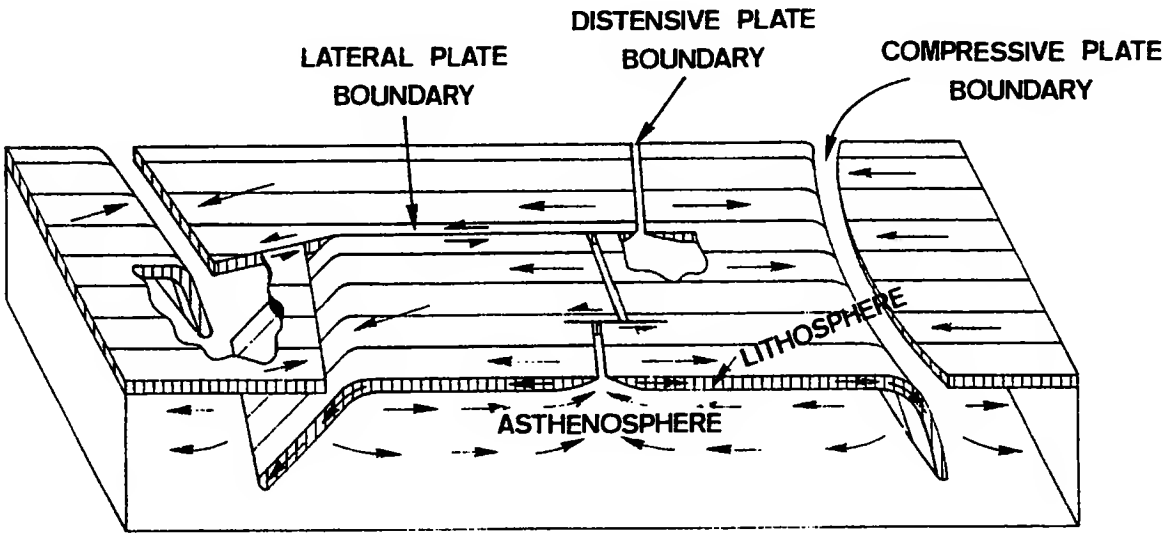


Fig. 2c Global Plate Tectonics Model



1.3 Rock Classification

1.3.1 General remarks

Rocks are classified according to the following criteria:

- mineral content
- genesis, place of formation
- age

Accordingly, three large groups - each of them divided into several subdivisions - can be established:

- igneous rocks
- sedimentary rocks
- metamorphic rocks

1.3.2 Igneous Rocks (Fig. 3)

The magma, which originates from the deeper part of the globe (mostly the upper mantle), rises towards the surface and forms different types of rocks depending on its cooling history and its differentiation process. The magma can change its chemical composition by fractional crystallization and by assimilation of rock fragments of the formations it penetrates. Slow cooling of the magma leads to the development of large crystals, rapid cooling e.g. in a volcano eruption, leads to very small crystals or even amorphous matter in form of volcanic glass.

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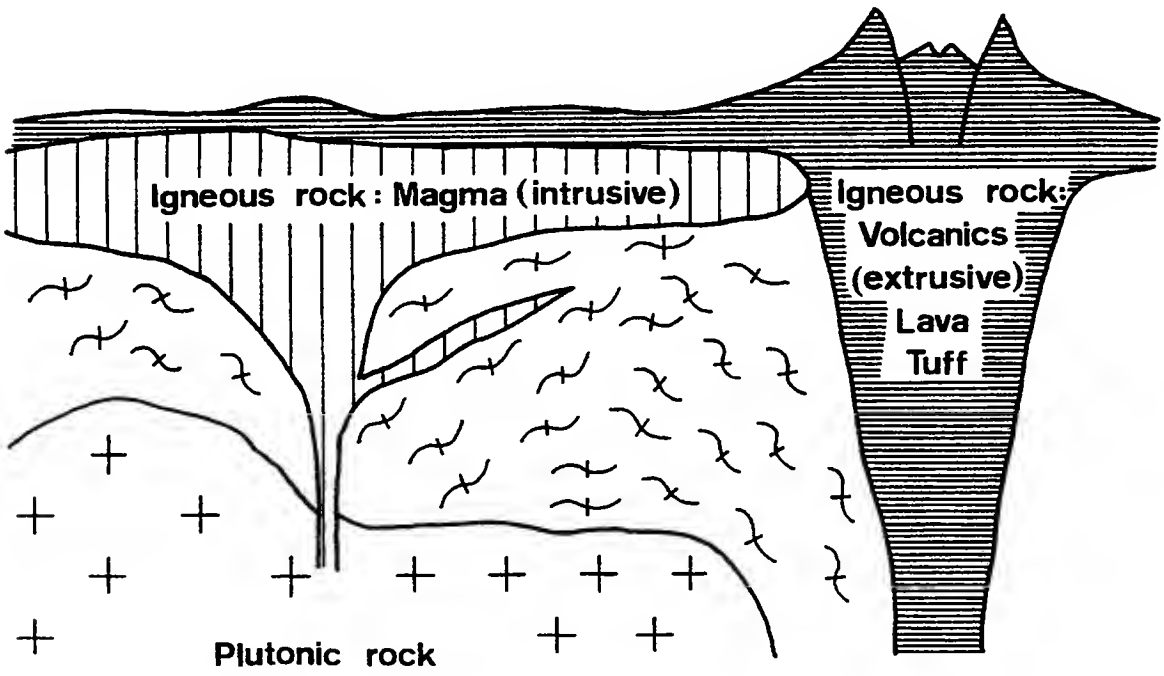
Fig.3 Igneous Rocks



Rocks of this type are often used in Cement and aggregate industry, e.g. as pozolana or crushed rock (granite, basalt), see table 4.

TABLE 4 Igneous rocks in cement and aggregate industries

Volcanics	tuff ash lava perlite agglomerate
Plutonics	granite diorite
Intrusives	andesite, basalt

1.3.3 Sedimentary Rocks

If rocks of any kind are exposed to weathering and erosion caused by temperature changes, atmospheric conditions, etc. on the surface of the earth, disintegration results. Basically, weathering includes two phenomena:

- weathering with undissolved products
- weathering with dissolved products

The formation of sediments includes the following stages

1) Disintegration of the solid rock
2) Transport of dissolved and undissolved products
3) Deposition and precipitation
4) Compaction (Diagenesis)

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According to this, three types of sedimentary rocks are distinguished:

mechanical (clastics)	only mechanical action
chemical	precipitation of dissolved matter
organic	remains of living beings precipitation by organisms

Deposition occurs frequently in the form of a more or less horizontal layering called stratification (strata = layer).

Sedimentary rocks are the most significant resource for the cement and aggregate industry (Table 5). Detailed descriptions and criteria of assessment are given in section 4 "Assessment of Cement Raw Materials".

TABLE 5 Sedimentary rocks in the cement industry

chemical	limestone gypsum anhydrite ironoxihydrate aluminiumoxihydrate rock salt
mechanical	sandstone, sand marl clay, claystone shale
organic	limestone coal oil

1.3.4 Metamorphic Rocks

During rock formation, every mineral and rock is in equilibrium with its environment at a distinct pressure (P) and the temperature (T). Metamorphosis (transformation) of rocks is mostly caused by disturbance to this equilibrium. If one or both of these parameters change, metamorphosis takes place. Metamorphic rocks may, therefore, be formed from igneous as well as from sedimentary rocks, whereby the chemistry of the metamorphic rock may be virtually identical with the composition of the original rock. Metamorphic limestone (marble) is often used as a raw material in cement industries. Other metamorphic rocks are suitable as aggregates, even for special applications (e.g. amphibolite) as aggregates.

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TABLE 6 Metamorphic rocks in the cement industry

Metamorphic rock	Original rock
Amphibolite	Basalt
Marble	Limestone
Phyllite	Shale
Quartzite	Sandstone
Slate *	Shale
Schist *	Shale

* inaccurate terms, to be avoided

1.4 Circuit of Geological Processes

Table 7 demonstrates the interdependence of the rock types as described previously.

The geological processes, which lead to the various types of rocks, can also be demonstrated in terms of a circuit (Fig. 4).

Fig. 4 Circuit of geological processes

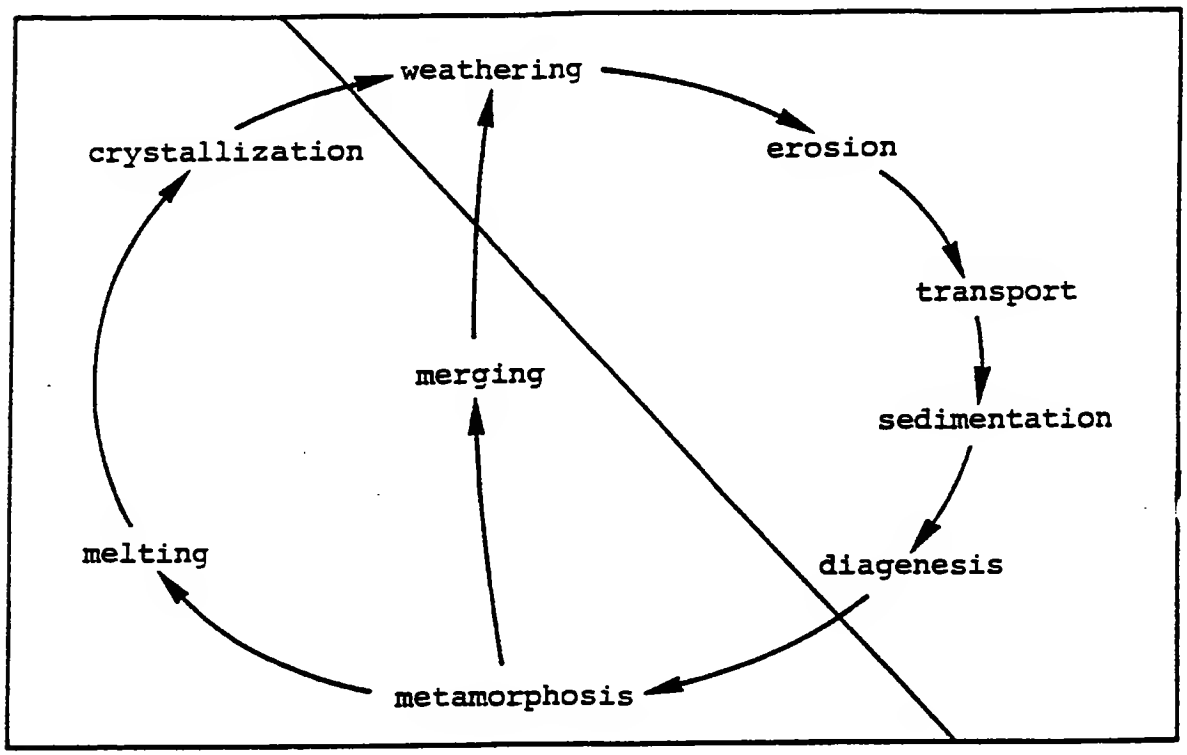


Fig. 4a Circuit of geological processes

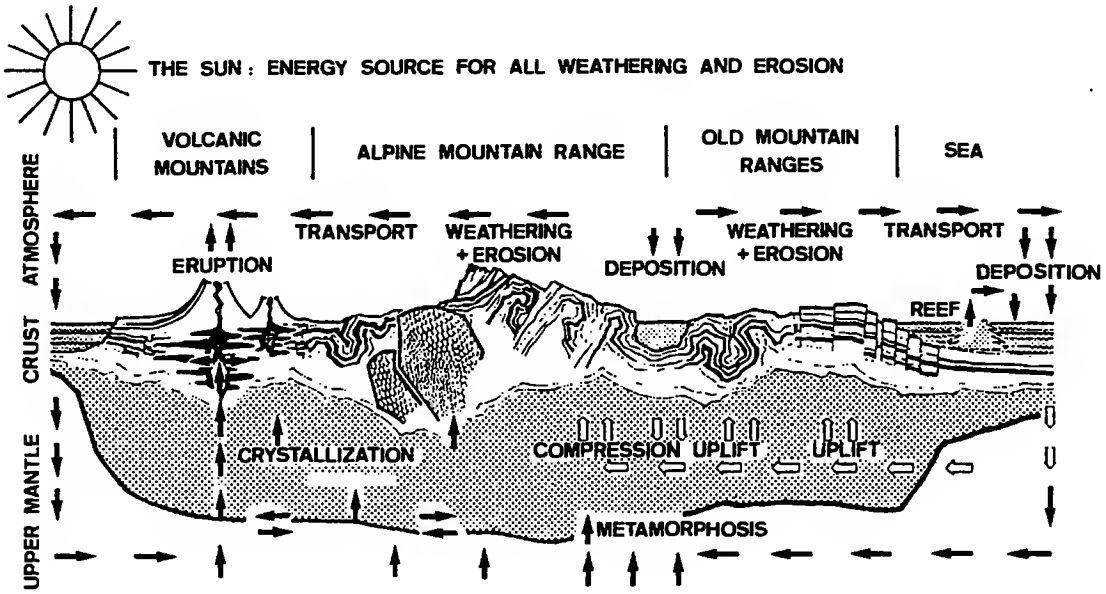
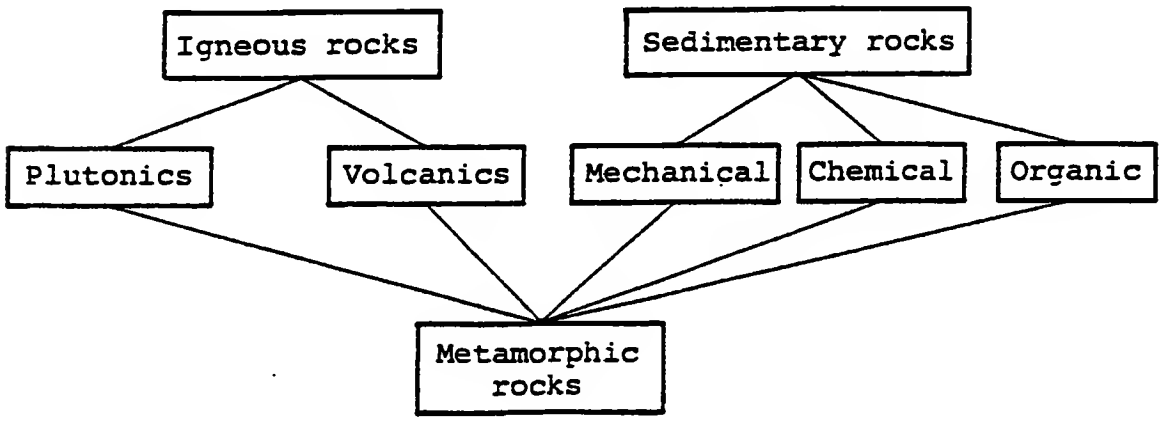


TABLE 7 Interdependence of rock types



1.5 Geological Time-Scale

A rock cannot only be classified according to its mineral content and the place of its formation, but also according to its age. The determination of the age of a rock is normally accomplished by palaeontological methods (investigation of the remainders of fossils) or by physical methods (radiocarbon, radio-active decay).

Table 9 shows the worldwide accepted geological terminology regarding the time-scale. Note that limestone being the most important raw material component for the cement industry, has occurred only (in larger quantities) since the beginning of the Paleozoic age.

Note further, that rocks of the same age but from different locations do not necessarily have identical characteristics.

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TABLE 9 Geological time scale

Era	System and Period	Series and Epoch	Stage and Duration	Absolute Age
Cenozoic	Quarternary	Recent		Last 10'000 y.
		Pleistocene	1,8 mio	(millions of years ago) 1,8
	Tertiary	Pliocene Miocene Oligocene Eocene Palaeocene	3.2 mio 20 mio 10 mio 20 mio 12 mio	1,8 - 67
Mesozoic	Cretaceous	Upper Lower	73 mio	67 - 140
	Jurassic	Upper Middle Lower	70 mio	140 - 210
	Triassic	Upper Middle Lower	40 mio	210 - 250
Palaeozoic	Permian		40 mio	250 - 290
	Carboniferous		70 mio	290 - 360
	Devonian		50 mio	360 - 410
	Silurian		30 mio	410 - 440
	Ordovician		60 mio	440 - 500
	Cambrian		90 mio	500 - 590
Pre-cambrian	Proterozoic Archean (Oldest rocks)			590 - 2500 (3700)

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1.5.1 Chemical Sediments

Chemical sediments are normally formed in sea water where they precipitate under special circumstances due to the ions concentration occurring in sea water. The average ion concentration in sea water are shown in table 11.

TABLE 11

Element	Concentration * in Ocean Water, 10^{-6} g/g	Concentration + in River Water, 10^{-6} g/g	Residence Time, 10^{6} years
Li	0.17		
B	4.6	0.013	15
C	28	11.5	0.1
N	0.5		
F	1.3		
Na	10,500	6.3	71
Mg	1,350	4.1	14
Al	0.01		
Si	3.0	6.1	0.02
P	0.07		
S	885	3.7	10
Cl	19,000	7,8	104
K	380	2,3	7
Ca	400	15	1.15
Mn	0.002	.02	0.004
Fe	0.01	0.7	0.006
Ni	0.002	0.01	0.008
Br	65	0.006-0.019	450-150
Rb	0.12		
Sr	8.0	0.09	3.8
Ag	0.00004	0.001	0.02
I	0.06		
Ba	0.03	0.054	0.02
Pb	0.00003	0.005	0.0003
Th	0.00005		
U	0.003	0.001	0.14

* B. Mason, Principles of Geochemistry (Wiley, 1966, p. 195).
\+ D. A. Livingstone (U.S. Geol. Survey Prof. Paper 440-G, 1963)

These ion concentrations are derived from weathering and dissolution processes onshore, e.g. in mountain ranges.

Fig. 5 summarizes the chemical conditions for common sedimentary minerals :

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Fig. 5 **Relation of Eh and pH to regions of formation of common sedimentary minerals (after Garrels and Christ)**

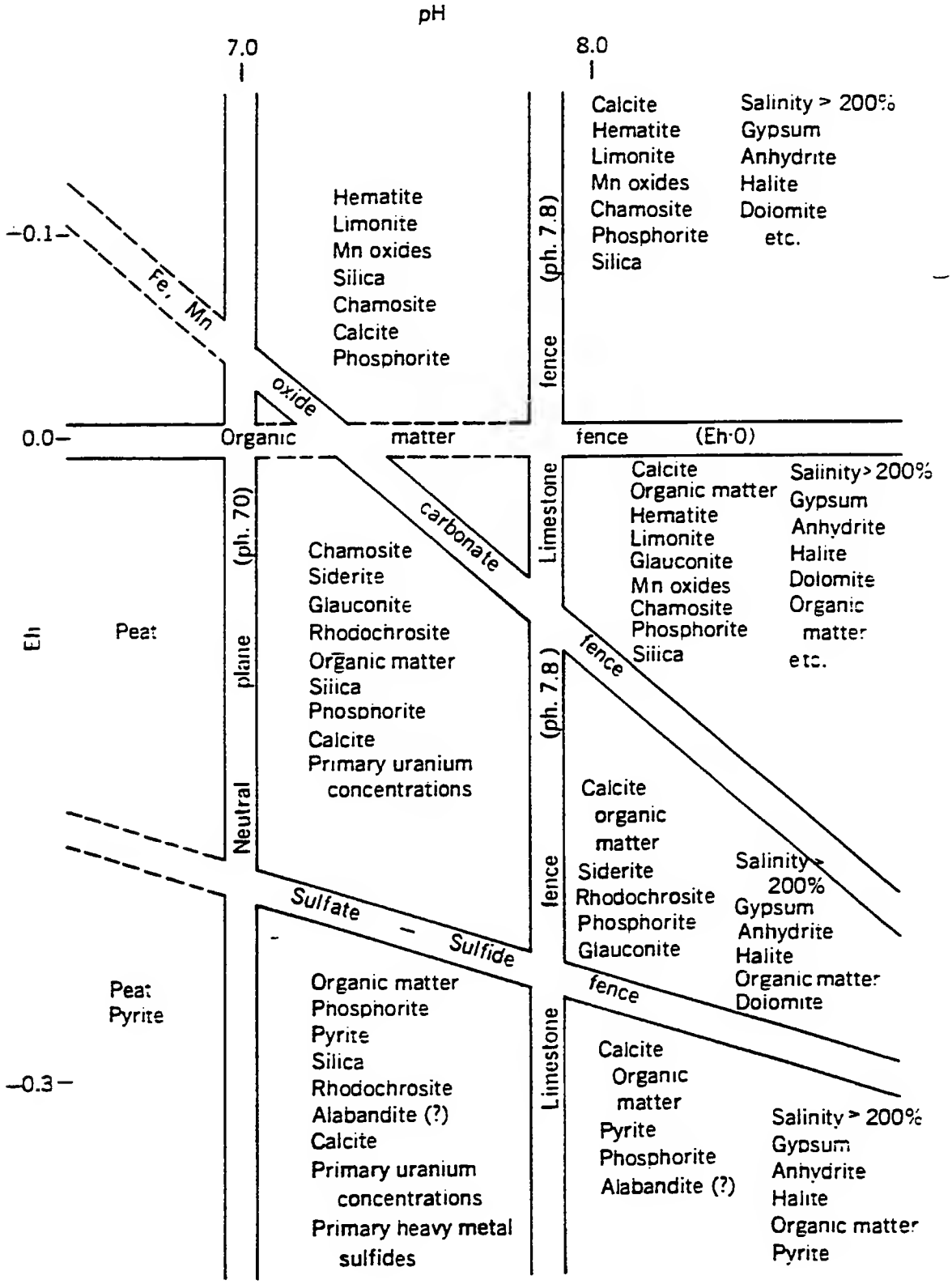

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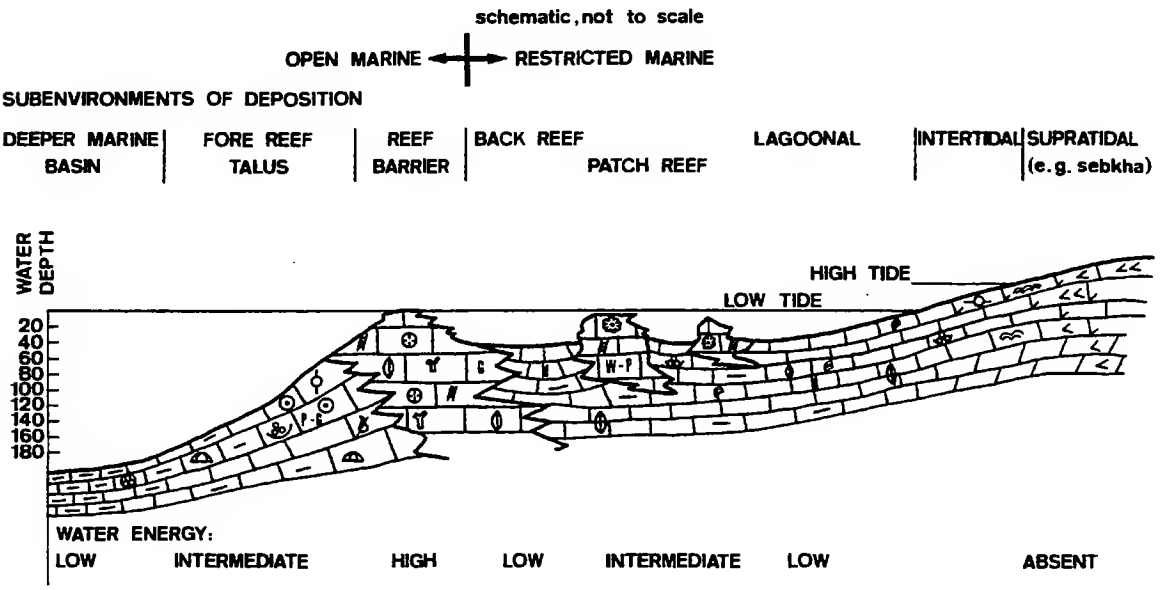
Fig. 5a General model of a shallow marine carbonate platform



1.5.2 Shallow marine carbonate platform

The additional presence of H_2CO_3 in seawater (from the air and supplied by river water) allows the formation of $CaCO_3$ under special circumstances. These are governed by a complicated system of chemical reactions:

$$\underset{\text{Solid}}{CaCO_3} \rightarrow \underset{\text{aq}}{Ca^{2}} + \underset{\text{aq}}{CO_3^{2}\text{-}}$$

$$\underset{\text{aq}}{CO_3^{2}\text{-}} + H_2O \rightarrow \underset{\text{aq}}{HCO_3\text{-}} + \underset{\text{aq}}{OH\text{-}}$$

$$\underset{\text{aq}}{HCO_3\text{-}} + H_2O \rightarrow \underset{\text{aq}}{H_2CO_3} + \underset{\text{aq}}{OH\text{-}}$$

$$\underset{\text{gas}}{CO_2} + H_2O \rightarrow \underset{\text{aq}}{H_2CO_3}$$

$$\underset{\text{aq}}{Ca^{2}} + \underset{\text{aq}}{CO_3^{2\text{-}}} \rightarrow \underset{\text{aq}}{CaCO_3}$$

$$\underset{\text{aq}}{Ca^{2+}} + \underset{\text{aq}}{HCO_3^{2}\text{-}} \rightarrow \underset{\text{aq}}{CaHCO_3^{+}}$$

$$\underset{\text{aq}}{Ca^{2+}} + \underset{\text{aq}}{SO_4^{2}\text{-}} \rightarrow \underset{\text{aq}}{CaSO_4}$$

$$H_2O \rightarrow \underset{\text{aq}}{H^{+}} + \underset{\text{aq}}{OH\text{-}}$$

The knowledge of chemical equilibrium constants and pH of natural sea water made it possible to calculate carbonate solubilities for various marine environments.

It has been found that in shallow, warm sea waters concentration of $CaCO_3$ is close to saturation. This depends largely on the peculiar behaviour of H_2CO_3, which dissolves calcite when present in higher concentrations and only stimulates $CaCO_3$ precipitation when present in small quantities. This is the case in relatively warm water, where plants additionally remove CO_2, calcite is readily removed by dissolution. Below ~ 4000 m, the so-called "compensation depth", no calcite is normally present.

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The preceeding conditions define the environment of deposition for the most important raw materials for cement manufacturing. Located on the so called "shallow marine carbonate platform" or the continental edges. These sites of $CaCO_3$-formation are shallow warm sea water, where organisms like corals, bryozoa, algae etc. form their skeletons (of $CaCO_3$) normally in form of Aragonite.

According to the actualistic theory, it can be assumed that the Bahamas bank, the Arabic Gulf, the Red Sea (sebkha) or the Great Barrier Reef in Australia serve as excellent models for the formation of limestone in shallow warm sea water.

A general model of a shallow marine carbonate platform is shown in fig. 5a. According to water depth, water circulation, water energy (waves), climate, fauna present, a number of subenvironments are discribed. Within these subenvironments a great variety of calcium carbonate particles are produced and deposited. These "soft" carbonate deposits form, after a long time of compaction and diagenesis, a real limestone. Geologists can "read" these different limestones and reconstruct the original environment of deposition of any limestone body.

1.5.3 Limestone diagenesis

All processes in time, which change the physical parameters of limestone, are summarised under this expression. The following points are of interest:

- mechanical compaction due to overburden → reduction of porosity
- recrystallization of aragonite (instable) to calcite, high Mg-calcite (dolomite)
- leaching by rain water or ground water: Early cementation of limestone grains by rapid recrystallization
- chemical compaction by dissolution of calcite and recrystallization in pore spaces
- presence of high Mg-content in pore fluid leads to recrystallization of dolomite → increase of porosity.

Depending on the specific history of a limestone body, diagenesis can produce quite remarkable differences in limestone quality, and often the local changes occur within very short distances. The knowledge of the detailed environment of deposition very often permits a geologic interpretation including an idea as to the spacial distribution of different limestone quantities. Especially important, is the location of dolomitic limestone bodies through reconstruction of the original set of conditions.

Advanced stages of limestone diagenesis are characterized by the absence of porosity, large calcite crystals as developed by progressive recrystallization, pressure solution features like e.g. stylolithes.

The original structure of the limestone is largely destroyed, and it becomes difficult to determine the environment of deposition. Even further diagenesis is called metamophism and leads ultimately to formation of marble.

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1.5.4 Classification of limestone

Due to their economic importance, limestone and carbonates in general have been studied in great detail by many researchers and as a consequence, there are many classifications of limestone and related sediments. One of the most widely used was originally conceived by DUNHAM and it describes the relation of lime components of different sizes and origin to the so-called "cement" or "ground mass". Table 12 gives an impression of the various limestone types and related names. This classification allows an adequate description of limestone in terms of environment of deposition, but it differs greatly from the classification of the clastic sediments.

TABLE 12: Carbonate textural classification (Dunham, slightly modified)

TEXTURE				NAME	ABBREVIATION	
	*) Original components bound together during deposition				informal	computer
Depositional texture reconizable	*)	Grain supported	Locks Mud	Lime-Boundstone	Bdst	B
	"		Contains Mud	Lime-Grainstone	Grst	G
		Mud-suported	Particles (> 20µ) > 10 %	Lime-Packstone	Pkst	P
				Lime-Wackestone	Wkst	W
				Lime-Mudstone	Mdst	M
.....			< 10%	aphanitic	aph	A
.....	Recrystallized texture		Fine	crystalline	xin	X
Depositional texture not recognizable			Coarse	sucrosic	suc	S

1.5.5 Clastic sediments

Clastic or "mechanical" sediments are formed from solid particles which are transported from areas of erosion to areas of deposition. The transport agent can be water, wind or ice, the most common being water. Coarse particles require high water energy for transport, normally linked to a steep topographic gradient, to be removed from their place of origin. During transport they are further fragmented, chemically altered and mechanically shaped. The end product, the clastic sediment, therefore depends on a variety of parameters, but mainly on:

- energy of transporting agent, defining in turn
  - grain size distribution
  - shape of grains or "clasts"
- time of transport, influencing chemical composition of particles, sorting of grains according to size and quality
- nature of originally eroded rocks, which defined to a large extent what type of particles are formed. In a rock with little chemical resistance very small particles are formed from partly altered minerals, the clay minerals.

The complex cycle of erosion - transport - deposition results in a separation of the particles and this is expressed with the term "maturity" of a clastic sediment. Long transport distance

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and time leads to the deposition of the most resistant particles in uniform grain size distribution and well rounded like a pure quartz sandstone. Short transport distance and time produces e.g. a breccia or conglomerate with large components of heterogeneous mineralogical composition.

Fig. 5b Clasical environment of "mechanical" or clastic deposition

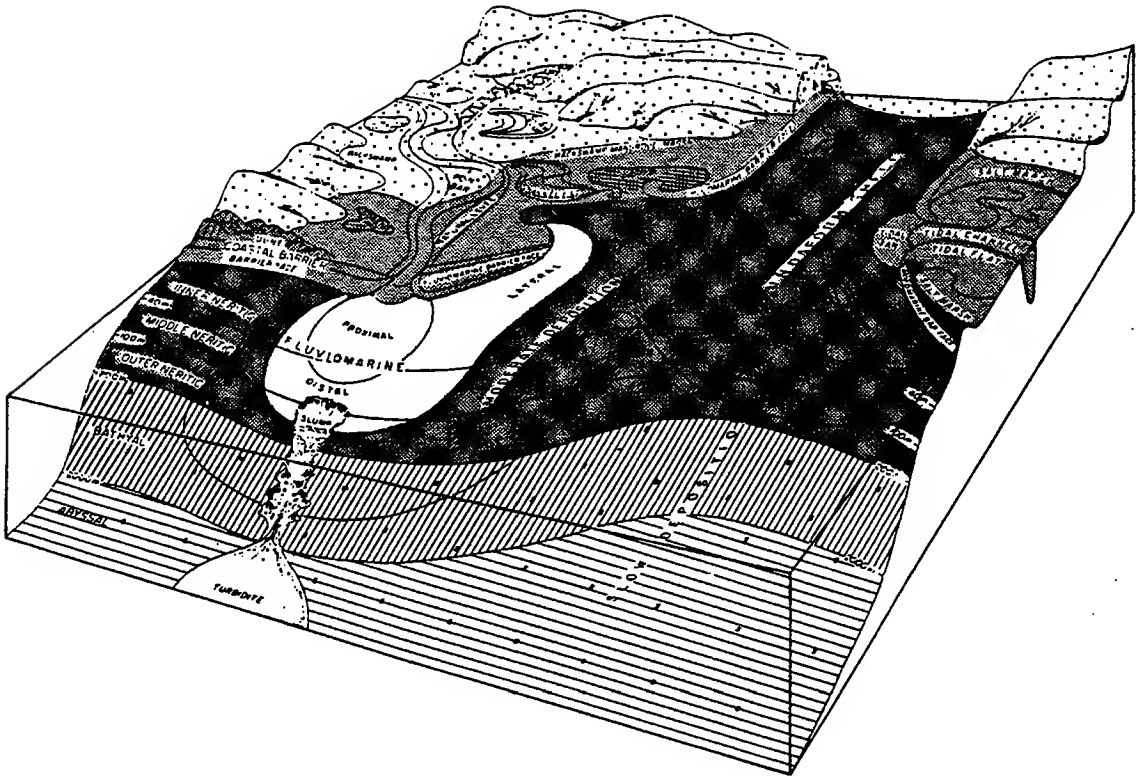

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Fig. 5b Legend

LEGEND:

CONTINENTAL ENVIRONMENTS

FLUVIATILE ENVIRONMENT

BRAIDED RIVER

MEANDERING RIVER

ALLUVIAL FANS

LACUSTRINE ENVIRONMENT

DESERT ENVIRONMENT

DUNES

WADI DEPOSITS

GLACIAL ENVIRONMENT

COASTAL OR „TRANSITIONAL MARINE" ENVIRONMENTS

DELTAIC ENVIRONMENT

SHALLOW SHELF ENVIRONMENT

INTERTIDAL ENVIRONMENT

COASTAL BARRIER ENVIRONMENT

EVAPORITIC ENVIRONMENT

HOLOMARINE ENVIRONMENTS

NERITIC ENVIRONMENT

BATHYAL ENVIRONMENT

ABYSSAL ENVIRONMENT

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1.5.6 Clastic environments of deposition

Inevitably, the clastic particles are formed on the continents, transported and ultimately end up in the oceans. A summary of the most important environments of clastic deposition is given in Fig. 5b.

The dominant factors in forming the specific sedimentary features of clastic environments is documented further in Figs. 5c, 5d, 5e, 5f and 5g. Three important environments, delta, coastal barrier and deep sea fans, each of which is characterised by very distinct sedimentary structures, fossils and trace fossils.

Delta Environment

Fig.5c Schematic Diagram of Delta

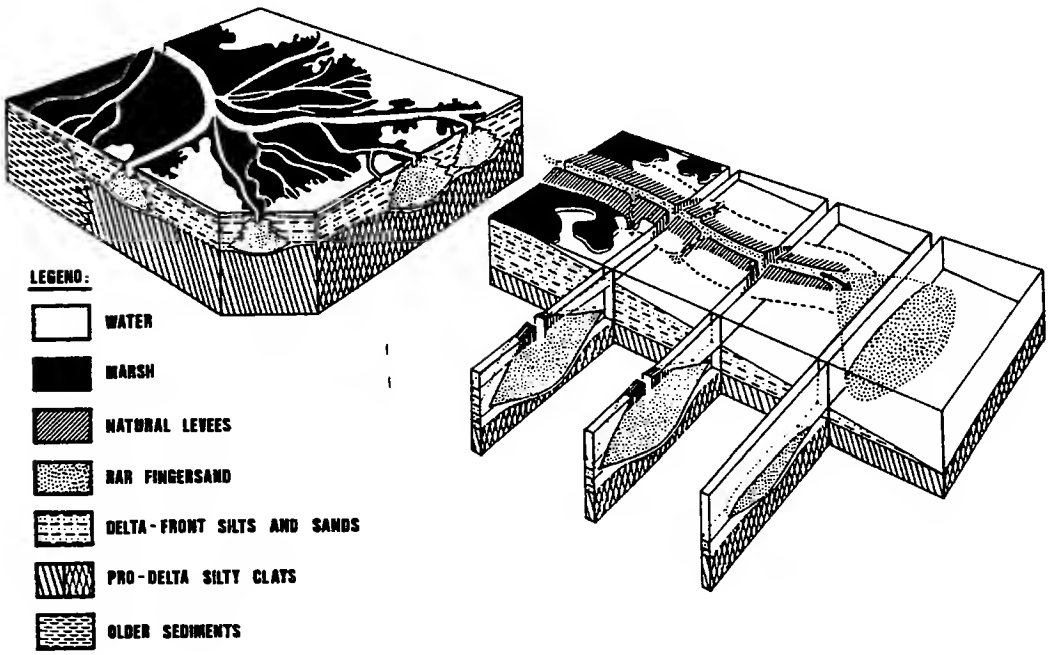
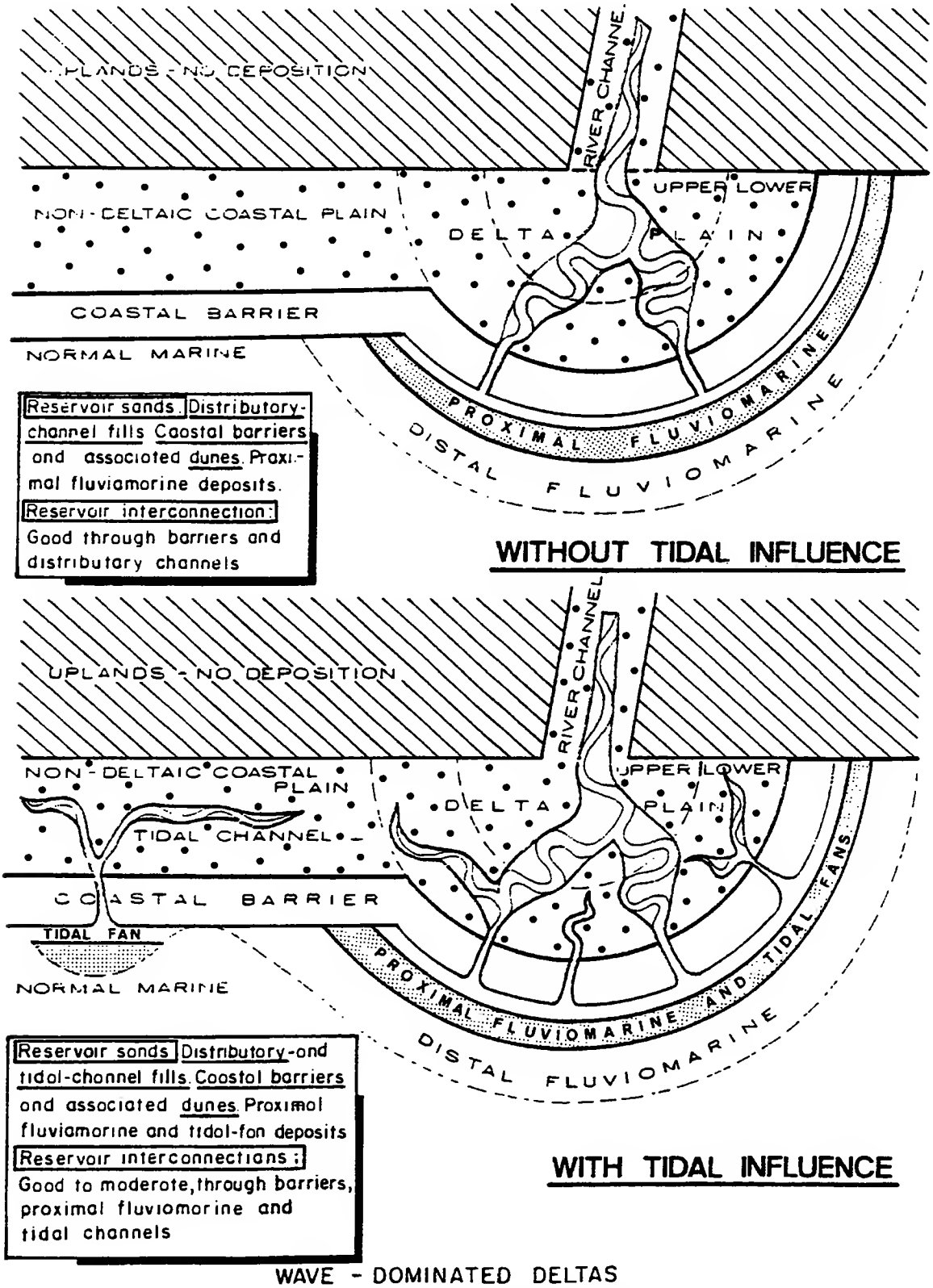


Fig.5d Wave Dominated Deltas



WAVE - DOMINATED DELTAS

CHARATERISTIC : Slow seaward growth, leaving waves time to rework fluviamarine deposits into coastal barriers

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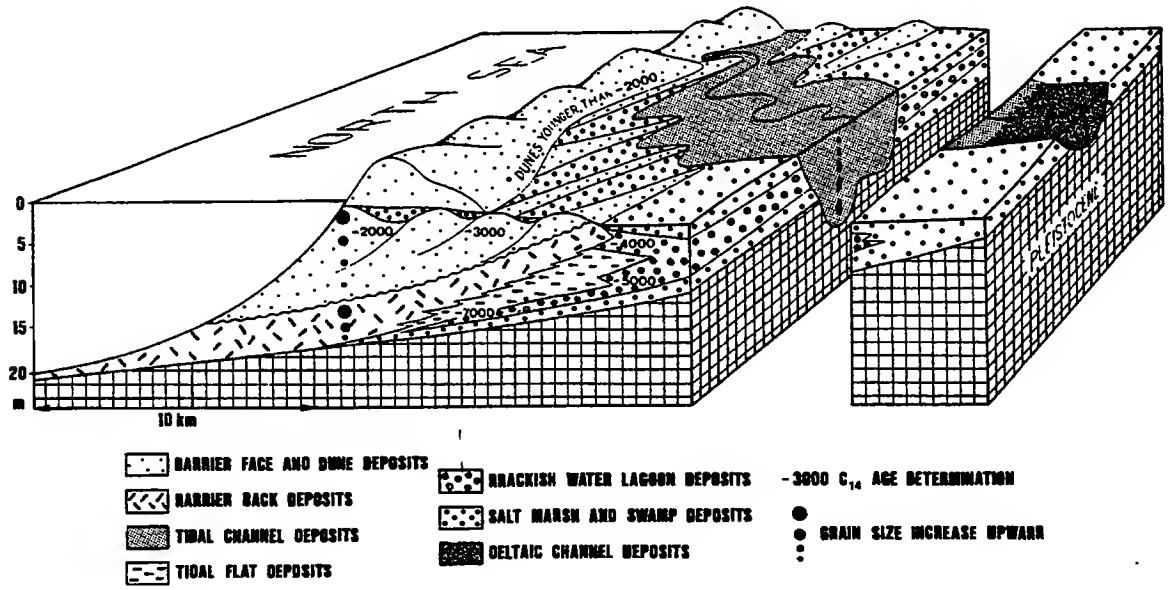
Sub environment	Type of particles deposited	Sedimentary features bedding	Contents of fauna / flora
natural levee	silty clay	laminated root disrupted current ripples	plants
marsh, swamps	silty clay, peat	no bedding Burrows	plants
channel fill	sand	cross bedding clay balls	(plants)
distributory channel	sand	cross bedding current ripples clay balls	nearly absent
mouth bar	sand, silt	trough cross bedding, wave + current ripples gas heave structures	nearly absent
distal bar	silt, clay	cross bedding ripples burrows	dense benthonic fauna
pro delta (delta front slope)	clay, silty clay	lamination ripples graded bedding bioturbation	benthonic fauna

Coastal barrier environment

Fig. 5e Holocene Coastal Plain Deposits



The dominant energy source is waves, the tide and long shore currents. Particles are extensively reworked and rearranged, refer Fig. 5e. Note the occurrence of dunes which depend also on wind transport. According to climate, tide differences, material supply and vegetation, many variations of the presented model may occur.

Deep sea fan (bathyal to abyssal environment)

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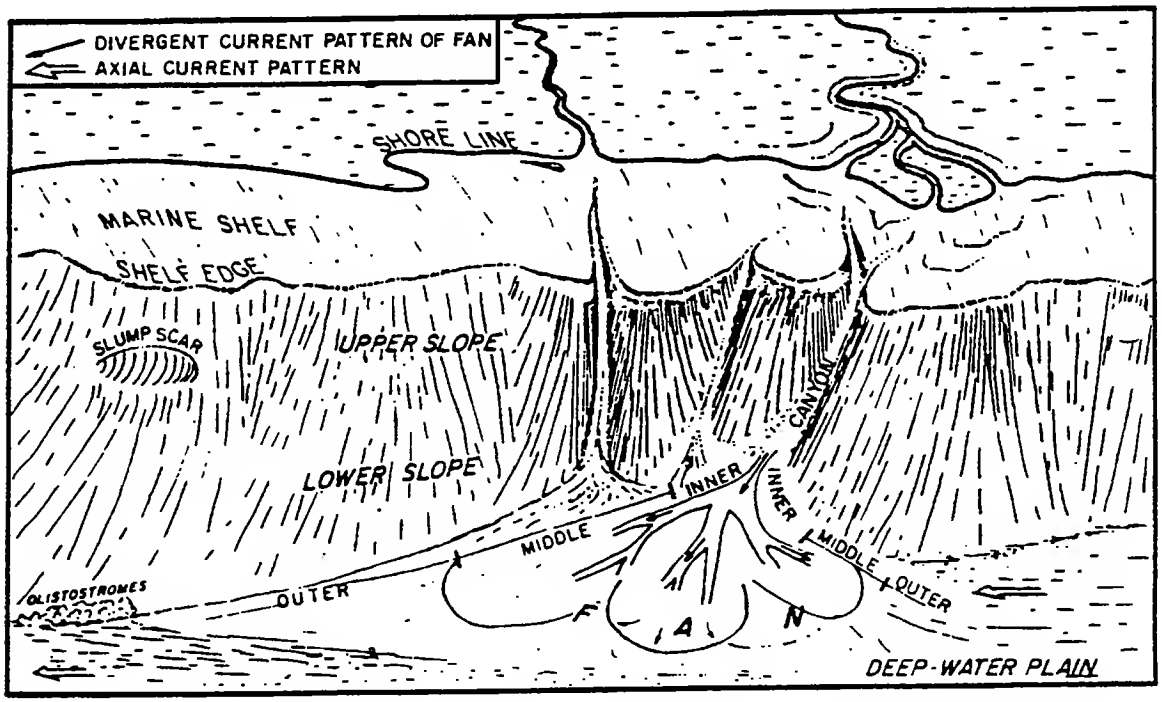
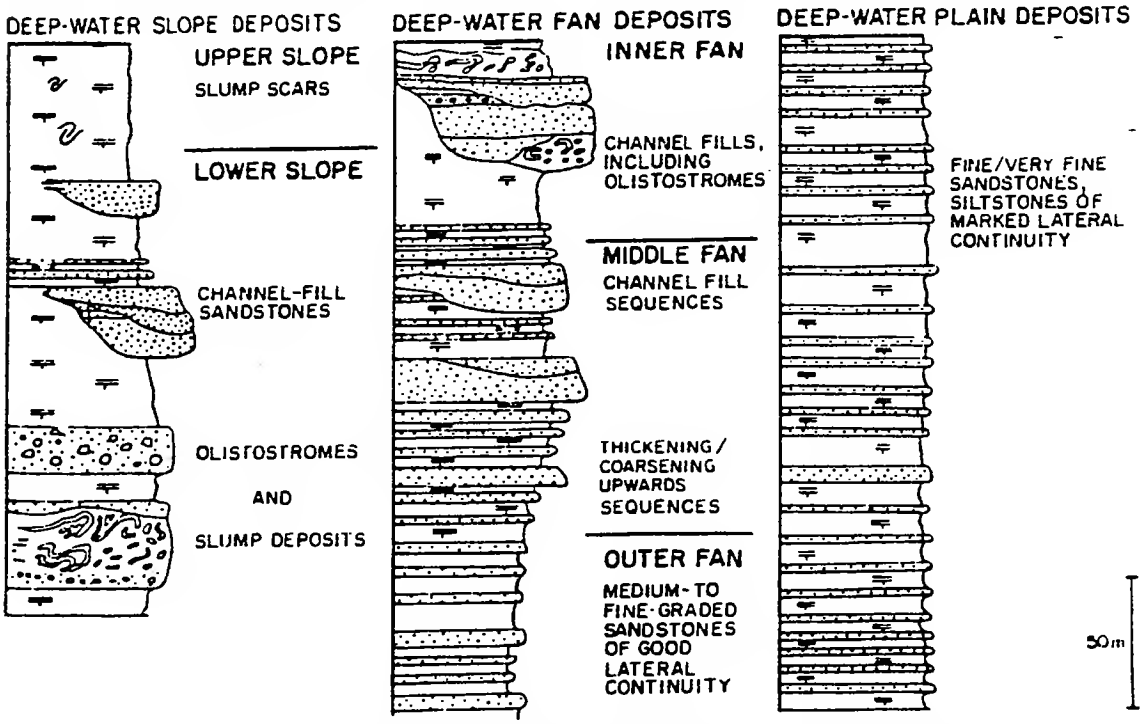
Fig. 5f Subaqueous fan and lithological columns of various fan deposits





A MODEL OF DEEP-WATER SLOPE AND FAN DEPOSITION

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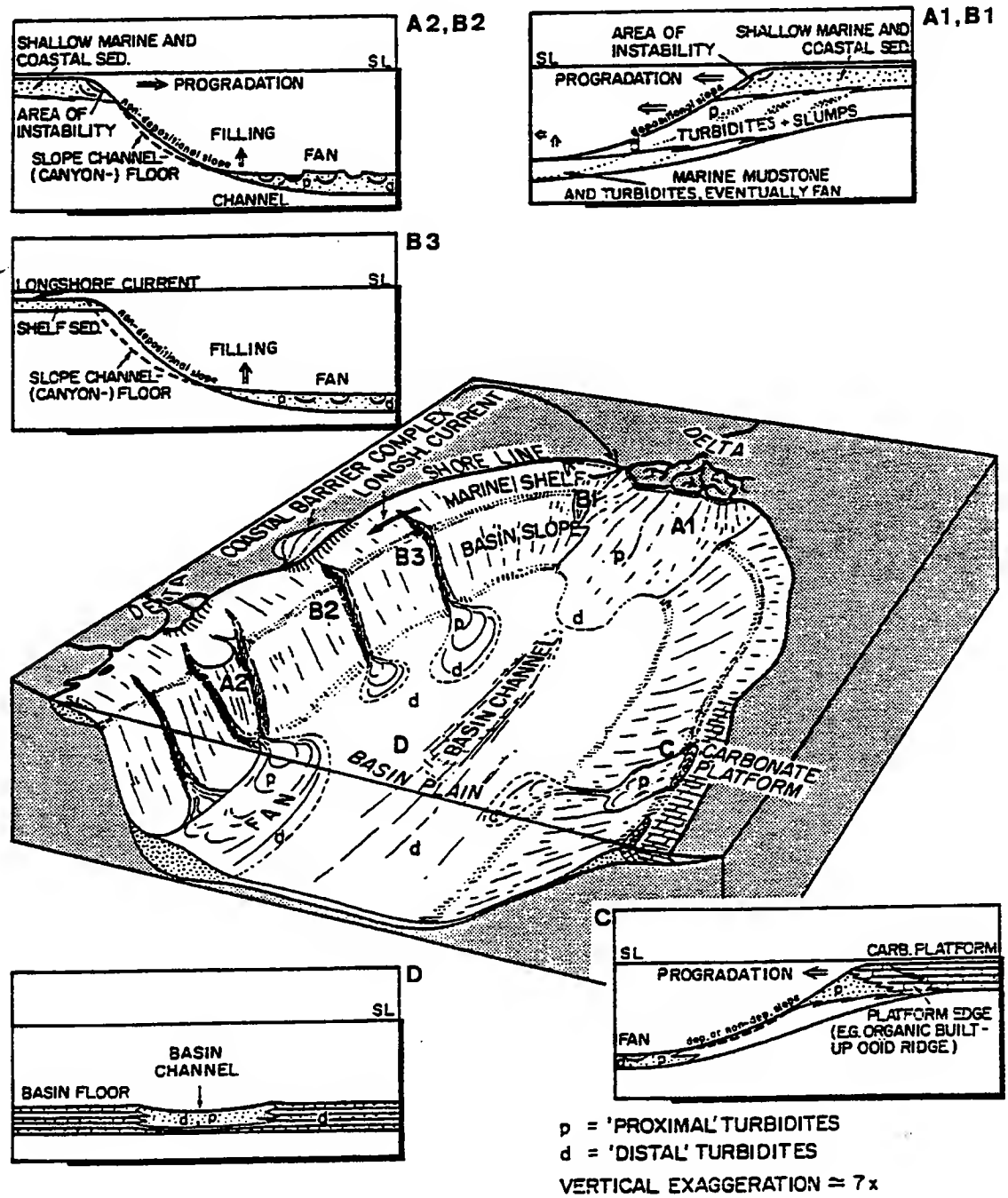
Fig. 5g Integrated model of deep-water sand distribution



Deep sea fans are formed by means of a special transport mechanism called "turbidity current". Unconsolidated particles which are deposited on the shelf edge may become unstable and start to flow downwards in form of an high density aquatic suspension. These carry large masses of sediments down the continental slope and spread in the deep marine basins over large areas (thousands of km^2). During settling of the particles, typical "graded bedding" develops.In addition to which various other sedimentary features such as flue casts, parallel lamination, current ripple lamination, convolute bedding and pelitic (clay) layers.

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Turbidity currents, which are triggered by earthquakes, occur frequently - about every 10'000 years under favorable circumstances. These form a single bed, nearly instantaneously. During the intervening time period, between consecutive turbidity currents, the normal "pelagic" deposition of clay and lime particles continues. This covers the top of the turbidite layer with a relatively thin layer which contains normally datable "nannoplankton" (remnants of calcareous algae floating at the surface of the open ocean). Turbidite sequences are extremely well bedded and exhibit normal characteristic alternations of sand and clay or limestone and clay, depending on the origin of the particles. They can represent a perfect natural raw mix for cement manufacturing.

1.5.7 Classification of clastic sediments

Determining factors for a classification are not only genetic features, as outlined above, like

- specific environment of deposition
- sedimentary features
- fauna/flora content

but also parameters which can be measured objectively and are of technical relevance like

- grain size distribution
- chemical composition
- mineralogical composition
- degree of consolidation
- porosity, permeability

Tables 13 and 14 give two examples of the classification on the basis of grain size distribution, considering well sorted clastic sediments.

TABLE 13: Classification of mechanical sediments

Grain size (mm)	Constituents	Degree of Consolidation	
		loose	solid
> 200	boulders, blocks		
200-20 20-2	coarse gravel) fine gravel) psephite	rubble	breccia conglomerate
2-0.2 0.2- 0.02	coarse sand fine sand) psammite	sand	sandstone
0.02-0.002 < 0.002	clay silt) colloidal clay) pelite	clay	claystone

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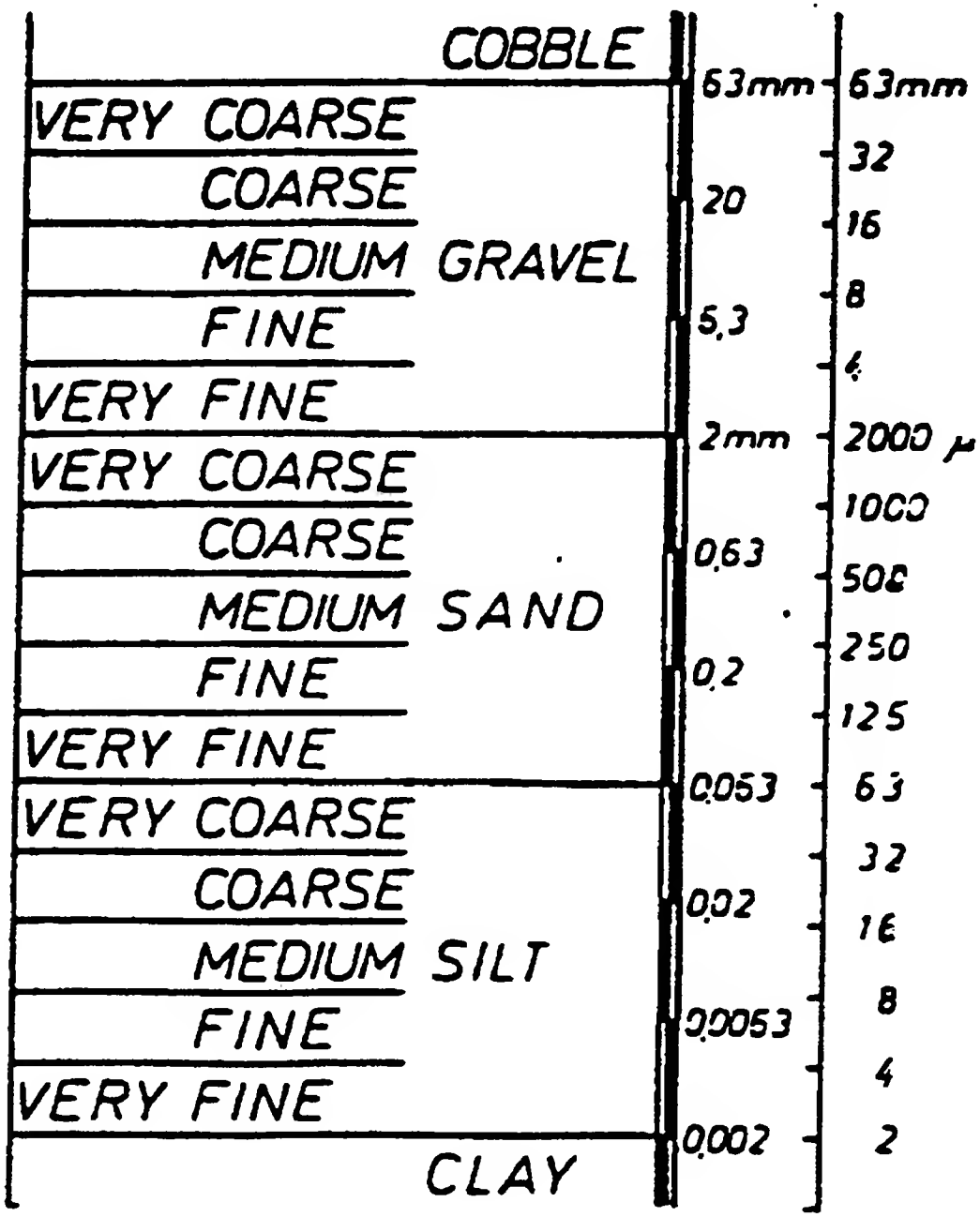
TABLE 14: Definition of grain size



1.5.8 Mixtures of Clastic and Chemical Particles

Since most of the natural sedimentary rocks are mixes of various types of rock, classification thereof can also be accomplished with a three-component diagram (Fig. 6).

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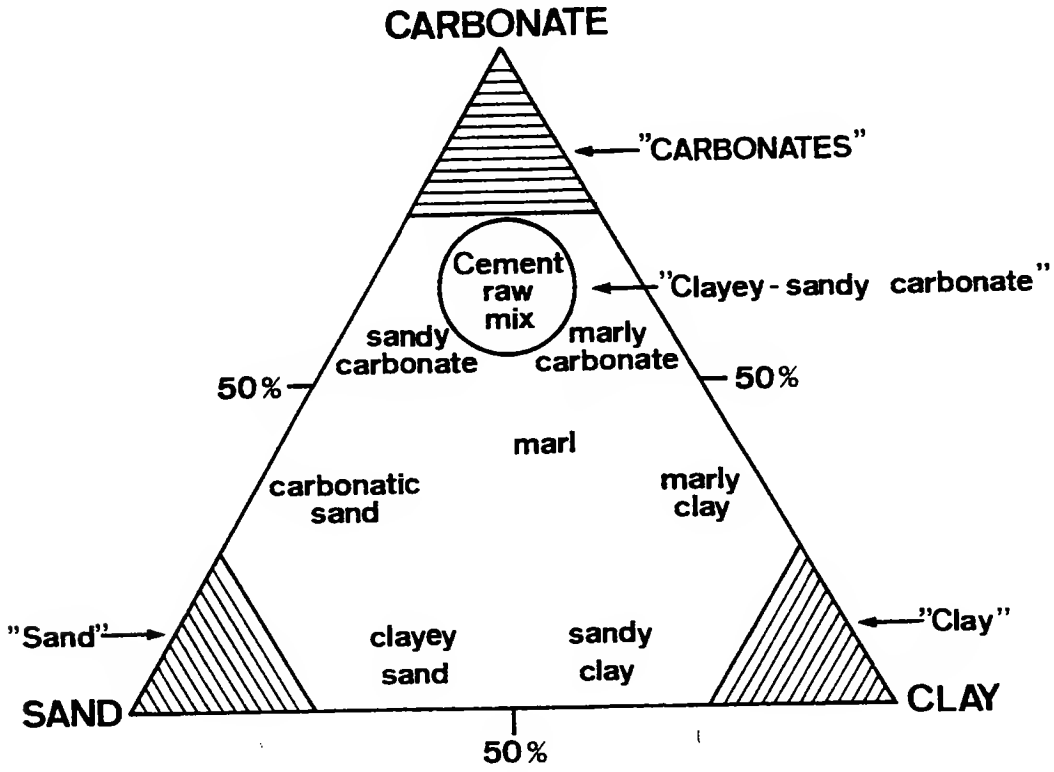
Fig. 6 **Display of the definitions of rock types in the sand-clay-carbonate-system**



Sandy-clayey-marly rocks are also designated as "siliceous" rocks; sandy-clayey materials as "argillaceous" rocks. In addition, the term carbonatic can be replaced by "calcareous".

As a further example (Table 15), the carbonatic (or calcareous) rocks are selected to illustrate the principle of classification based on chemical characteristics.

TABLE 15 **Classification of calcareous and clayey materials (according to HMC's practice)**

$CaCO_3$ %	Clay minerals %	Nomenclature
100 - 95	0 - 5	high-grade limestone
95 - 85	5 - 15	limestone
75 - 85	15 - 25	marly limestone
65 - 75	25 - 35	calcareous marl
35 - 65	35 - 65	marl
25 - 35	65 - 75	clayey marl
15 - 25	75 - 85	marly clay
5 - 15	85 - 95	clay
0 - 5	95 - 100	high-grade clay

The same principles may be applied to the group of calcareous-siliceous (sandy) materials and the group of siliceous and clayey rocks.

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1.6 Structural geology

Structural geology deals with the deformation of rocks. This deformation is a result of tectonic (mechanical) forces, which occur in the solid lithosphere due to movements of the continental plates. In detail, a stressfield applied on the rock formations results in deformation in form of rupture in the case of brittle deformation or flow in case of plastic deformation. These stressfield forces are very large and act over long time periods. In the case of brittle deformation, the theoretical approach by means of physical - mathematical methods is not too complex. However, as soon as plastic deformation over a long period of time is also considered the analytical approach becomes very complicated. The study of the phenomena of rock deformation is known as "tectonics." A science, which incorporates higher mathematics.and physics.

In our industry the structural behaviour and characteristics of the raw material deposits is of high interest, since it has an important impact on the distribution of rock qualities within the deposits. For the description of deformation, one requires a reference system, which shows the effect of deformation with regard to an original, undeformed situation. One very common system is stratification, a frequently observed sedimentary feature. In it's original state stratification is practically horizontal, due to the gravity forces. If strata are found in inclined position in the field, a deformation phase is normally responsible.

By measuring the inclined position of bedding planes, fault planes, joint surfaces, inclination and orientation of folds etc. the degree and type of deformation can be determined.

The corresponding measurements are called strike and dip. For example, in order to measure the orientation and inclination (strike and dip) of a bedding plane (refer figure 8). The intersection of the inclined bedding plane and an imaginary horizontal plane (water line) represents a straight line in space. This line or 'strike direction' has an astronomical orientation, which is measured, with a compass, in degrees clockwise from the (Azimuth).

The inclination, or 'dip', of the bedding plane is measured along a line perpendicular to the strike direction. The dip angle is measured with an inlinometer in degrees with respect to the horizontal plane. There are normally two solutions for this dip angle depending on the orientation 'strike' of the bedding plane ie. dipping in a N or S direction.

A complete measurement of strike and dip of a bedding plane at the locality (or position) A21 would be "A21 - 124 / 25° N". There are various conventions in different countries for recording these measurements. In the USA the dip direction is very often given in form of an azimuth reading such that the above reading would be "A21 - 034 / 24°."

By means of these measurements it is often possible to define the type and intensity of rock deformation and therefore work out the "tectonic style" of a given raw material deposit. In the case of geologic faults, three main fault types are defined, based on the orientation of the three main stress axis of the applied stressfield ($Ó_1$, $Ó_2$ and $Ó_3$, whereby $Ó_1 > Ó_2 > Ó_3$) (Fig. 9):

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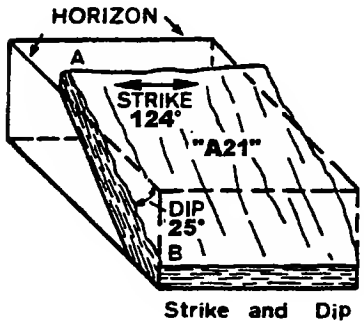
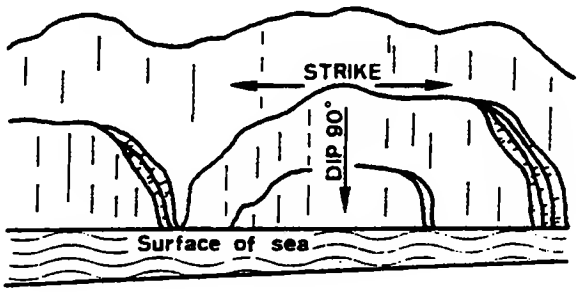
Fig. 8 Strike and Dip Measurements



Strike and Dip



- $\acute{O}_1$ vertical, $\acute{O}_2$ and $\acute{O}_3$ horizontal: the gravity becomes the largest force and therefore blocks of rock glide downward along typically inclined faults of 60°. These faults are called normal faults. They are typical for extensional tectonics.
- $\acute{O}_1$ and $\acute{O}_2$ horizontal, $\acute{O}_3$ vertical: This situation represents a compressional regime, where the rock body is shortened by overthrusts and or folds. Typically, the fault planes are inclined 30° with respect to the horizontal plane.
- $\acute{O}_1$ and $\acute{O}_3$ horizontal, $\acute{O}_2$ vertical: The result of this configuration is the so called strike-slip faults (or wrench faults) which are generally vertical and indicate a strike which deviates 30° from the $\acute{O}_1$ direction.

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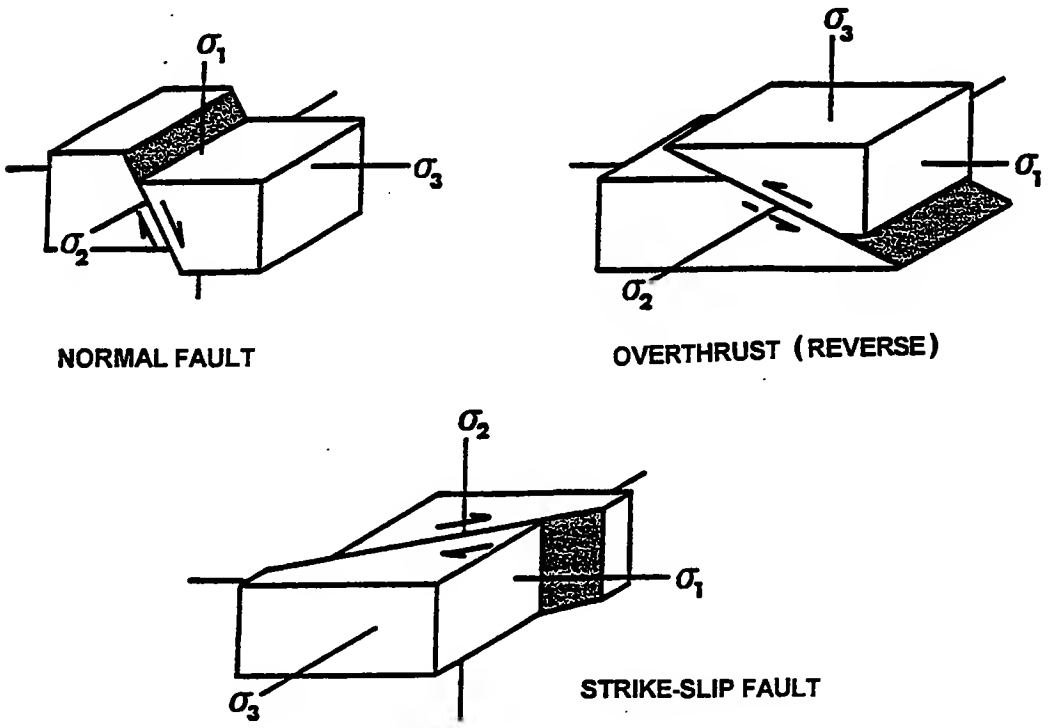
Fig. 9 Classic Geologic Fault Types



σ_1 main stress σ_2 intermediate stress

σ_3 lowest stress

In nature, these clear-cut cases are seldom found. Fault zones are mostly accompanied by secondary faults and by a pattern of "joints" (small fractures).

Combination of fault types and repetitions thereof are frequently found, e.g. series of normal faults, series of overthrusts and folds etc.

In case of folding, the variability of structures produced is also bewildering (see fig 10): depending on the geology of the rocks involved and the stress forces applied, a wide range of different of folds types develop.

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Fig. 10 Classic Fold Types

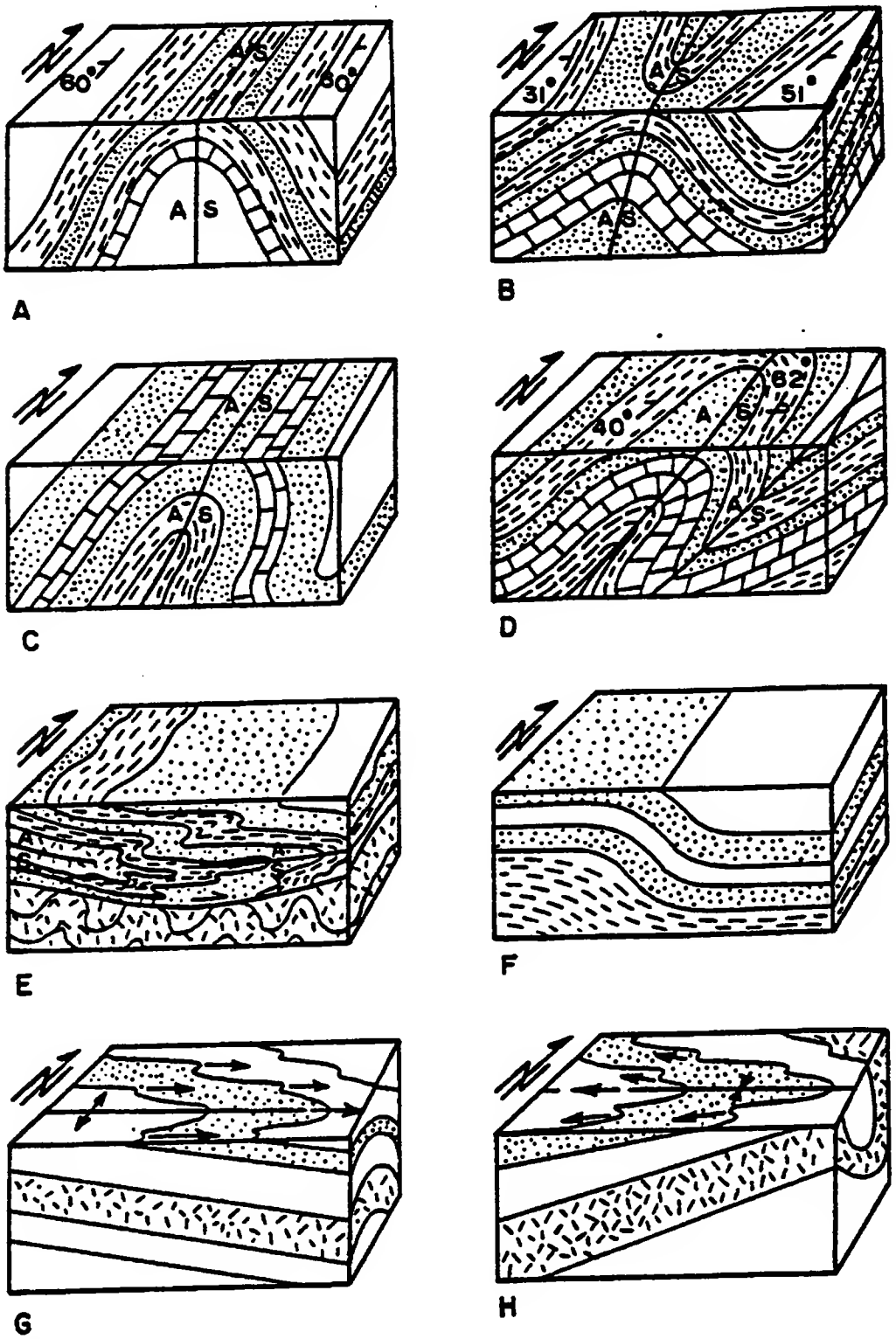

Fig. 10 (A) Symmetrical, open, nonplunging anticline, showing the position of the axial surface (AS). (B) Asymmetrical south-plunging folds showing the position of the axial surface (AS) in the anticline. In this case the axial surface to the west. (C) Isoclinal, nonplunging, closed, inclined fold. (D) Overturned north-plunging fold. Overturning is to the east. The axial surfaces (AS) dip west-ward. (E) Recumbent, nonplunging fold. The axial surface is essentially horizontal. Sometimes referred to as "nappe structures," although the

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underlying limb is not always present (e.g., Austro-Alpine nappes). (F) Monocline. (G) Cylindrical, east-plunging anticline. (H) Conical, west-plunging syncline. The plunge symbols diverge from a vertex located near the east edge of the diagram.

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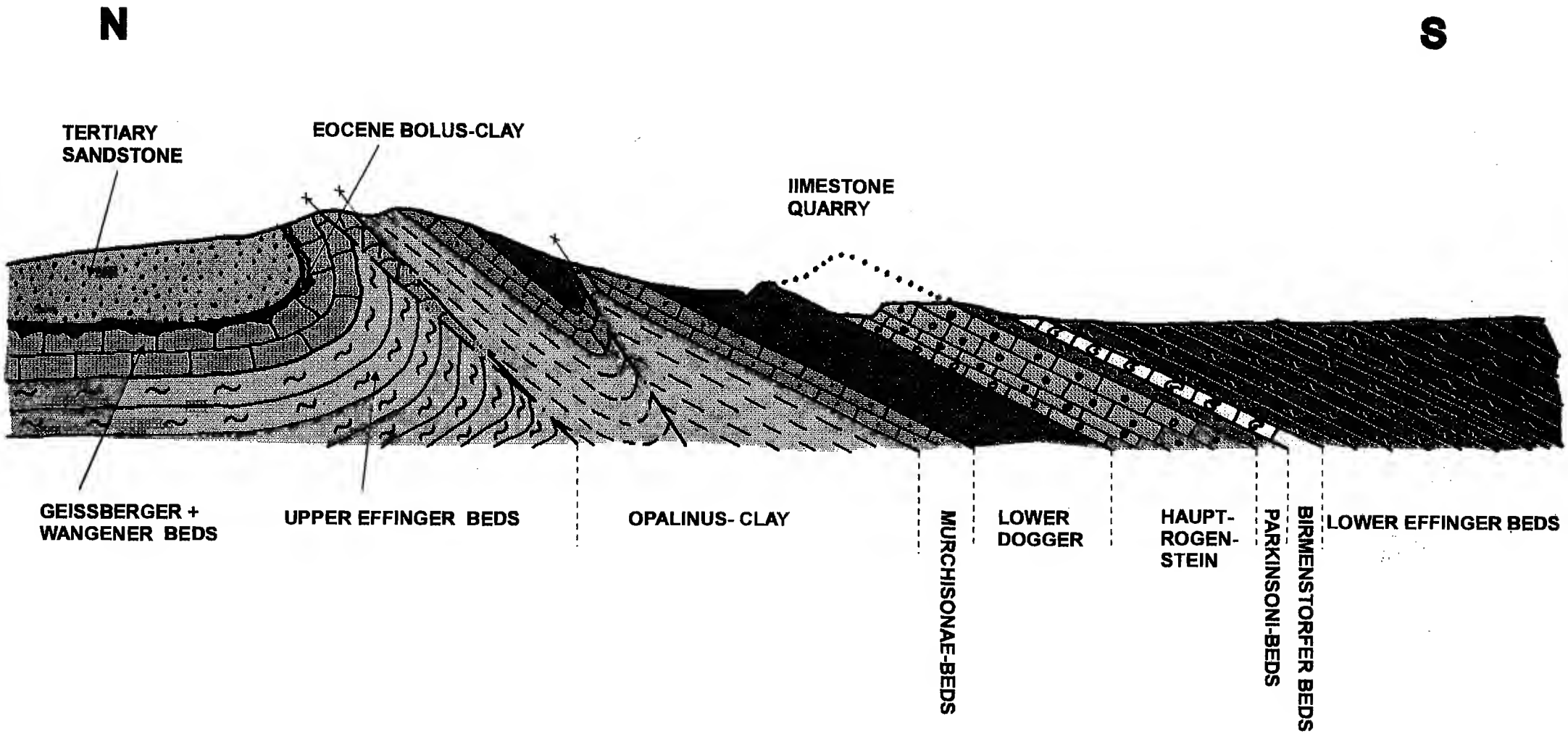
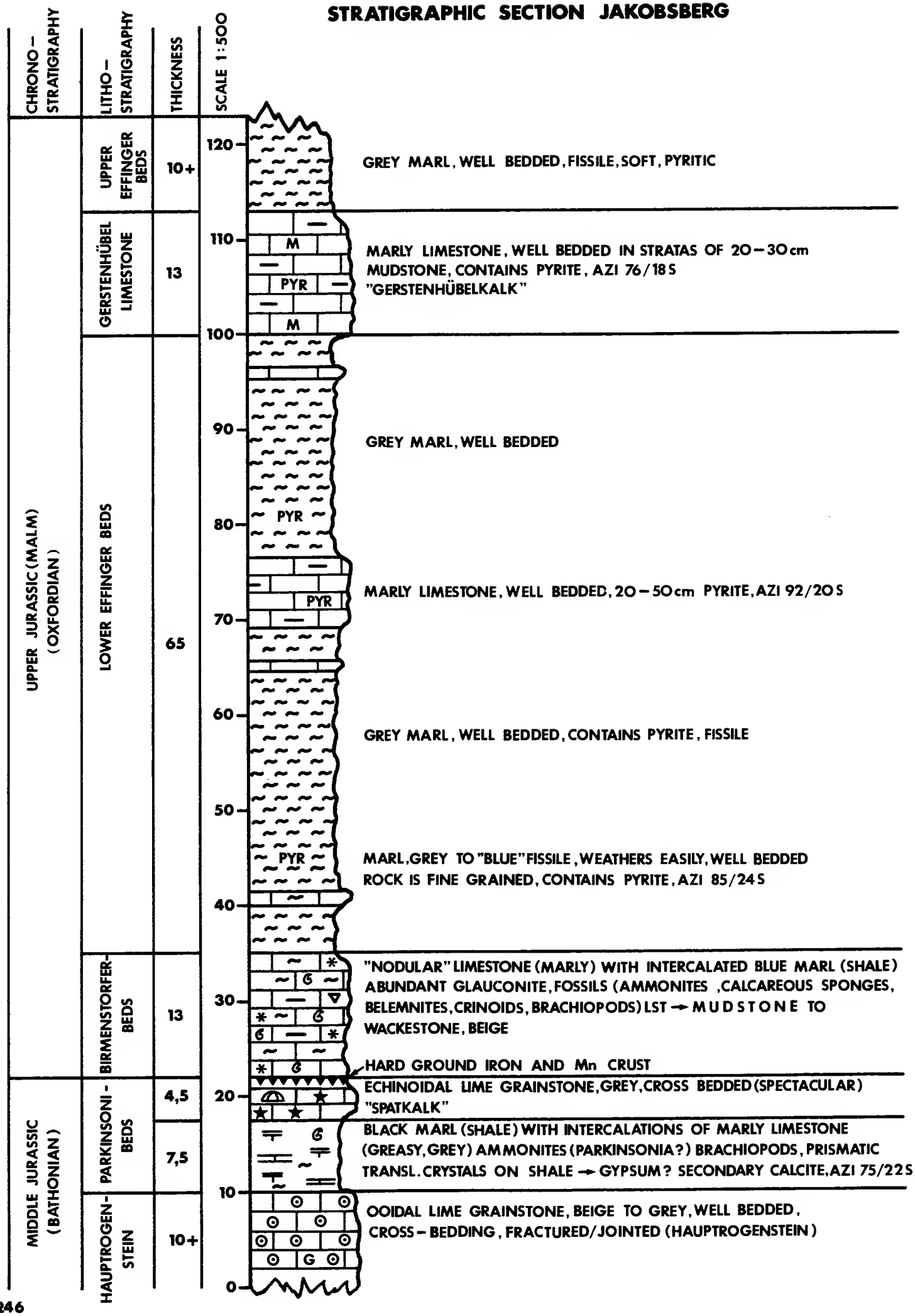
GEOLOGICAL CROSS-SECTION
GISLIFLUH ANTICLINE

SCALE 1:5000, HOR.+VERT.



STRATIGRAPHIC SECTION JAKOBSBERG



Chapter 3

Desk Study for Raw Materials Deposits

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Desk study for deposits of raw materials

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2.6 Results of the desk study 54

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- Holderbank approach
- Desk study

1. INTRODUCTION

In the following chapters, we will elaborate on the procedures developed especially for the exploration of raw materials for cement and aggregates. The proposed scheme is based on experience and modeled according to the needs of the cement industry. Of course, every exploration project is unique and requires a particular approach.

Basically, the scope and extent of every exploration procedure is governed by the following series of criteria:

- **Scope of work**
- **Status of geological knowledge of the area to be examined**
- **Raw materials: exposures, structures, overburden**
- **Time and financial resources available**
- **Climatic conditions**
- **Physiography and topography of the area**

The Holderbank approach

The deposits are investigated in a stepwise fashion from a general overview to more detailed exploration activities. Each step is defined with regard to the respective scope, extent, costs, time requirement, reliability of the results (error limits) and the development of risk.

In practice, the proposed procedure encompasses the following sequence of events:

1) Desk study
2) Preliminary prospecting of raw materials
3) Overall investigations of raw materials
4) Detailed investigations of raw materials

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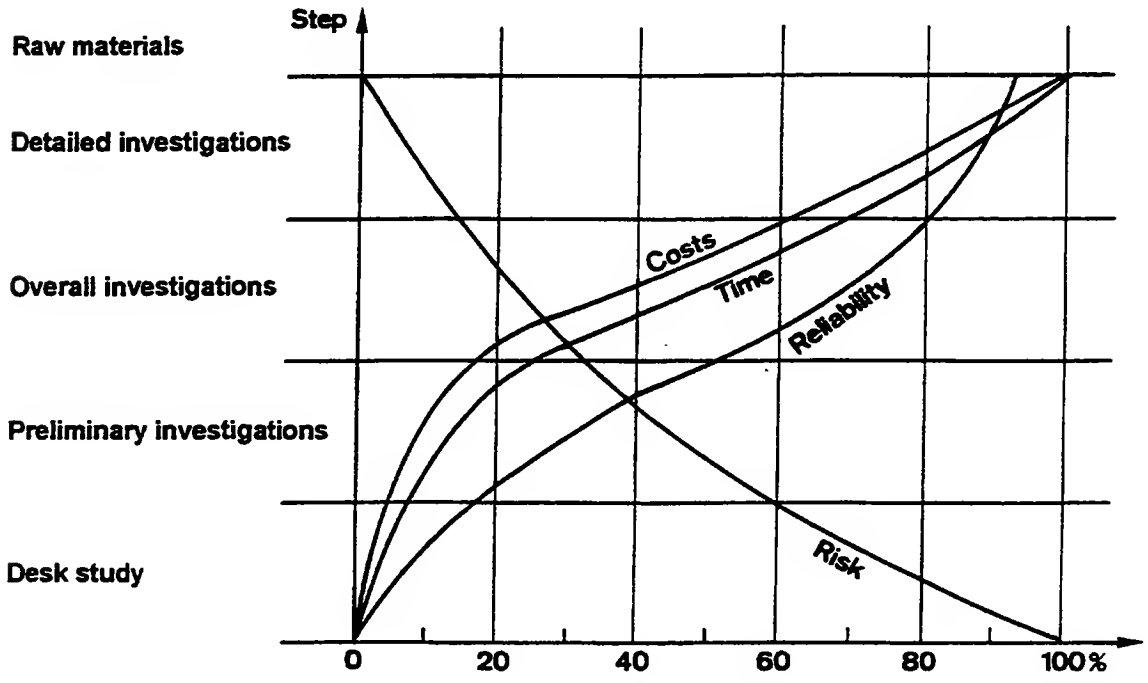
Fig. 1 **Shows the development of costs, time, reliability and risk during the investigations of raw materials.**



2. DESK STUDY

Once the scope of work for required investigations has been agreed on and a quotation therefore accepted a fact finding study is conducted. This study, called a desk study, is carried out with the aim of gathering and evaluating all the available data and information.

In consideration of the cost, field work must be focused on selected areas. A successful exploration programme will therefore begin with a literature search and the geologist will look for the followings documents:

2.1 Geological maps

Scale 1:1'000'000 to 1:25'000 or larger scale if available. Such maps show the distribution of the stratigraphical units exposed in the considered region and the structures, i.e. style of deformation.

Interpretation of thematic maps such as hydrogeological, geotechnical or geophysical can give valuable information.

For example, the geological map of the north west sector of the Kingdom of Saudi Arabia(Fig. 2). The map depicts exposures of limestone and clay on the Red Sea coastal plain, and further inland an amphibolic schist within the metamorphic shield.

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Fig. 2a Geological map of the region of Duba.

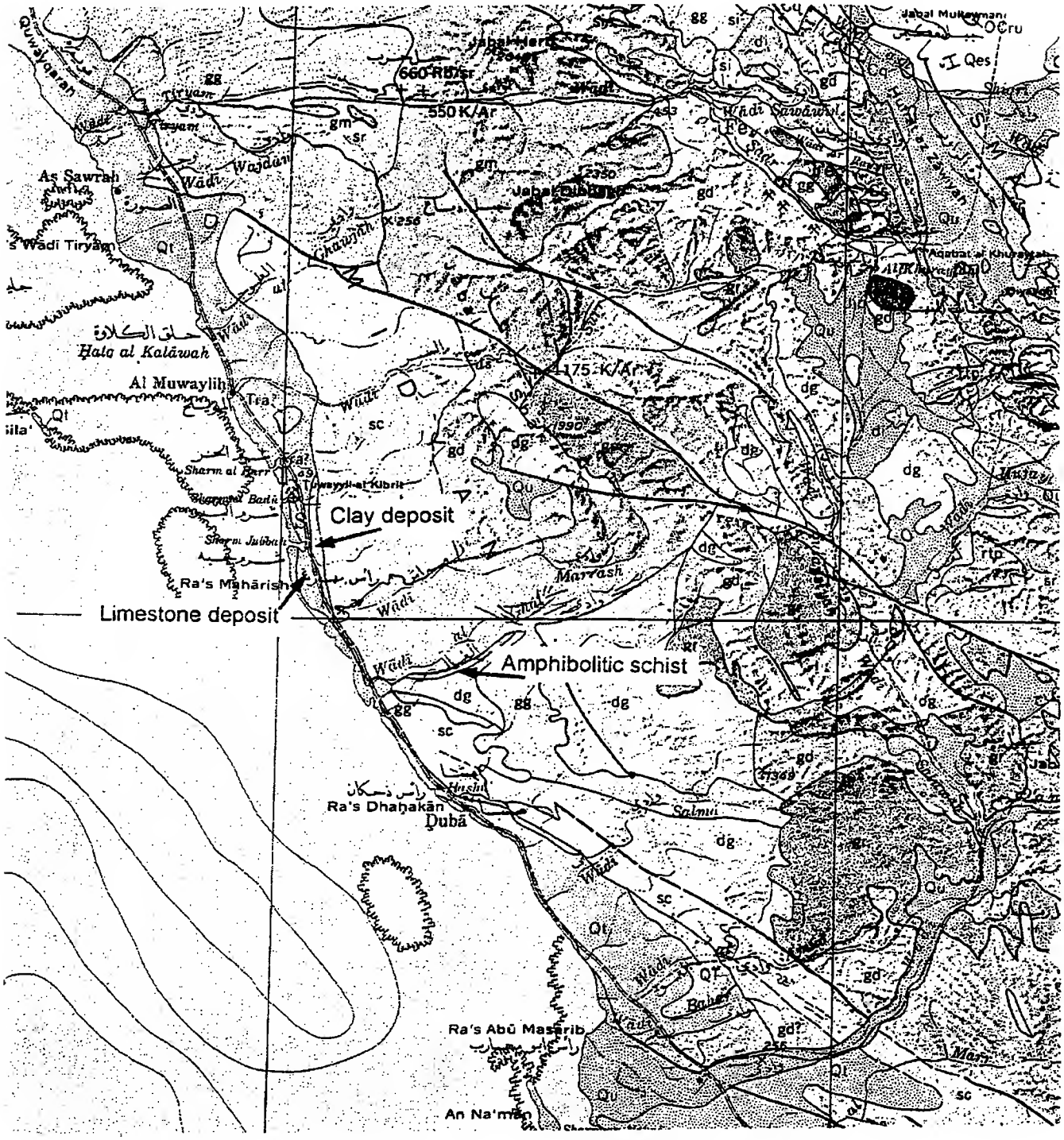

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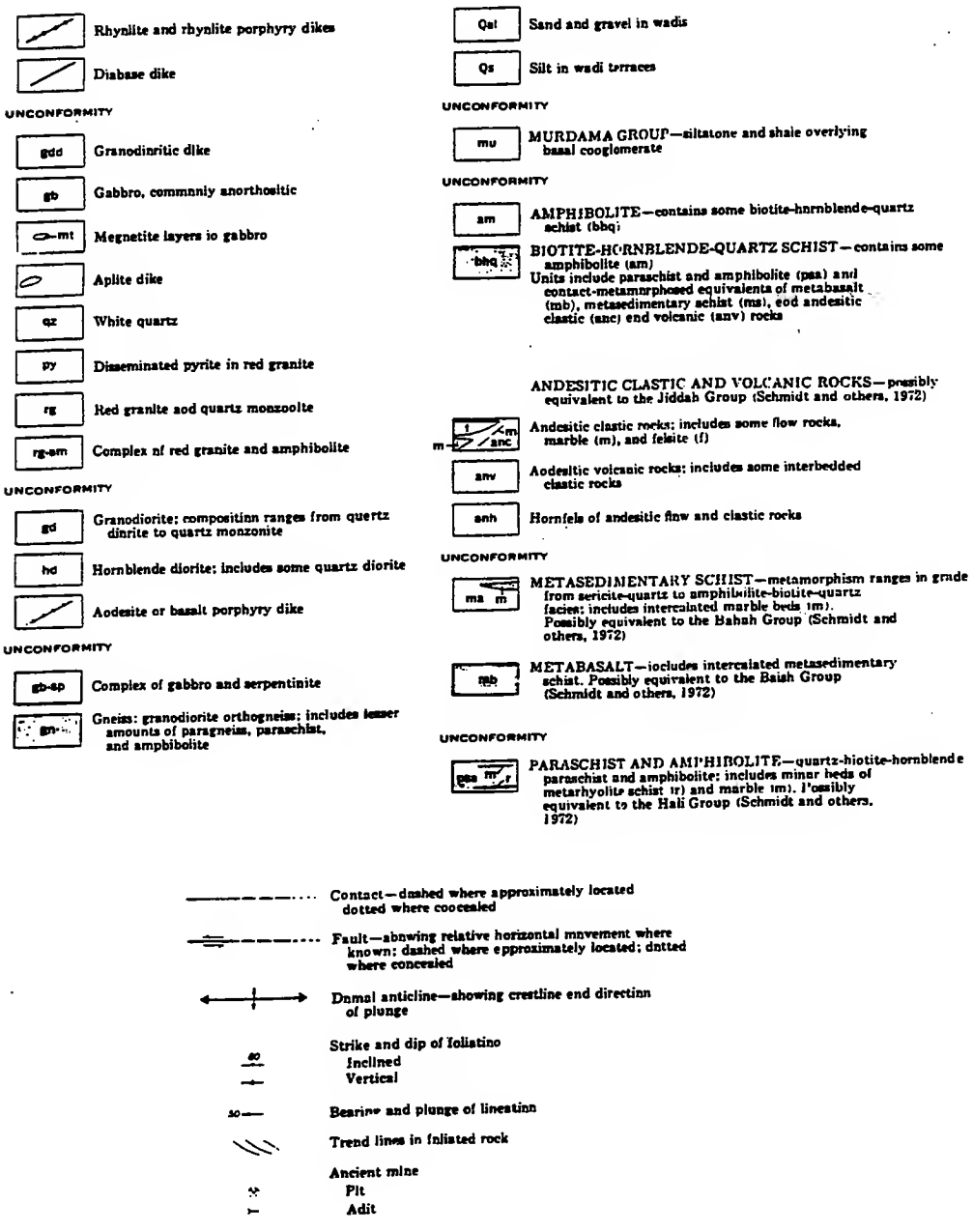
Fig 2b Legend for Figure 2a

DESCRIPTION OF MAP UNITS



2.2 Documents

Documents describing the stratigraphy, (i.e. type of rocks, age, thickness, correlation between different units, variation of facies) and the lithology (i.e. composition of the rocks, the texture, the sedimentological structure) are very valuable to familiarise oneself with the regional geology.

The most detailed reports depict the distribution of the lithologies as well as the mineral and chemical compositions. Such reports or studies are normally done by students at the universities (diploma or PhD, professors). In addition existing company reports can give a lot of information. Thes are however not always available, whilst academic reports usually are.

2.3 Photogeology

A considerable amount of geological information can be obtained from the interpretation of from aerial photographs,. Different rock types can be identified because of their strong tonal and textural contrasts due to:

- erosion levels and patterns
- colour and reflectivity
- vegetation cover
- structure

Lineaments (faults or other discontinuities) may be distinguishable as any alignment of features on an aerial photograph.

Folds are easily recognized and it is often possible to define the fold type (i.e. open, tight).

Fig. 3 shows a folded structure where it is possible to differentiate several lithologies. Dip and strike of the layers and fold axis can be determined.

From an aerial photograph, structure and a lithology can be quickly defined.

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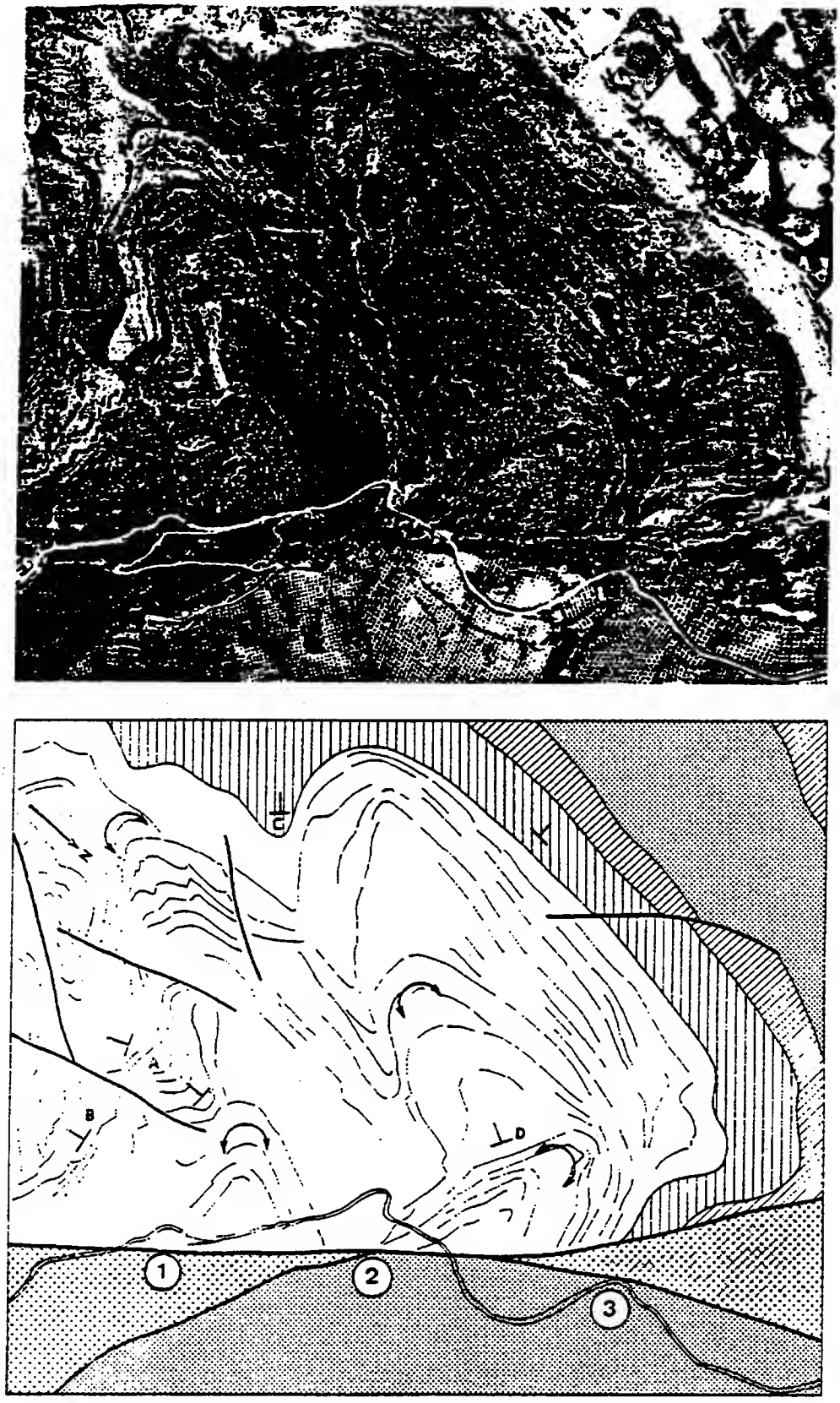
Fig 3: Aerial photograph of a folded region and geological interpretation



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2.4 Remote sensing

Apart from the traditionally used aerial photographs, the satellite pictures have become available as a useful tool for scanning large areas.

With the help of these pictures, the general trends of the regional geology and the lithology can be determined, which is very helpful for any prospecting campaign.

The technique employs devices such as cameras, lasers, infrared, and ultraviolet detectors in satellite or on aircraft.

The following tools are available :

Tools	Resolution (pixels)
Landsat MultiSpectral Scanner (MSS)	80 m
Landsat Thematic Mapper (TM)	30 m
SPOT MultiSpectral Scanner (SPOT-XS)	20 m
SPOT Panchromatic (SPOT-P)	10 m
ERS-1 (European Radar Satellite)	10 m
RadarSat	
Airborne Radar (Radar)	

Passive sensors like Landsat and SPOT can not see through clouds since they depend on naturally reflected or radiated energy. Only an active sensor, like radar, which emits a signal and records its reflection can penetrate clouds.

In heavily vegetated areas or in regions with high sun angles or with a low relief, the use of radar will significantly increase the visibility of the structure.

An example of prospecting for raw materials at Polpaico - Chile, by remote sensing, is given at the end of this chapter.

2.5 Where to look for information?

- HMC Information Services
- University libraries
- Geological survey of the country

Aerial photographs can very often be obtained at the department of the geological survey.

2.6 Results of the desk study

The result of the desk study must depict the geological situation of the most promising deposits.

The geologist will recommend carrying out preliminary field investigations on 2 to 3 deposits.

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Modern Methods of Prospecting for Limestone at Cemento Polpaico

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Searching for limestone deposits in the coastal Cordilleras of Central Chile with satellite photos.

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1. GENERAL

There are very few limestone deposits in Chile. This applies particularly to the central region of this extremely long country of 4000 km. Only in the south (Madre de Dios archipelago) and in the cordilleras of the Andes at an altitude of 3000 to 5000 m above sea-level near the border to Argentina, are there any significant limestone deposits.

The two factories which satisfy the needs of the greater part of the market round Santiago and the rest of the country utilize deposits which incur heavy raw-material costs. Cemento Polpaico utilizes a deposit of siliceous limestone 40 km north of Santiago, a material which requires a special flotation treatment, while Cemento Mélon exploits a lime-stone deposit of good quality situated some 120 km north of Santiago, where it is mined 500 m below ground-level.

In view of this situation, it was decided in 1992 that prospecting for new limestone deposits should be carried out along the coastal Cordilleras, roughly between 31 and 33° of longitude S, representing an area of some 12 000 km^2. One third of this area exhibited geological formations in which limestone could be expected with a certain degree of probability.

The exploration of such a large area demanded many days of field work on the spot. To reduce this to a minimum, the most interesting zones had to be selected beforehand, and then work concentrated on them. The most obvious question was: Where are these areas and how can they be selected? To obtain an answer it was decided to resort to the analysis of satellite photos.

This procedure is used quite often when prospecting for metallic raw material, especially in Northern Chile, and astonishing results have been achieved. One company, for instance, located a large gold deposit in the region of Zapaleri (high cordilleras of the Atacama region) solely with the aid of satellite photos, and this long before the geologists working for the competitors had returned to their offices, analyzed their data and likewise reached the conclusion that the area was of interest. With non-metallic deposits not so much experience has been acquired, but it is nevertheless very promising, as the discovery of large saltpetre shows deposits in the Atacama Desert. And thereby hangs a tale: When examining the satellite photos, a line was observed which crosses the entire desert and consisted of saltpetre. When this remarkable phenomenon was examined on site, the line was found to be the route by which the Incas used to transport saltpetre to Peru. The saltpetre recorded in the photos was lost on the way by the pack animals.

In view of the excellent results obtained with this method it was decided that it should be used for limestone, despite the lack of experience with this material. First the feasibility of the method was tested with photos of the known deposits in the area surrounding the works of Cemento Polpaico. They are located between 32°30' and 33°30' longitude (Fig. 1). Owing to the proximity of the area and its good accessibility, it was an easy matter to check the results. It ought to be possible to see the limestone deposits of Cemento Polpaico and Mélon, which were of use in analysing the satellite information.

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These photos and the software for their analysis cost about US$ 4000 per photo, plus US$ 10 000 for the program. The Centro de Estudios Espaciales of the University of Chile also possesses some photos, as well as suitable equipment and experts, so that it was possible to cooperate directly with this institute at a cost of US$ 500 per day. For the three days occupied by evaluating the photos the total cost worked out at US$ 1500 for an area of 5000 km2. Assuming that a field geologist would have cost US$ 50 per day, it would have been possible to finance 30 field days at this price. But then the geologist would have had to explore 170 km2 per day, which of course is impossible. Thus the method of analysing satellite photos costs decidedly less.

2. THEORETICAL INTRODUCTION

2.1 General

The use of satellite photos of the Earth's surface has grown in importance for geological investigations since the seventies, when the first satellite of the LANDSAT series was put in orbit. With satellites certain information can be acquired quicker and more economically than with conventional methods.

The photo gives a representative impression of the surface, though vegetation can prove disturbing. For this reason the method provides better results in semi-dry, desert-like areas where the rock strata are more exposed.

In the latest method known as remote sensing, a sensor (or a group of sensors) suitable for measuring the intensity in a given waveband in the electromagnetic spectrum is mounted on an aircraft, a helicopter or a satellite, so as to cover the largest possible area for prospecting. There are a number of satellites fitted in this way, such as LANDSAT, SPOT and NOAA, which were developed for the particular purposes.

2.2 LANDSAT satellites

The LANDSAT satellites (MSS and TM) which were put in operation by the NASA, are located in a polar orbit which passes the illuminated area of the Earth several times a day moving west-wards so that within 16 days the whole of the Earth's surface has been covered.

The photos come from two sensors: a Multi-Spectral Scanner (MSS)and a Thematic Mapper (TM). Both of them divide the image into small elements (pixels) and measure their intensity of radiation. The components of the sunlight reflected by the Earth are detected in both the visible and the infrared region. This digital information is transmitted to Earth, where it is processed to products resembling aerial photos, false-colour photos, which are then analysed and evaluated.

The TM sensor is more useful for geology than MSS. It produces images in 7 bands of the electromagnetic spectrum, each covering an area of 185 x 185 km, the resolution (definition of objects) being between 30 and 120 m (thermal infrared). Each band represents a segment of the electromagnetic spectrum, defined as follows.

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Band	Micrometer	Spectral range
B 1	0.45-0.52	visible blue
B 2	0.52-0.57	visible green
B 3	0.63-0.69	visible red
B 4	0.76-0.90	near infrared
B 5	1.55-1.75	remote infrared
B 6	10.8-12.5	thermal infrared
B	2.08-2.35	remote thermal infrared (SWIR)

2.3 Generation of images

The data stored on magnetic tape can be processed at any time to obtain the desired products. These result from a combination of three information bands, allocated beforehand to the three traditional colours red, green or blue (R/G/B).

The information bands may contain one of the following: The various bands which are received directly from the satellite, the product of a mathematical operation based on information from various real bands (band sum, band quotient, etc.), or the PC (principal components). Each separately measured band contains a definite component of the other bands depending on the technical features of the sensor. Via statistical treatment of the data of the various bands of an image, algorithms are generated (PCI, which in turn generate new bands with bands independent of one another. When an image is displayed, it can be modified so that the information needed is acquired.

2.4 Analysis of the information

When generating an image, one should have a clear idea of what information is needed and how it should be interpreted. Experience has proved that there are definite bands and combinations that are particularly well suited for use in geology. In order to analyse an image (on the monitor or as a colour printout), one should have a clear idea of what the false colour concept involves. This means that the colour on the picture is not necessarily the colour of the Earth's surface, but:

- the colour results from the allocation of a definite band to a colour on the monitor
- the colour results from the actual colour of the surface;
- the various colours in the picture indicate a difference in the composition of the reflecting surfaces this information is purely of a qualitative nature and does not provide any information on the material, nor the percentage of a definite compound present.
- The colour observed results from the reflective properties of the analysed surface, the degree of weathering, the humidity, the structure of the surface etc.
- It should also be pointed out that in view of the fact that a satellite photo is based on the reflection of sunlight by the Earth's surface, it depends directly on the time of day, the weather conditions, the season of the year and the angle of observation at the moment the photo is taken.

The simplest method is to compare the false colour pictures of satellite photos with known geological formations in an area. If it proves possible to allocate a colour to a definite kind of stone, it may be assumed that at places where the colour is the same, the lithological conditions will be the same as those in the corresponding reference area.

Although the various bands alone already provide definite information, they can be combined and the properties of the false colours utilized to stress other useful features.

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Below, some of the most frequent combinations will be explained (shown in brackets is the colour that appears on the monitor):

B3 (red), B2 (green), B1 (blue): With this combination the various bands are shown with their real colour, a picture being obtained which corresponds to the natural colouring.

B5 (red), B4 (green), B1 (blue): Here a distinction can be made between various rocks and soils in an area with no vegetation. This combination is useful for exploring lithological conditions in volcanic and sedimentary strata.

Other pictures are obtained with the aid of the quotients between the bands. B7/B4 (red), B4/B3 (green), B5/B7 (blue): In this case red demonstrates the presence of iron, green corresponds to vegetation and blue to hydyrophiles and carbonates.

In areas covered with vegetation the ground cannot be seen directly, but employing the following assumption, conclusions can be drawn as to the lithological conditions and the composition of the subsoil. Plants react to the presence of different mineral compounds, the composition of which in turn depends on the lithological conditions. Thus on soil that is rich in calcium carbonate, the plants growing on it may suffer from a disease known as cloriosis (which affects their chlorophyll content). If this anomaly is observed in the plant cover, the area observed may be regarded as of interest in the prospecting for rock containing lime.

The analysis of the PC information (principal components) is rather more complicated because it is obtained with the aid of an algorithm and therefore bears no direct relation to the real conditions on the Earth's surface. This algorithm is specific for each PC and each photo. Its importance can be determined by analysing the formula, the terms of which are different weightings of the various bands. This means that, according to the weighting factor a definite PC is influenced more by definite bands, which then give it is true meaning.

3. GEOLOGICAL EXAMINATION

3.1 Introduction

The area selected for the examination corresponds to a segment of the coastal Cordilleras situated roughly between 32°30' and 33°30' S in longitude. A photo was used which had been taken by LANDSAT TM at 10 a.m. on a summer morning (March).

In summer this area also exhibits quite an appreciable coverage of vegetation (about 70%), but it decreases notably with increasing altitude. Although this affects the method, it was hoped that interesting places would be found in the uncovered areas as well as limestone outcrops in covered areas. At the strongest resolution of the sensor deposits 30 m or more in size would saturate the photo to such an extent that identification would be quite easy.

In view of the fact that carbonate cannot be identified by the bands detected by TM and, moreover, certain bands are easily mistaken for hydroxyls, it was decided that the PC and the comparison method should be used, using the limestone outcrops in the region of Polpaico as reference pattern. The limestone deposits of the open quarry could not be used as reference because their white colour and the outstanding reflection properties of the surface saturated all bands. The photos obtained by combining different PCs and bands were therefore modified so that their colours stressed the zones in which there were known to be limestone outcrops, whereas the rest of the photo was darkened, assuming that areas containing unknown limestone outcrops would be stressed to the same extent.

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On three pictures in which known places were discernible with high definition, zones with the same colour were identified. These stressed regions were observed in at least two of the pictures. Finally, those places within the geological unit were selected which very probably contain limestone deposits.

3.2 Results of the photos

Below various places are presented which were selected from the photos and corresponded to the following combinations: Fig. 1: PC1/PC2/B1, Fig. 2: PC1/PC2/PC3, Fig. 3: PC2/PC3/PC4. They are shown to a scale of 1 :1 000 000 on the enclosed geological map.

From the comparison of the results of the various photos, the areas marked in Fig. 4 were selected. Already known limestone deposits are also marked on it. There is a clear relationship between their position and the selected regions for which reason it was assumed that a method suitable for the subsequent field work had been employed.

Fig. 1: PC1/PC2/B1: Points with a clear yellow colour and/or very weak red pigmentation were selected.

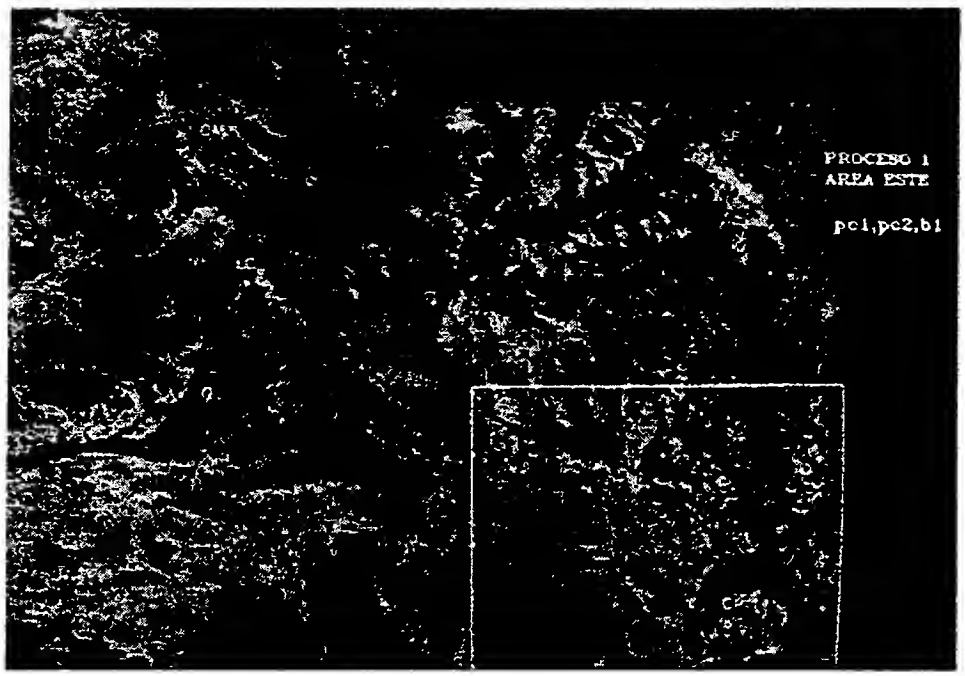

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Fig. 2: **PC1/PC2/PC3: Points with a pink colour and light green pigmentation were selected.**

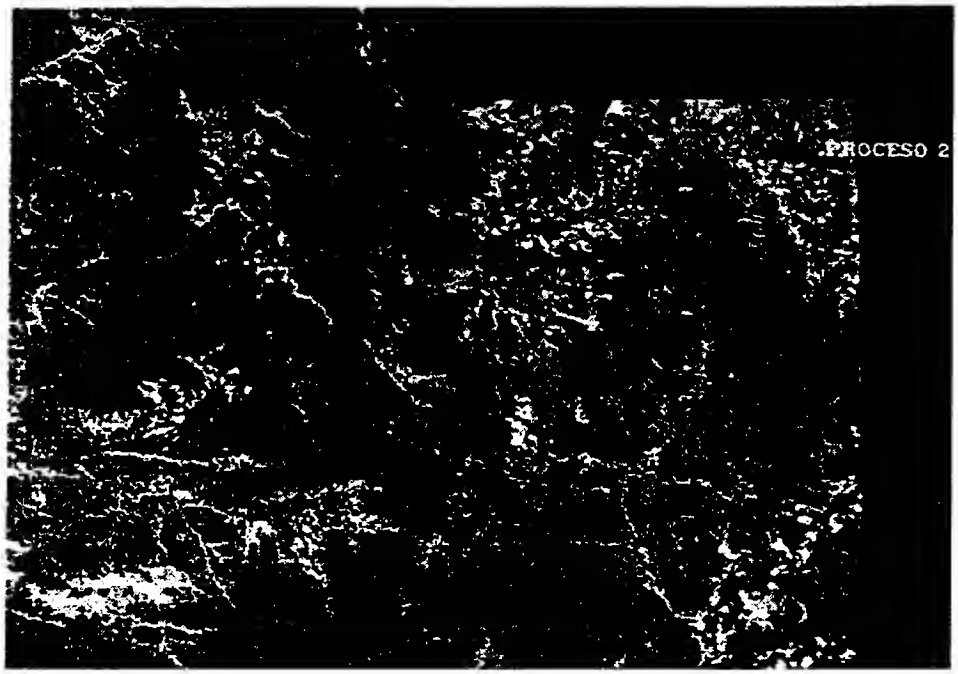
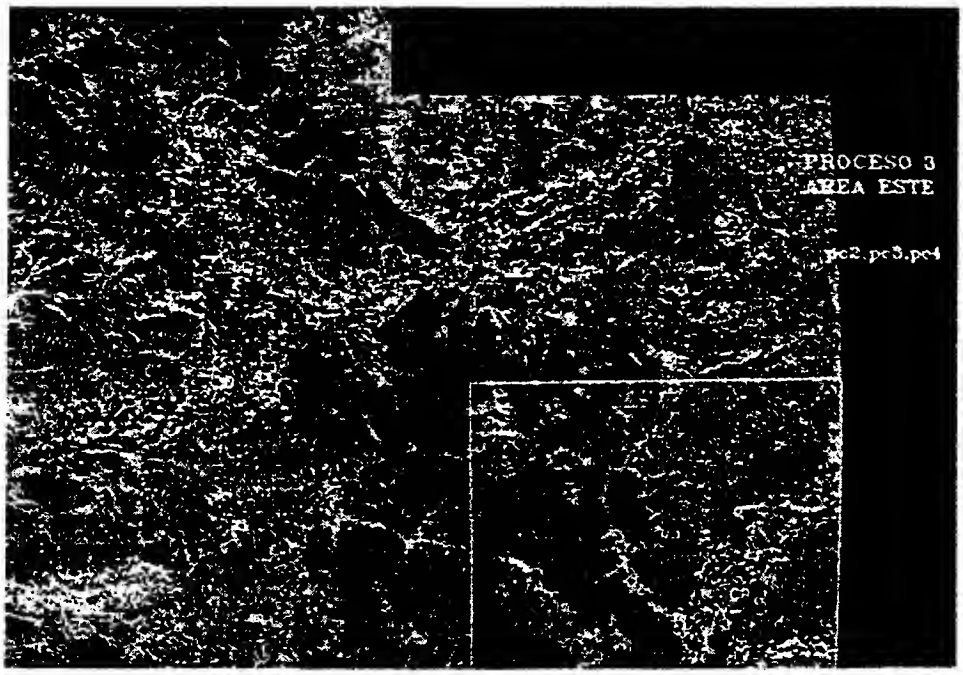


Fig. 3: **PC2/PC3/PC4: Points with a definite olive-green colour and a clear light-blue tinge were selected.**



3.3 Results of examinations on site

Having determined the interesting areas, the zones were subsequently examined on the spot where there were no known deposits. These were located in the geological units Jld, Jdm J2 and west of the middle of Ki2 and KT (this work was carried out on a geological map to a scale of 1 :250 000, which explains why much of the analysed geological information is not reproduced) Limestone deposits were found.

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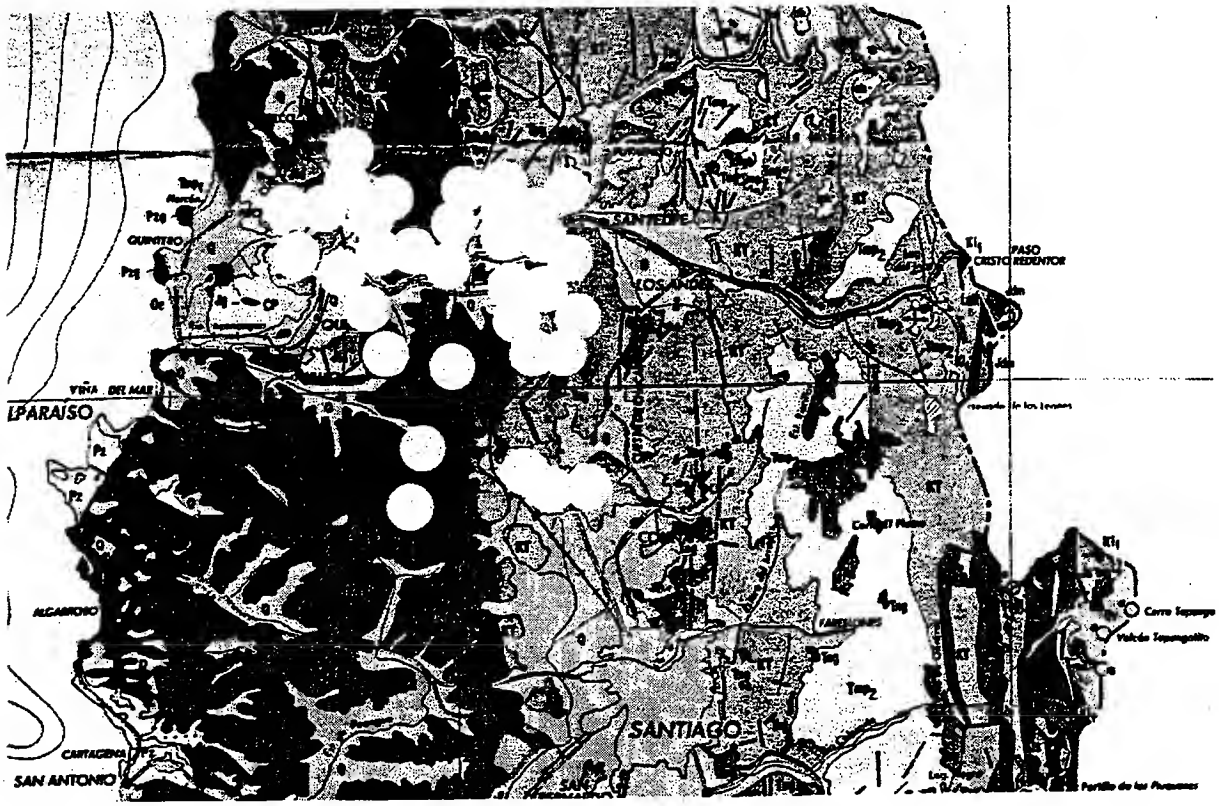
Of the limestone deposits found only two proved favourable (> 80% CaCO3). Again, only one of these exhibited a volume of 1 million tonnes. It was located to the East of Puchuncavi, which had already been surveyed 2 months previously.

Judging from the limestone deposits found the project could not be declared as successful; this by no means applies to the geological aspect. Indeed, the new method proved to be extremely effective, because despite the difficult conditions of the terrain (above all due to the vegetation) a high correlation index was obtained (in 20 of the selected 32 areas there were known or newly found deposits).

The results proved that no further deposits can be expected in the area, which in turn forces the company to develop new geological strategies.

In the selected areas where finally no limestone was found, large areas of Andesite were found the colour of which closely resembles that of limestone or rocks with a weathered surface rich in carbonate. Other methods were also employed for exploring areas in which, according to literature, there are limestone deposits but which, on account of the dense vegetation (100%), could not be recognized on the photos (above all west of Polpaico).

Fig. 4 **On account of the comparison of the results of the various photos the areas shown were selected**



4. CONCLUDING REMARKS

This method can be used in the regional search for limestone deposits. The subsequent local search does not last long and involves lower costs than when conventional methods are used (about 0.5 US$/m2), although the initial investment appears high.

Exclusive use of this method cannot solve the problem. It is essential to carefully study the literature at the same time, in order that adequate results may be obtained.

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As regards the investigation carried out, it is judged to be a success from the geological point of view. In addition to the results (no limestone reserves in the area examined) it showed Polpaico the necessity for developing new strategies to locate additional reserves of raw materials in other areas.

We should like to thank Cemento Polpaico for financing and supporting us in the development of this new method, and the experts of the University of Chile for their active cooperation in analysing the information.

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Chapter 4

Preliminary Field Investigations of Deposits

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Preliminary investigations of deposits

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1. INTRODUCTION

In the desk study, the regional geology is depicted and some potential areas are selected for further investigations. All information obtained so far, has been collected from available published documents and maps. During the following step, known as preliminary field investigations, the geologist is confronted with geological features and problems as found in the field. The geologist will describe the nature, disposition and the structure of the rock exposures of the selected areas in order to gain a first impression of the quality and reserves of several **potential** sites.

The preliminary investigations are the highest risk stage in exploration. For this reason, the geological knowledge, experience and the skills of the observer are of paramount importance to ensure a sound basis for further work. The preliminary investigations must therefore have the lowest unit cost ($ / per square km). If the results are negative, the whole project has to be stopped and similar investigations will have to be conducted in other regions or areas. Positive results call for further, thorough investigations in the promising areas. In any case, after the completion of this step, the client should be in a position to decide on whether or not further investigations should be carried out.

The first of the so-called GO / NO GO decisions.

2. GEOGRAPHICAL SITUATION OF THE DEPOSITS

In this first reconnaissance, the morphology of the areas under consideration will be described in order to show a real picture about

- Transportation infrastructure
- Potential plant site
- Climate
- Morphology of the deposit
- Quarrying possibility
- Environment

3. GEOLOGICAL SITUATION OF THE DEPOSITS

The exposures are described according to lithology, colour, mineralogy, fossil and microfossils. Each outcrop is recorded on a map (sketch or good draft) of the area drawn by the geologist. The structure of the rocks, for instance dip and strike, are measured and recorded on the map.

The aim is to show

- the distribution of the different lithologies of the deposit under consideration
- to calculate the inferred reserves and
- to make a preliminary qualitative determination based on surface sampling.

Further on in this chapter, we will to discuss the quality of the raw materials and the problems associated with sampling procedures

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4. SAMPLING

In the preliminary investigations, the quality of the raw materials is determined by surface sampling only. These samples provide an initial, tentative assessment of the quality of the deposit being investigated. The geologist establishes a sampling programme with due consideration of the geological situation, time and finances available. The sampling locations and the methods should be chosen in order to provide maximum information.

Surface sampling is a delicate task, which requires skill.

Sampling should be **representative** of the exposure.

For example, the exposure in Fig.1 shows alternating thick layers of limestone and shale. Vegetation covers the interbeds of shale. A poor sampling process, in which only the exposed layers of limestone are sampled, will give an incorrect image of the chemical character of the outcrop. The titration value of the limestone layers amounts to 90%. However, the real titration value of the total outcrop (including the interbeds of shale) is only 72 %.

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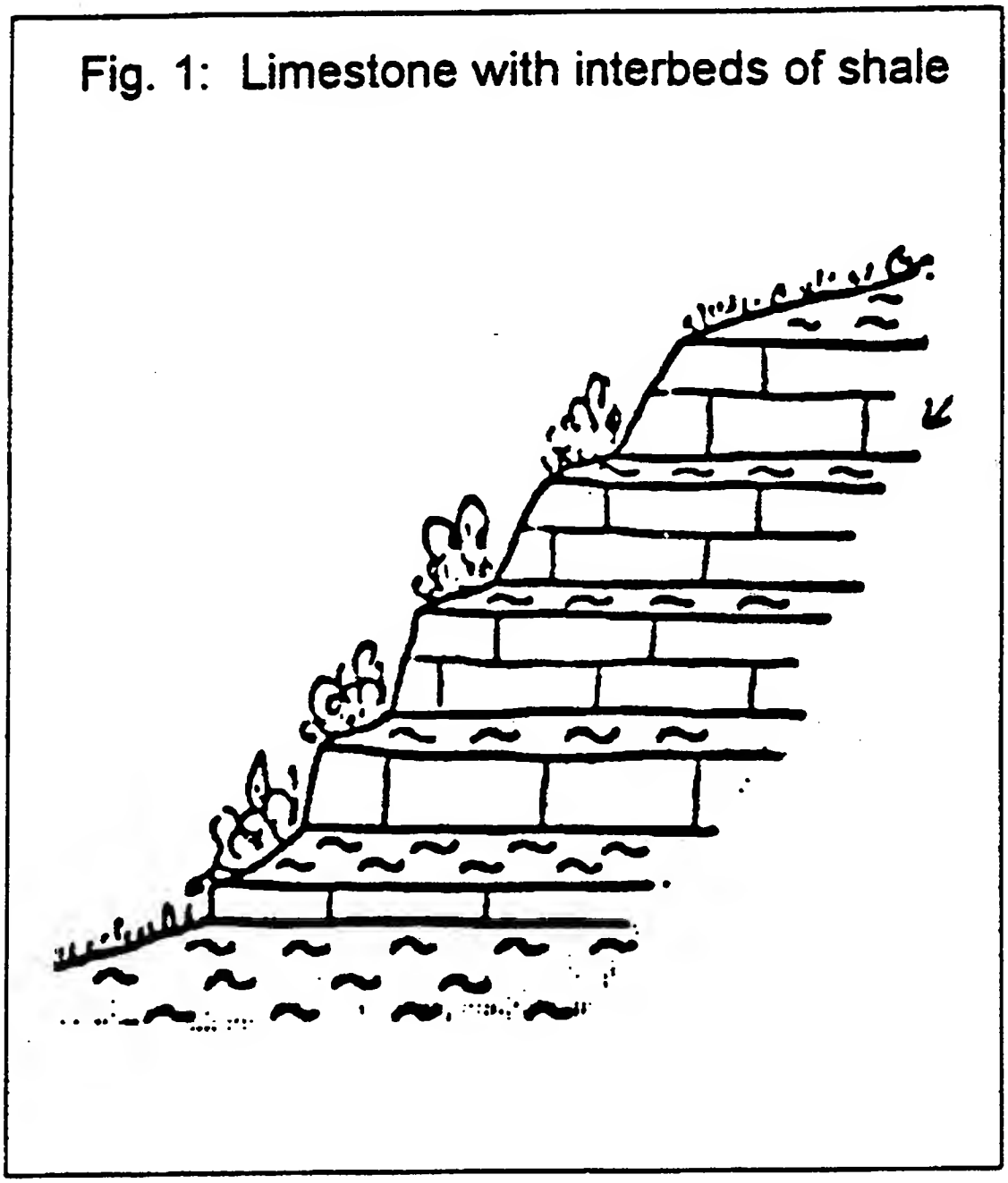
Fig.1 Limestone with interbeds of shale



4.1 Sampling methods

Surface sampling can be carried out in different ways depending on :

- homogeneity
- hardness of the rocks

Sampling must be carried out perpendicular to the stratification.

4.1.1 Homogeneous and hard rocks

Chip samples

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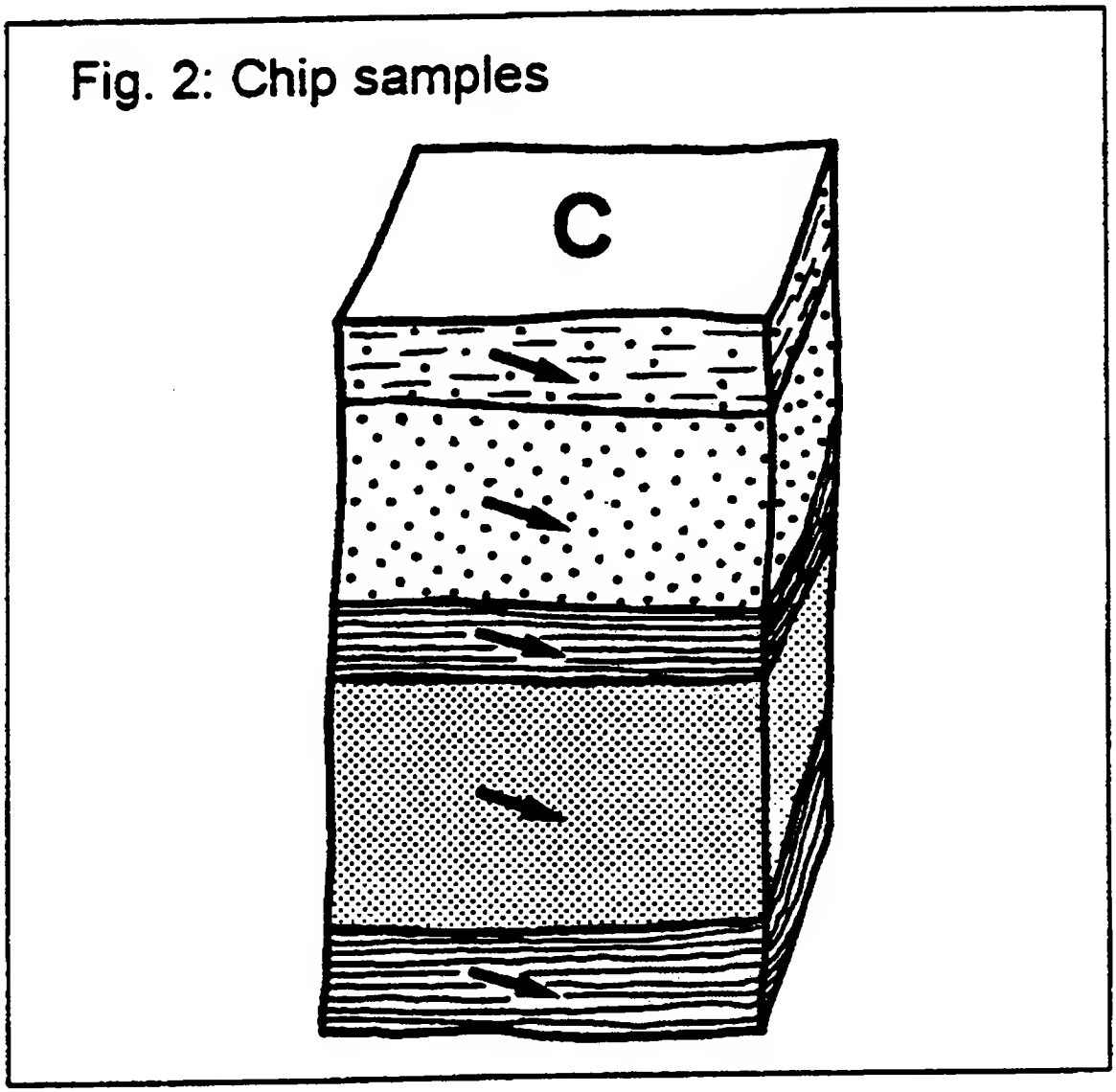
One sample, which is considered as representative, is taken from every horizon (strata) or from available exposures.

Fig. 2 Chip samples



Composite chip samples

The same quantity of chip material is taken at a constant interval. All chip samples from one exposure are placed in a bag which is representative of a given thickness or composite horizon. A good knowledge of the quality of the exposure is obtained.

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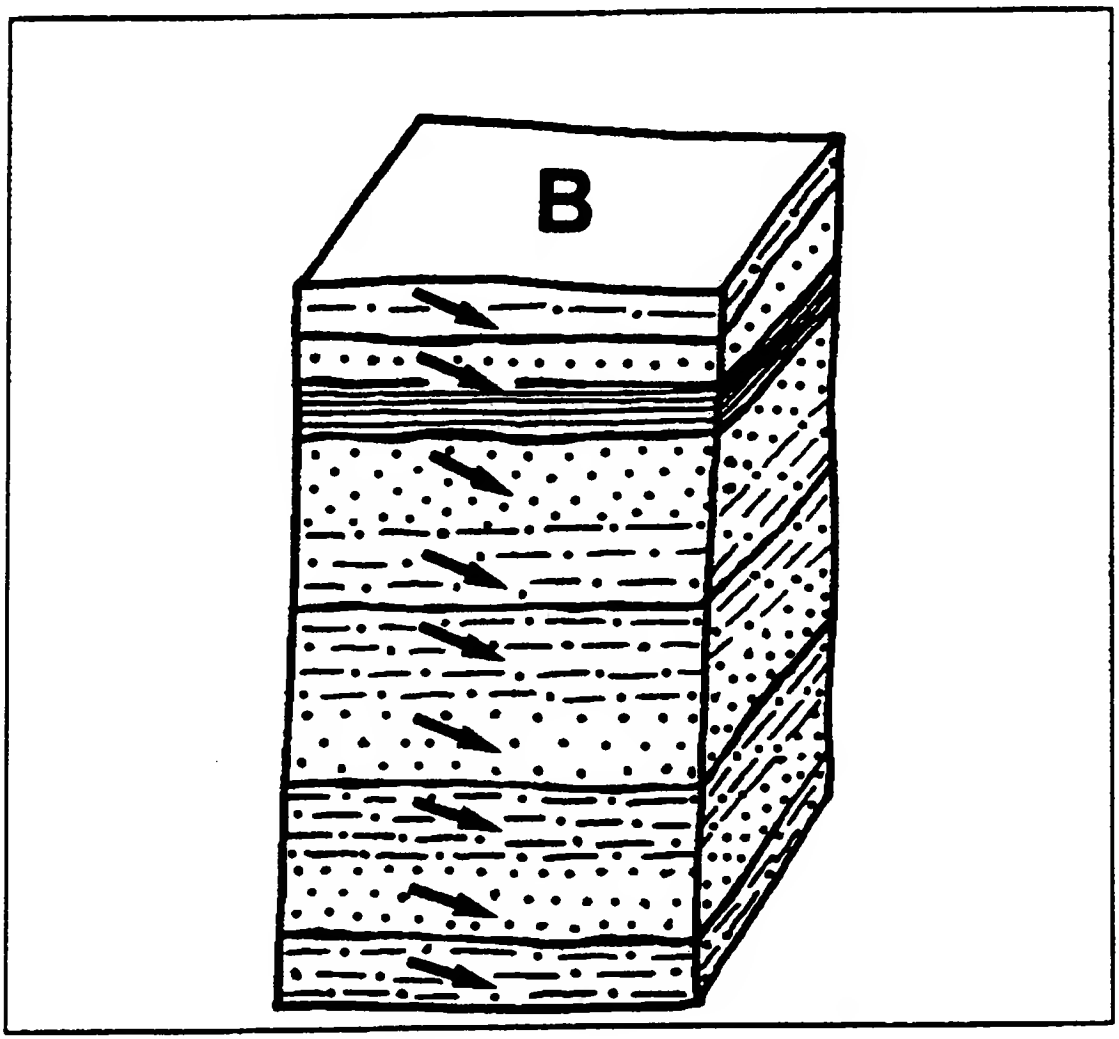
Fig. 3 Composite chip samples



4.1.2 Heterogeneous and hard rocks

Pondered composite sample

Every layer will be sampled. The quantity of materials is dependent on the true thickness of the strata.

The chemical composition of the exposure is well defined.

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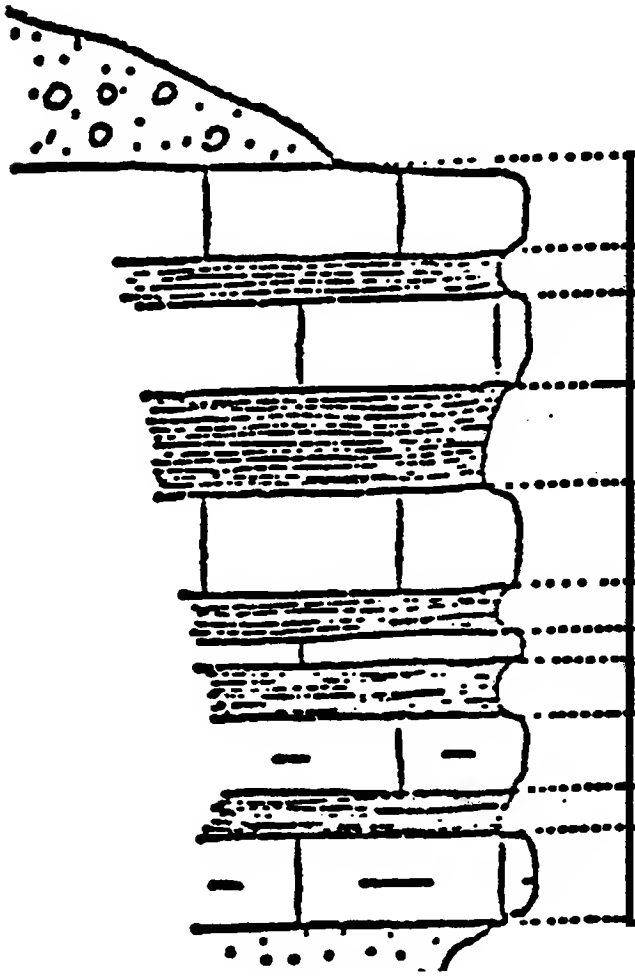
Fig. 4 Pondered composite samples



Thickness	Proportion in the mix
1.20	3
0.60	1,5
1.40	3,5
2.00	5
1.40	3,5
0.80	2
0.40	1
1.00	2,5
1.20	3
0.40	1
1.60	4

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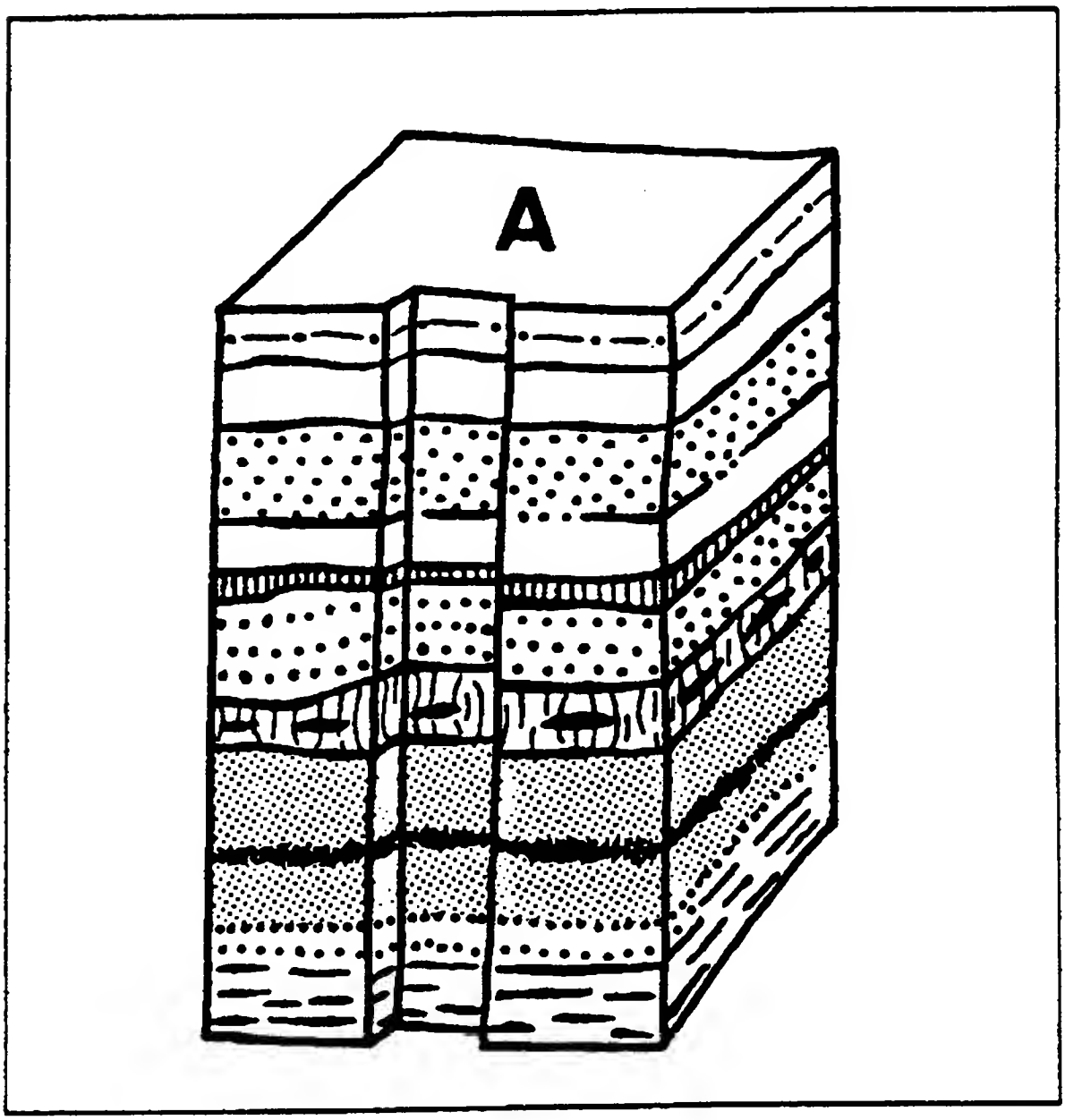
4.1.3 Heterogeneous and soft rocks

Channel sampling

The sample is collected along a continuous channel cut at right angles to the stratification. That is the best method to obtain an accurate chemical composition of the whole exposure.

Fig. 5 Channel samples



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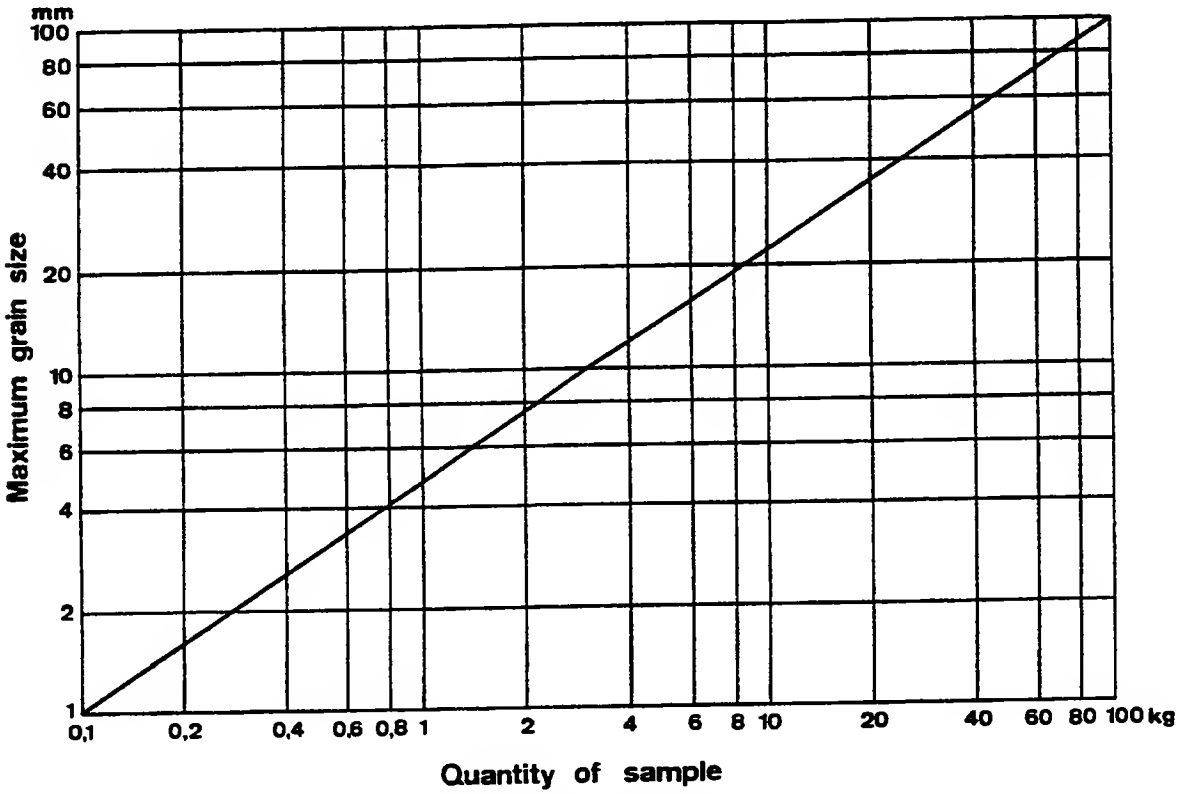
4.2 Quantity of materials

To obtain a representative sample, the quantity of material to sample depends on the granulometry of the material. Fig. 6 shows the relation between maximum grain size and the sample quantity.

Fig. 6 Maximum grain size versus sample quantity



In some cases, the sample quantity is also determined the testing programme to be carried out. Experience has shown that the following quantities of cement raw materials are necessary for laboratory tests conforming to HMC standards (Table 1).

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Table 1 Sample quantity for various tests

Test	Required Quantity
Chemical analysis Mineralogival analysis Physical properties	1 - 2 kg per component or fist-sized specimens, eventually
Water Analysis	1 litre
Fuel	1 kg, resp. 1 litre
Compressive strength	30 x 30 x 40 cm block
Crushing properties	blocks of > 20 cm length or edges
Grindability properties	10 - 20 kg
Lime burning	10 x 10 x 10 cm block
Slurry properties	1 litre
Filtration test	50 - 150 kg

Normally, not only one but also a combination of tests is conducted on the same sample. 1 to 2 kg of raw material component is sufficient to yield results on chemical and mineralogical compositions, for examinations on granulation and burnability properties, to determine physical parameters such as moisture content, porosity, density, water absorbtion etc.

5. QUALITY OF THE RAW MATERIALS

The quality of the raw materials can be appraised by

- geological description of the outcrops
- testing for chemical and thechnical parametrs

For all the types of raw materials, the following geological description of the exposure will be done:

- type of rocks, mineral content
- diagenitic process
- granulometry
- colour
- fossil content
- type of porosity
- structure:
  dip and strike of the layers; thickness of the layers laminations; joints, density of joints, length; faults and folding.

The macroscopic features control the physical and mechanical properties of the rocks. For instance, the hardness of the rock depends on the type of diagenesis and porosity; the shape of the crushed rocks depends on the anisotropy of the rocks (fine stratification, lamination, foliation).

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5.1 Quality of raw materials for cement manufacturing

In the preliminary investigations, the raw materials are appraised by the chemical analyses of the surface samples and by the general features of the rocks at the outcrops.

We want here to focus our attention on few components which could be deleterious for the clinker production. A complete assessment of the raw materials quality parameters is shown in chapter 6.

The table 2 depicts the most frequent deleterious constituents containing in the raw materials

Table 2 Maximum permitted levels of deleterious elements in the raw meal

Constituents	Max % in Raw Meal (LOI free basis)	Remarks
MgO	< 5 % (abs. max. 6 %)	MgO come mainly from the calcareous component
SO_3	< 1 %	In general, SO_3 comes from the argillaceous-siliceous component
K_2O Na_2O Na equivalent Low alkali	< 1.4 % < 0.8 % < 1.2 % < 0.6 %	Argillaceous-siliceous components contain the alkalis
Cl^-	< 0.02 %	All components can have high Cl^- content

From a physical point of view, the outcrops are investigated in order to determine the following features:

- Homogeneity of the deposit (interbeds, horizontal or vertical variation in lithology,...)
- Presence of chert, i.e. cryptocristalline quartz, occurring as nodules, lenses in limestone and shales
- Proportion of abrasive minerals (quartz grains)

All these parameters are significant to assess the suitability of a deposit. Heterogeneous deposits will require selected quarrying.

5.2 Quality of pozzolanas

Pozzolana is a siliceous or siliceous - aluminuous material which in itself possesses little or no cemtitious properties. Finely ground and in presence of moisture (water), the material reacts chemically with calcium hydroxide, $Ca(OH)_2$ at at ambient temperature to form compounds possessing cementitious properties (calcium silicate and calcium aluminate minerals).

The natural pozzolanas are mostly volcanic products such as ashes, tuffs (consolidated ashes) and lavas. Other well-known natural types are diatomaceous earth, and opaline shales.

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The quality of pozzolanas can only be appraised by testing of samples. The following properties of the pozzolanas should be considered in the testing programme:

- Petrography: description of the texture and the mineral composition
- Chemical composition
- Mineralogical composition: Proportion of amorphous, glassy material or content of zeolithes by X ray diffraction
- Hydraulic index: Keil index
- Pozzolanic activity. Holderbank norm follows the ASTM C-311-90 standard.

Selection of suitable deposits should take place based on the tests results of several representative samples.

Once selected, a grindability test should be carried out on the deposit.

5.3 Quality of aggregates

The outcrops have to be carefully described as mentioned under paragraph 5. However, several tests must be carried out on representative samples.

- Physical tests:
  - density
  - porosity
  - surface texture of crushed material
  - shape of the crushed material
  - proportion of dust/ fine material
- Mechanical tests
  - compressive uniaxial strength
- Chemical analysis
  - Cl content
  - Alkali-silica reaction
  - Alkali-carbonate reaction
  - Organic content

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Applied Geophysics

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1. APPLIED GEOPHYSICS: ASSISTANCE IN THE INVESTIGATION OF RAW MATERIALS

When investigating raw materials, the geologist is often confronted with many problems such as: thickness of overburden, localisation of lithological contacts or geometry of the deposit in complex geological situations. A geophysical reconnaissance can assist the geologist: to accurately define the geometry of the deposit between the drilling locations, to determine the waste/ore ratio, as well as the accurate reserves of raw materials. Due to high costs, the number of drill holes which can be done is limited. Accurate correlation between them is not always assured and potential nasty surprises can arise during the quarrying.

Applied geophysics may also be helpful for the determination of the physical characteristics of the rocks, i.e. rippability. In the quarry, detailed geophysics can determine cavities. In a karstified deposit, this geophysical reconnaissance can overcome quality fluctuations caused by the cavity materials. A constant quality can be maintained.

2. APPLIED GEOPHYSICS: MANY ADVANTAGES AT LOW COST

The main advantages of geophysics are as follows:

- Investigation over a greater surface
- expedient execution
- results in form of maps and profiles
- accurate calculation of reserves
- planning of the drilling campaign
  - optimisation of the drilling location
  - reduction of the quantity of drilling
- Lower costs

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Table 1 compares the cost of a geophysical investigation with that of a theoretical drilling campaign in a gravel deposit.

Table 1

		Geophysics	Drilling
Logistic	portable	heavy drilling machine	
Data	Information	17 vertical electrical soundings	17 drill holes (850 m)
	Additional exploration	3 drill holes (150 m)	none
Duration	Field acquisition	4 days	85 days
	Interpretation	2 days	3 days
	Additional drilling	15 days	-
	Total	21 days	88 days
Cost		geophysics: 10'400 US$	88'000 US$
		add. Drillings: 16'000 US$	
	Total	26'4000 US$	88'000 US$

drilling advance: 10 m/day drilling cost: 100 US$/m specialist: 1000 US$/day
2 labourers: 300 US/day each geophysical tools: 2000 US$

The cost of the application of refraction seismics will be within the same range. On the other hand, the cost of the application of reflection seismics would be around 30% more expensive.

Nevertheless, the geophysical methods are indirect methods of reconnaissance. The results are physical values which have to be interpreted in terms of rocks and/or sediments. The experience of the geologist / geophysicist and a good knowledge of the local geology are indispensable prerequisites for obtaining very accurate results. The main disadvantages are as follows:

- no samples
- drill hole of calibration

Calibration of the geophysical values within a few drill holes considerably improves the accuracy of the results. It is also advisable to sink two to three drill holes after a geophysical campaign.

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3. GEOPHYSICAL METHODS AVAILABLE AND THE CRITERIA OF CHOICE

To solve the above problem, which often appears during the investigation, the geologist has the following methods at his disposal.

3.1 Geoelectrics

An electricity field is emitted by one or two electrodes, and the difference of potential created by this field is measured by two other electrodes. Different types of array are available to the specialist. The most frequently applied arrays for prospecting are as follows:

- Vertical electrical sounding (YES) Schlumberger type
- Four pole profiling (4P-PRFL)

The goal of the geoelectrical methods is as follows:

- With VES, to define the true resistivity . This allows to determine the kind of rocks and the thickness of the formation
- With 4P-PRFL, to map the distribution of the apparent resistivity. This reveals the structure of the subsurface for a given depth of investigation depending of the array configuration

3.2 Seismics

Waves are artificially generated by means of explosions or hammer within the earth crust. The waves are reflected and refracted on the in homogeneous plans (fault) or on stratification. The wave characteristics and the time during which the seismics waves pass from the emission centre to the geophones are recorded.

In refraction seismics, only the first arrival waves are considered. The Fig. 1 shows a record of refraction seismics. This method is suitable for determination of rock quality (i.e. rippability).

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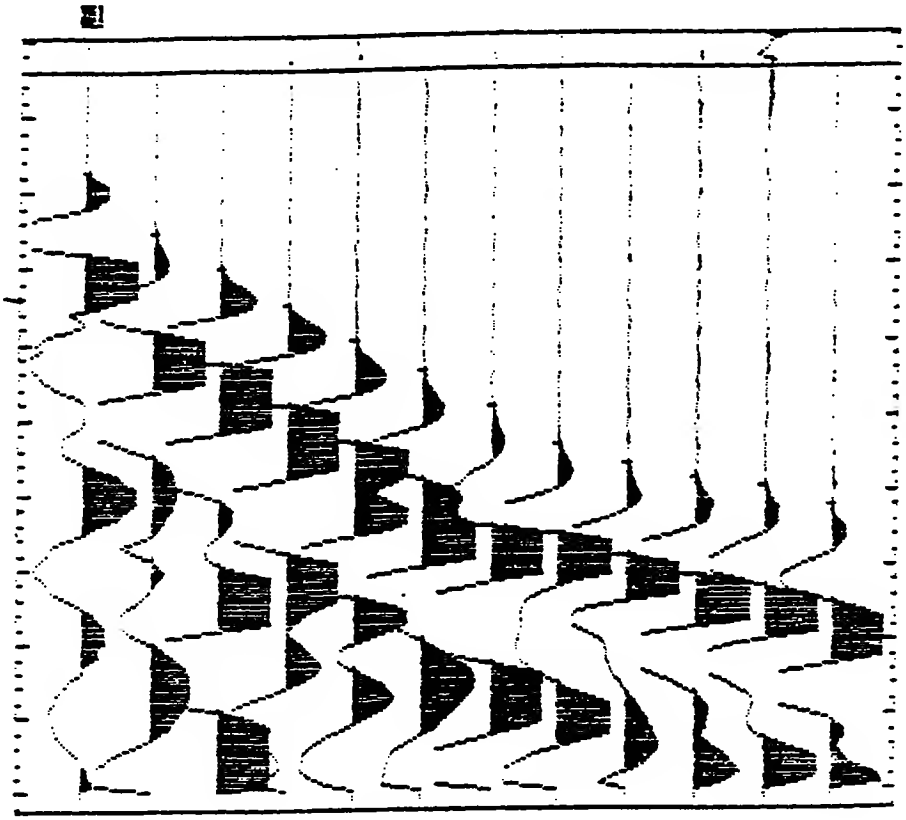
Fig. 1: Traces display



Listing of first breaks of profile

Area	: BERNA	Profile	:	15	Record	:	874
Shot pos	: 3.0	Lay start	:	4.0	Lay end	:	15.0
Sampling	: 100	Delay	:	0.00			

Trace	Position	First break	Trace	Position	First break
1	4.00	17.30	2	4.50	
3	5.00	25.40	4	5.50	
5	6.00	30.20	6	6.50	
7	7.00	34.90	8	7.50	
9	8.00	39.70	10	8.50	
11	9.00	43.80	12	9.50	
13	10.00	48.60	14	10.50	
15	11.00	53.70	16	11.50	
17	12.00	56.30	18	12.50	
19	13.00	58.10	20	13.50	
21	14.00	59.30	22	14.50	
23	15.00	61.80	24	15.50	

In ***reflection seismics***, all reflected waves are considered. A shallow reflection seismics allows the interpretation of complex geological situation on the subsurface for a few years.

In shallow reflection seismic, acquisition of raw data and its processing must be treated with great care.

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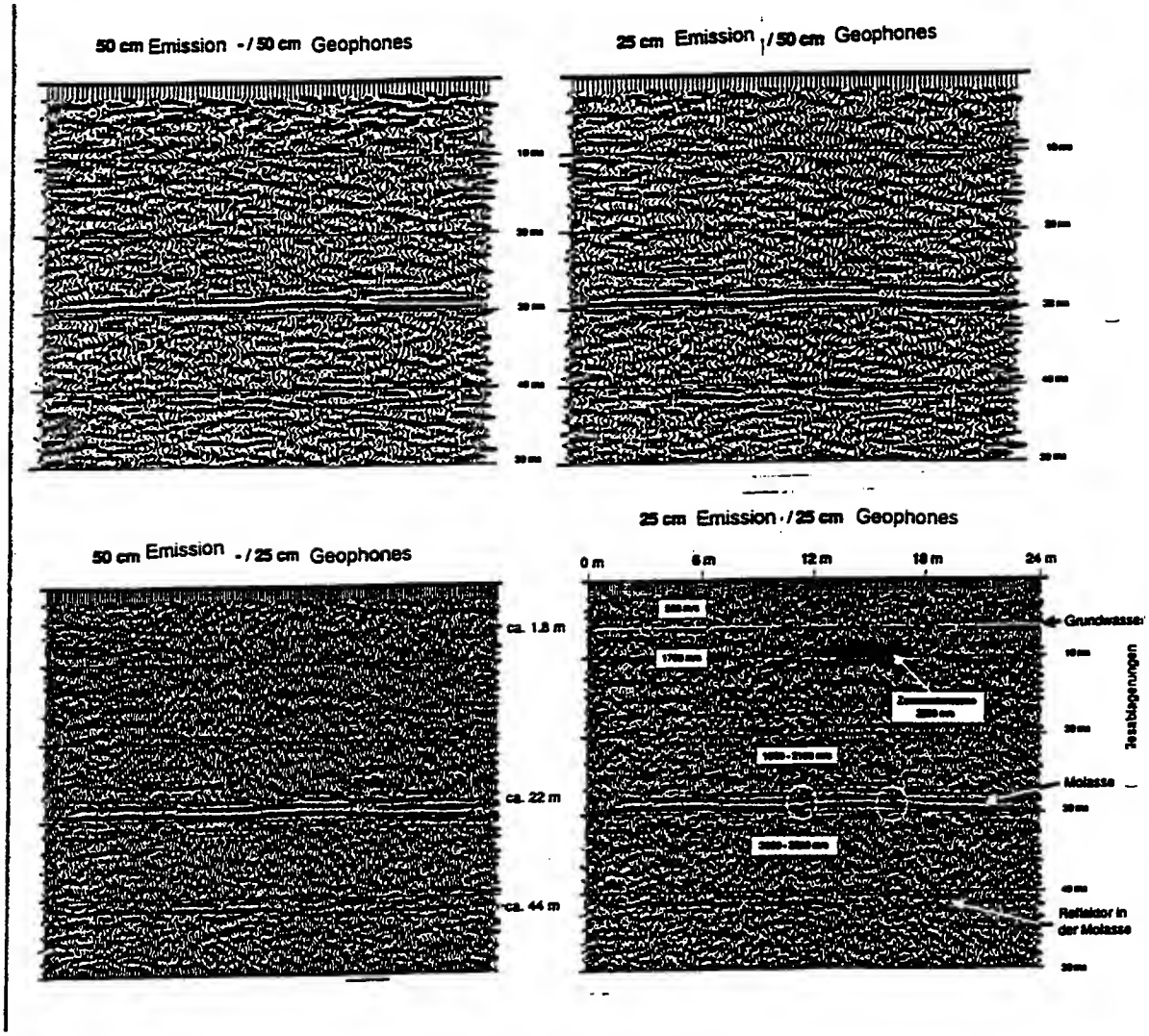
In the acquisition of raw data, the distance between the points of emission and the distance between the geophones are very important . These distances influence the data density which can be defined by the two following parameters:

- Coverage: how often a point on a reflector is reached by a roll along acquisition
- Common depth point (CDP) corresponds to half the distance between the geophone

An example of the importance of the raw data acquisition is given in Fig 2. The effect of the variation of the data density can be clearly seen.

Fig 2: Results of a shallow reflection seismic for different distance of emission and geophones array.



For shallow seismic surveys, the consistent use of both reflection and refraction profiling is recommended. Although the reflection method is unrivaled for imaging the more complex geological structures, usually better velocities of the subsurface layers are obtained by refraction analysis.

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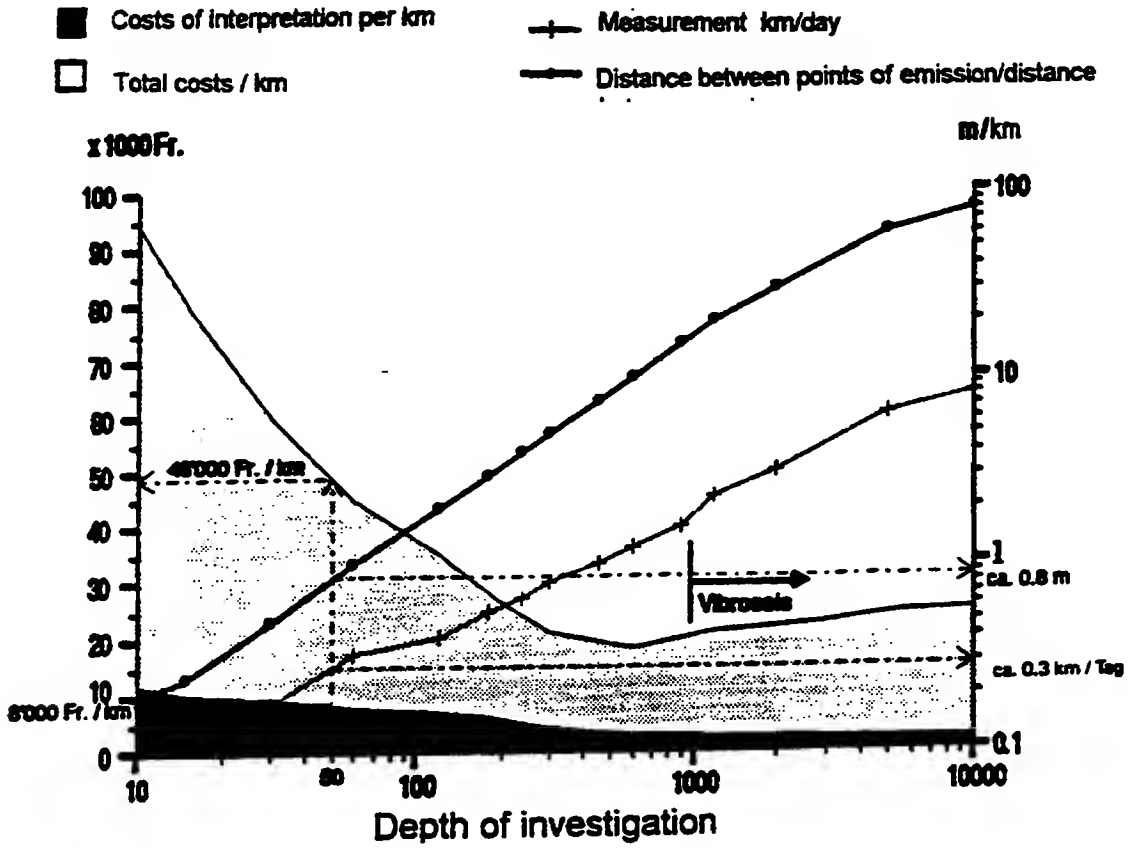
The table 2 provides a comparison of the two methods:

Table 2

	Refraction seismics	Reflection seismics
velocity determination	very good	good
thickness	very good	very good
geometry	weak	very good
complex geological situation	bad	very good
price	cheap	expensive

An estimate of the price of the shallow reflection seismic is given in Fig. 3 in relation to the depth of investigation.

Fig. 3: Estimate of the cost of the shallow reflection seismic.



3.3 Gravimetry

This method is based on the difference of density. After applying various corrections to the raw data, gravity anomalies can be mapped. The gravimetric maps show the distribution of heavy and light masses in the subsurface. It is also possible to discern a dense limestone from relatively lightweight clay in a karst system. This method is seldom applied in the investigations of raw materials for cement.

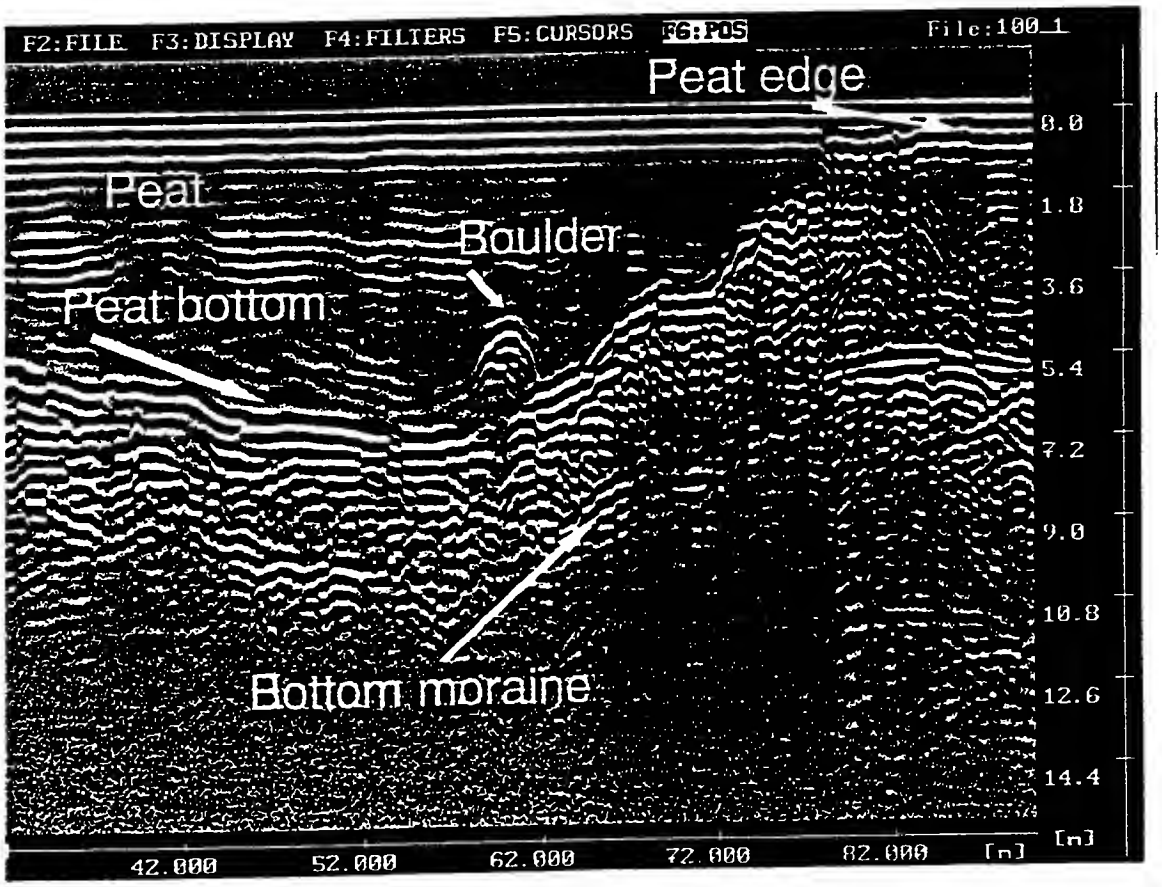
3.4 Geomagnetics

Geomagnetic method is the measure of the geomagnetic field on the surface. Iron-ore deposits are particularly suitable for investigation with this method, but it is less suitable for investigations of raw materials for cement.

3.5 Radar

Radar is a new method of prospecting, based on the electromagnetic waves. The tools consist of two antennae - transmitter and receiver. The transmitter generates electromagnetic impulses of 22 up to 100 MHz frequency. The depth of investigation depends on the frequency and of the mineral composition of the subsurface. An example of a radar is shown in Fig 4.

Fig. 4: Results of a radar investigation in a peat deposit



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3.6 Bore hole logging

Nearly all geophysical parameters can be measured in drill holes in order to define the nature and the quality of the rocks. This is especially convenient when no core samples are available (reverse circulating drilling), or when core recovery is very poor. Nowadays slimhole logging offers extraordinary flexibility thanks to the fact that probe series containing up to 7 probes can be assembled as desired. Thereby, the number of runs needed is minimised and substantially reduces the field operation times.

A new logging tool is now available in the petroleum industry to determine the chemical composition of the rock in the drill hole. The geochemical logging tool measures concentrations of 12 elements (SiO_2, Al_2O_3, Fe_2O_3, CaO, SO_3, K_2O, TiO_2, Cl, Ga, H, Th, U) and provides data to calculate the MgO concentration of the formation. The accuracy of this tool must still be tested for use in the investigation of raw materials for cement.

3.7 Choice of method

The choice of method of application depends on the contrast of the physical parameters between the different rocks. The charts (Fig 5) show the expected physical value for different rocks and sediments.

- geoelectrics: parameter is resistivity
- seismics: parameter is velocity

For instance, to determine the contact aggregates - sandstones, the best method would be seismics because of the better contrast of velocity between aggregates and sandstones.

In conclusion, geophysics is an indirect method of investigation. The results must be interpreted in geological terms. The most important advantage of the geophysics is the quick execution at lower costs by keeping a low profile in the field. The choice of the method is dependent on the physical parameters between the layers under investigation.

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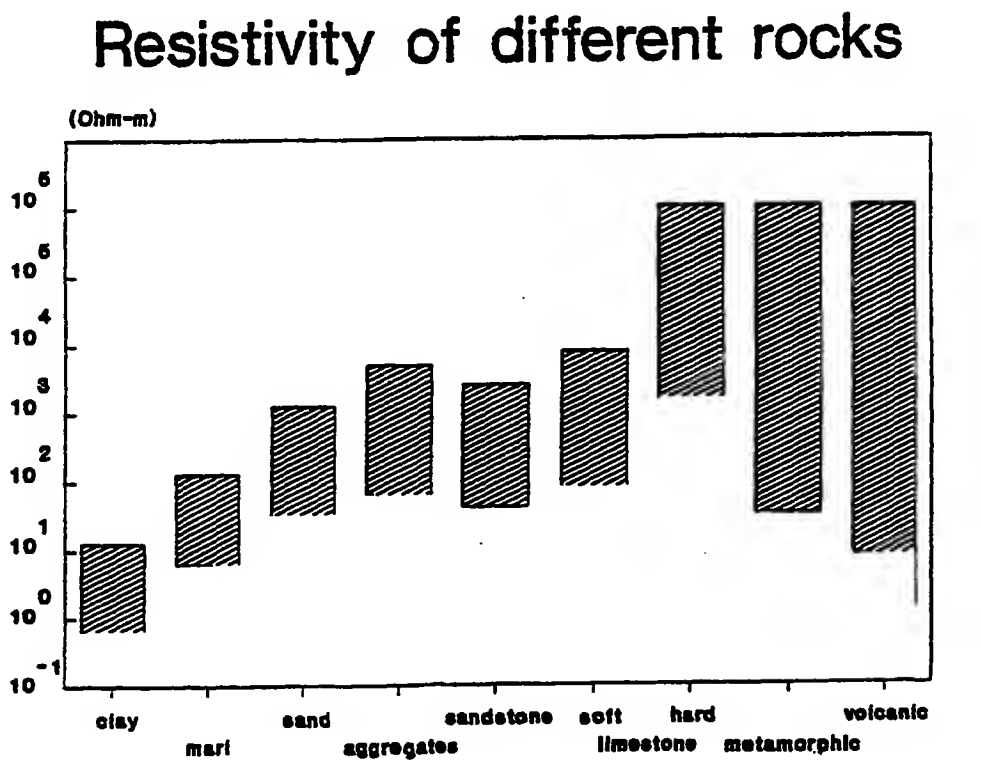
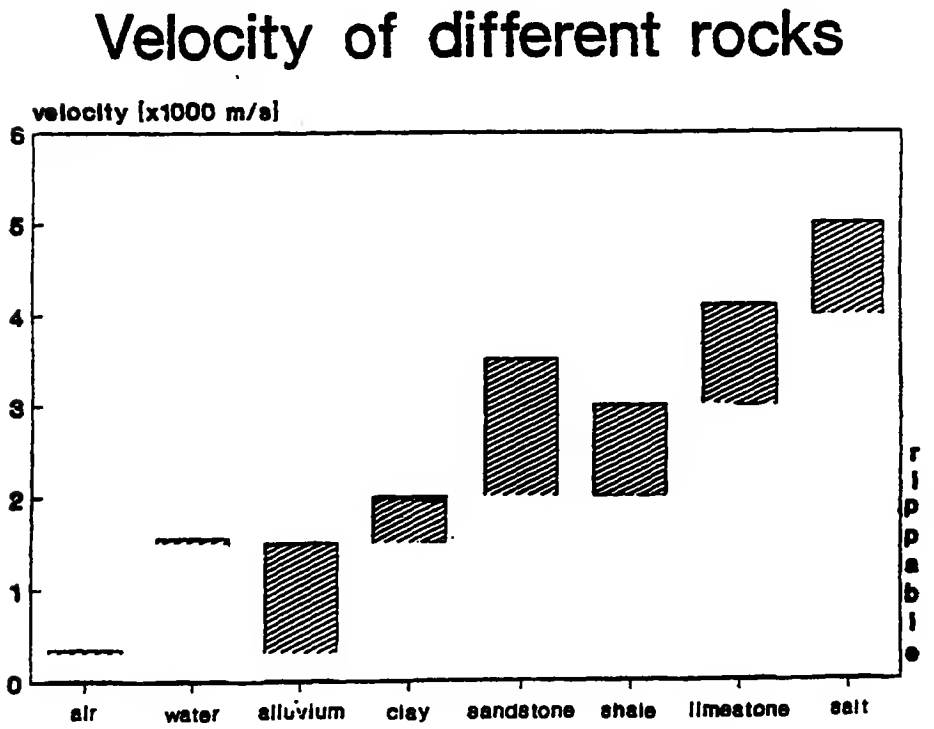
Fig. 5 **Physical parameters**





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4. CASE STUDY

4.1 Determination of the contact basalt - granite

Geology

In a phase of extension of the aggregates quarry, the reserves of basalt must be accurately defined. The deposits consist of the following lithologies:

- overburden: fluvial sand and moraine
- basalt with some intercalation of tuff
- granite

The top of the granite is weathered and consists of argillaceous material.

Method

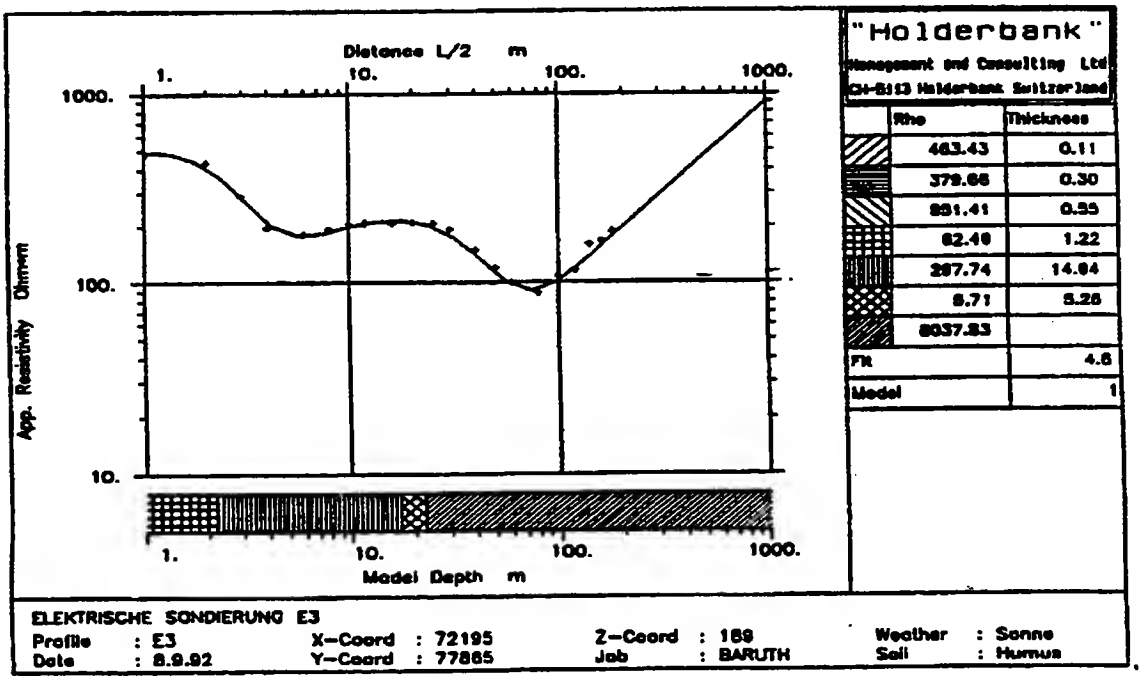
The weathered granite underneath the basalt shows a very good geoelectrical contrast. Indeed the argillaceous material has a very low resistivity, and can be easily detected by the geoelectrical methods.

Results of the geoelectrical prospecting

8 vertical electrical soundings and four poles profiling were carried out.

In the vertical soundings, the low resistivity layer under the basalt is clearly identified (Fig. 6).

Fig. 6: Interpretation of a vertical electrical sounding



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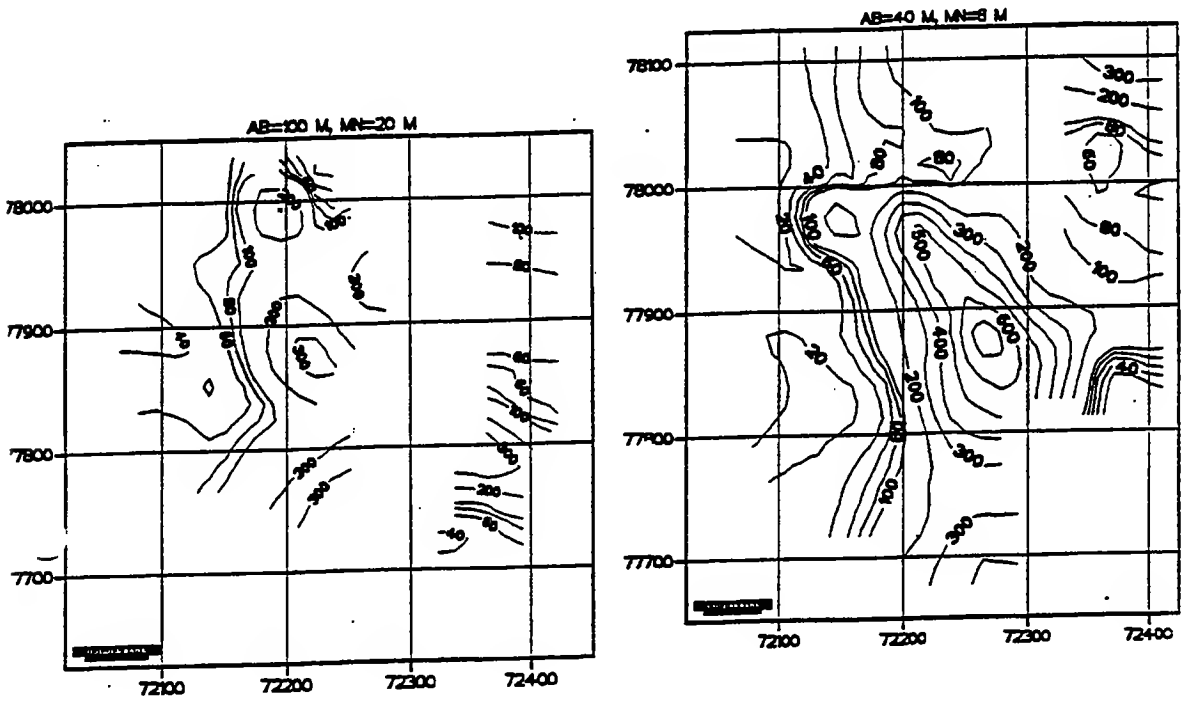
The apparent resistivity maps in Fig. 7 show the apparent resistivity for two different depths of investigation. The interpretation of these maps gives the geologist an indication of the structure of the subsurface for the investigated depth.

Fig. 7 Apparent resistivity maps



The result of this investigation is shown in the contour line map of the granite (Fig 8)

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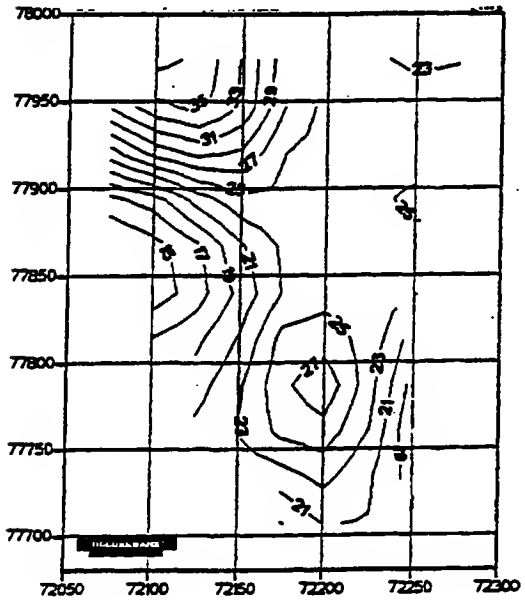
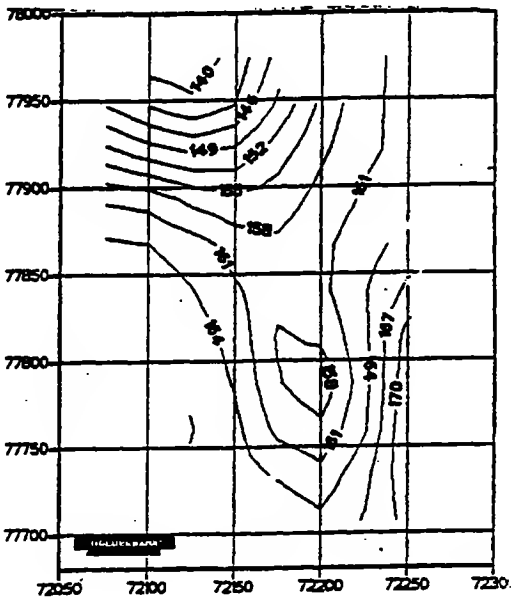
Fig. 8 Thickness of the basalt and contour line map of the top of the granite

Thickness of the basalt



Contour line of the top of the granite



4.2 Determination of the thickness of the lahar layers and their qualities

Geology

The field under consideration consists of several layers of lahar. A lahar consists of pyroclastic (volcanic) materials on the flank of a volcano. The deposit is produced by a landslide or mudflow. The quality of the lahar fluctuates due to the varying size of the matrix (coarse to fine grained). In this region, the lahar is overlaid by a volcanic tuff

Method

With vertical electrical soundings, the quality of the lahar can be appraised according to the fine materials in the matrix or the quantity of stone/gravel. Indeed, a fine matrix lahar or a sand have a true resistivity less than a sand matrix lahar containing many stones and gravel. The difficulty is to differentiate a fine matrix lahar from a sand layer, because it could have the same true resistivity.

The overburden (tuff) can be easily determined by geoelectrics due to the anticipated low resistivity.

Results

According to the results of the vertical soundings, three types of lahars have been determined. The true resistivity of the different lahars is as follows:

- Lahar 1: 600 to 1300 Ohm*m. Is considered to be of good quality. The highest resistivity indicates a consolidated lahar
- Lahar 2: 150 to 500 Ohm *m. Could correspond to a lahar with a fine matrix, or a sand with stones and gravel.
- Lahar 3: 90 to 150 Ohm*m, corresponding to a sandy - silty layer

The overburden (tuff) has a true resistivity in the range of 10 to 30 Ohm*m.

On the basis of the VES, a geological cross section is carried out (Fig 9).

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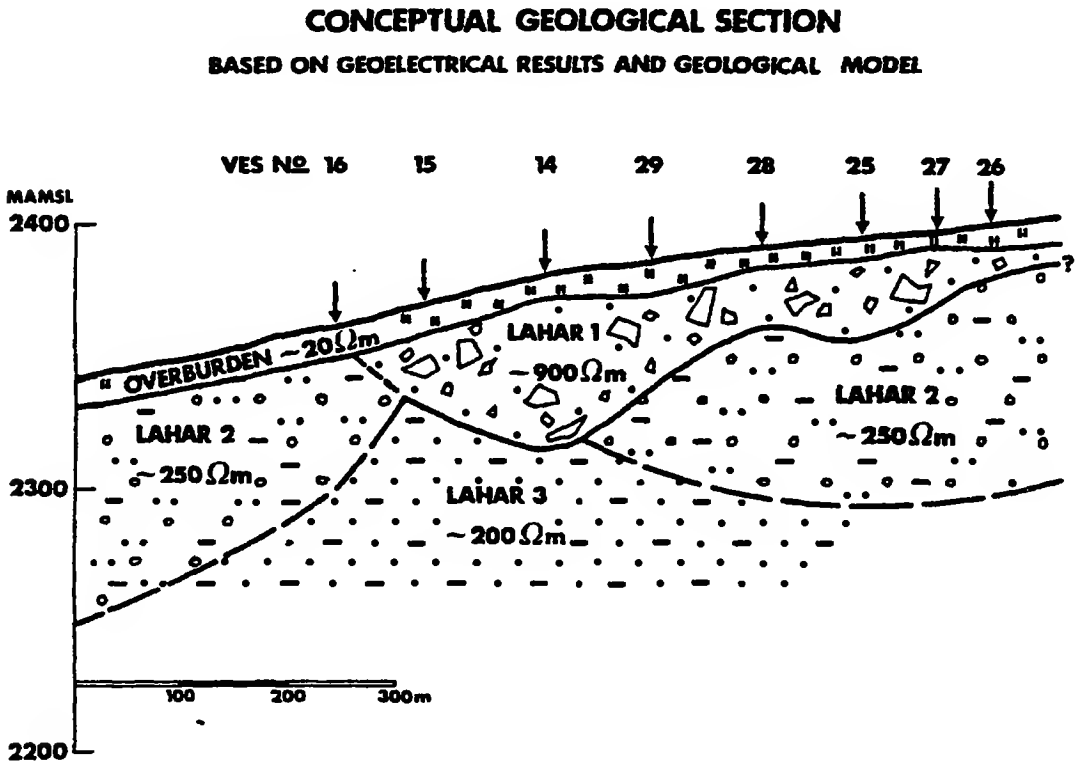
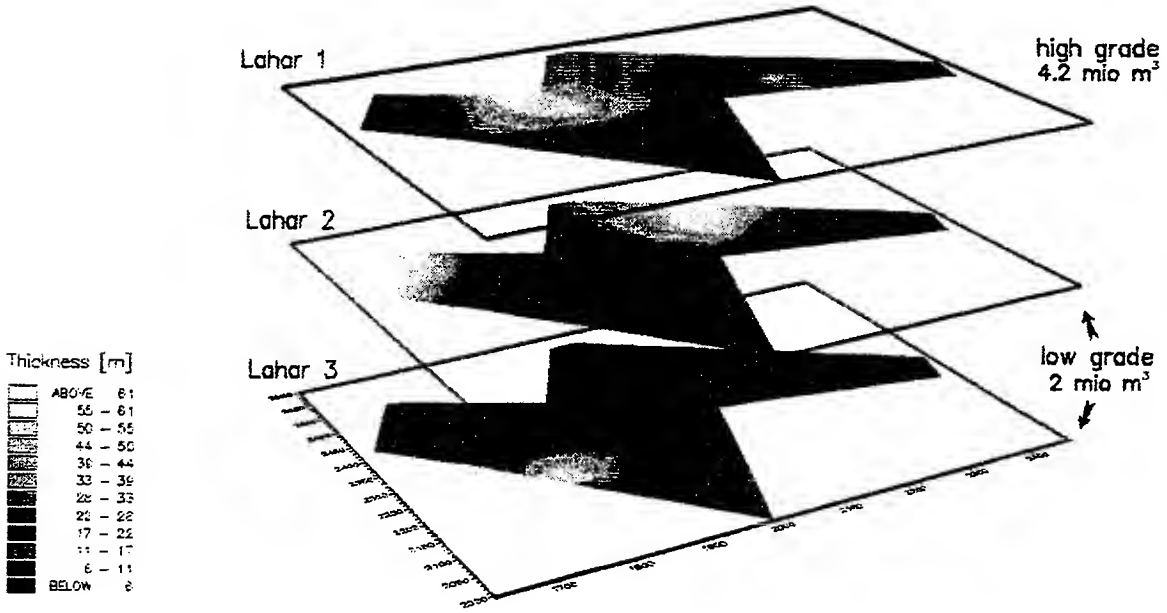
Fig. 9: Conceptual geological section based on geoelectrical results.



The results are shown in isopach maps, maps representing the thickness (Fig 10).

It was recommended to put down a few drill holes of calibration.

Fig. 10: Thickness maps of the lahar layers



4.3 Determination of the overburden thickness

Geology

The region under consideration consists of marl. In a tropical climate, the upper part of the marl is completely weathered in kaolinite. The thickness of the weathering layer has to be determined.

Method

The application of the refraction seismics in this situation has been considered because of the difference of velocity between the argillaceous weathered layer and the marl.

Results

With the refraction seismics, three layers have been determined. A profile is shown in Fig. 11. The considerable thickness of the weathered zone was prohibitive for exploitation.

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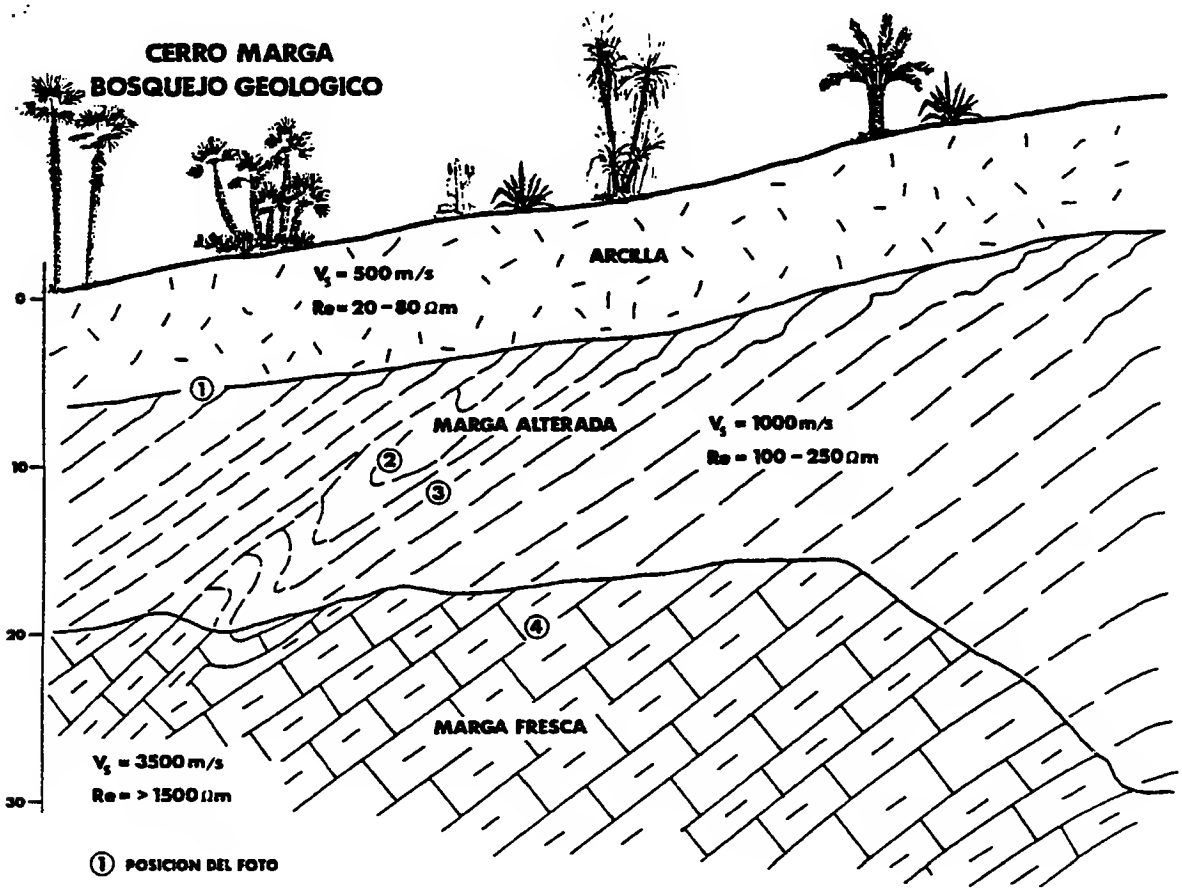
Fig. 11: Geological cross-section based on the results of refraction seismic



4.4 Determination of the geological structure

The purpose of this investigation was to determine the structure of the bed rock under an overburden. The overburden consists of moraine and landslide materials.

Method

Reflection seismics are applied accurately to determine the geometry of the bedrock

Results

The results of this investigation are shown in Fig 12.

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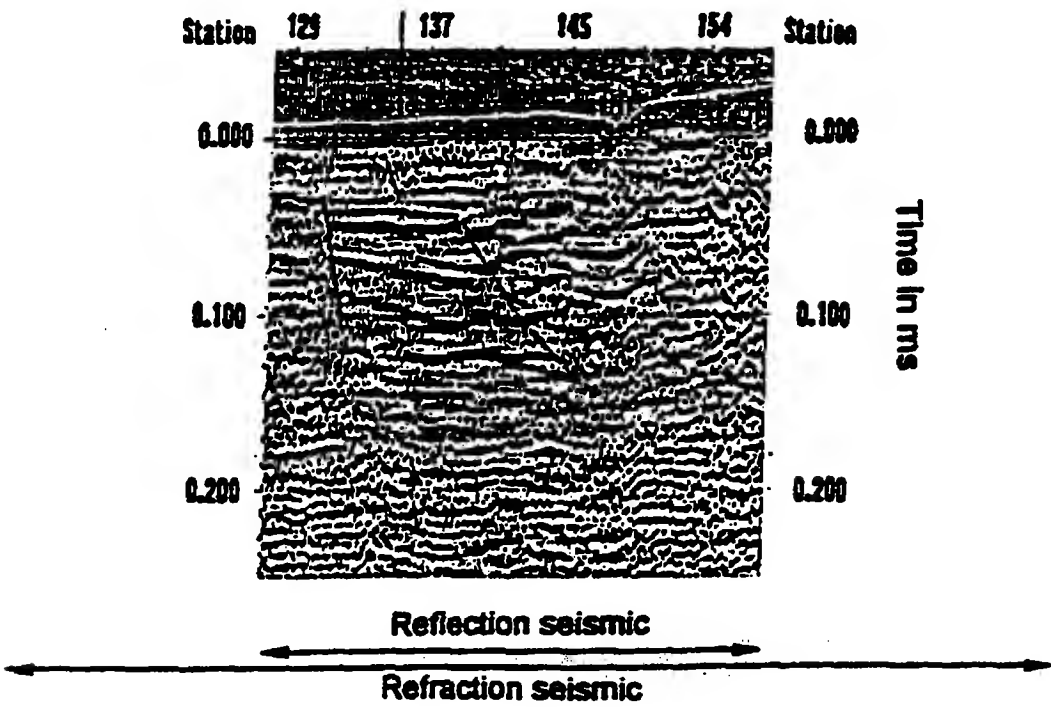
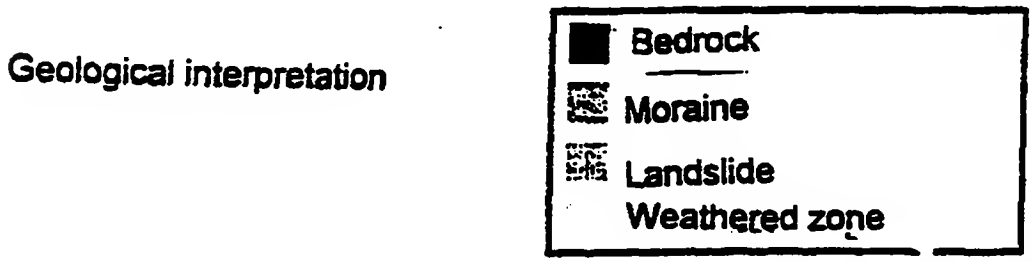
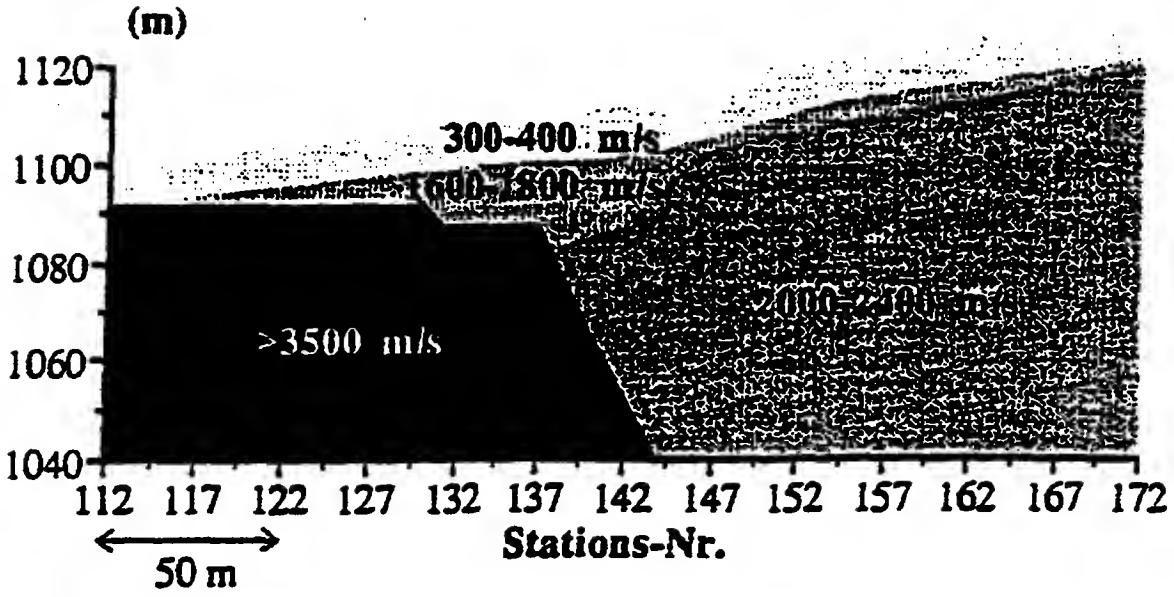
Fig. 12: Profiles of reflection seismic and geological interpretation







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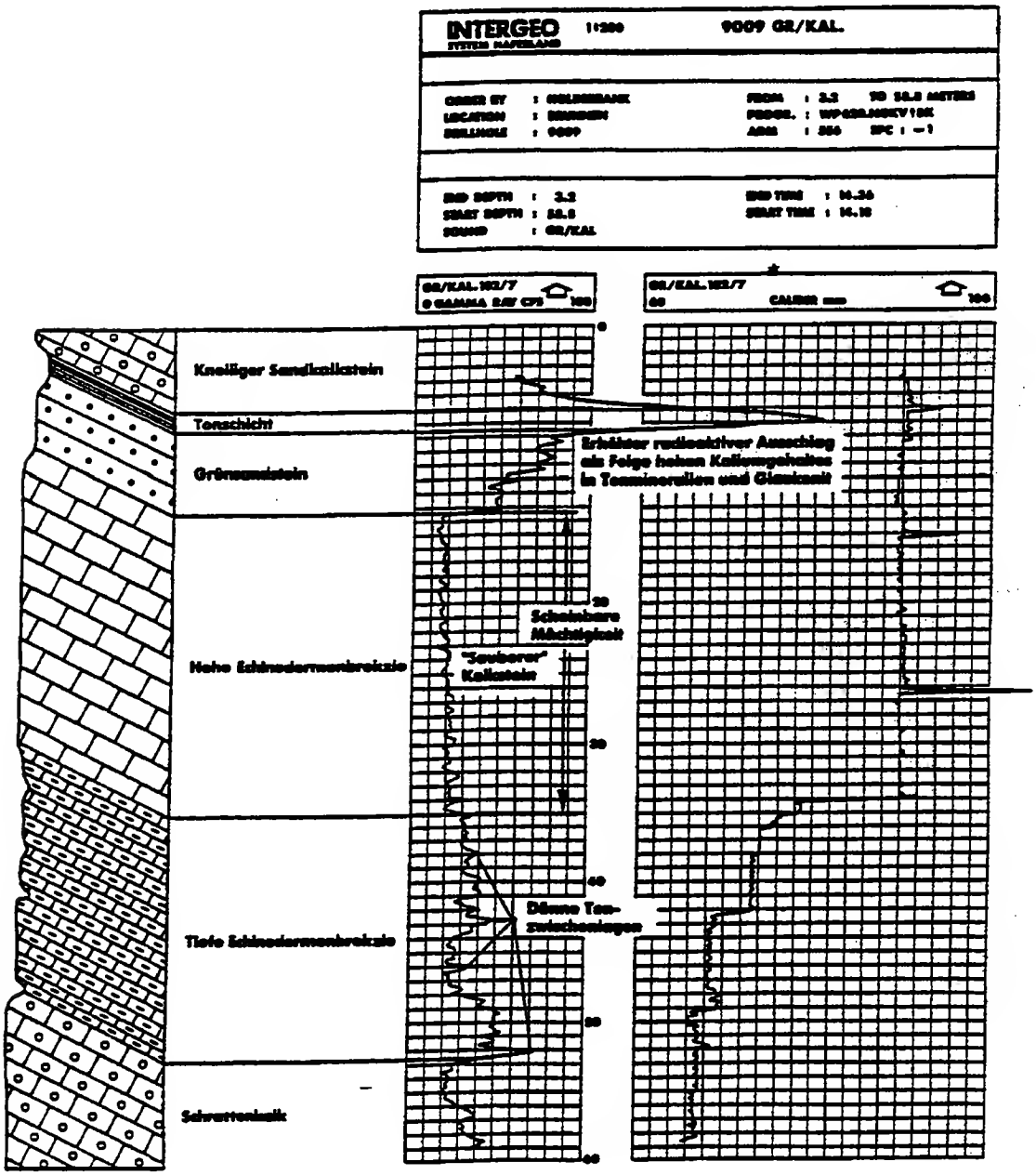
4.5 Evaluation of reverse drill hole

In the investigation of raw materials, the drilling campaign was carried out by reverse circulating drilling. No core was recovered. In order to determine the lithological contact and to localise thins interbeds of clay, a natural gamma ray logging was carried out.

With the present situation, the localisation of the thin interbeds of clay was indispensable for geotechnical purposes. The bench stability was precarious.

The results of the gamma ray logging is shown in Fig 13.

Fig. 13: Interpretation of gamma ray logging



The lithological contact as well as the thin interbeds of clay are clearly identified.

4.6 Karstified limestone

Recognition of cavities and determination of their size is always an important factor in geotechnics.

Preliminary drill holes had shown that the subsurface consists of strongly karstified limestone. Due to the lack of correlation between them, it was impossible to obtain a realistic image for the geotechnical calculation. A special logging tool was used by EBRA - Tee.

A radar tomography campaign has been carried out on each drill hole pair. Spacing between two drill holes ranges from 12 to 24 m.

Fig 14 shows the results of the tomography.

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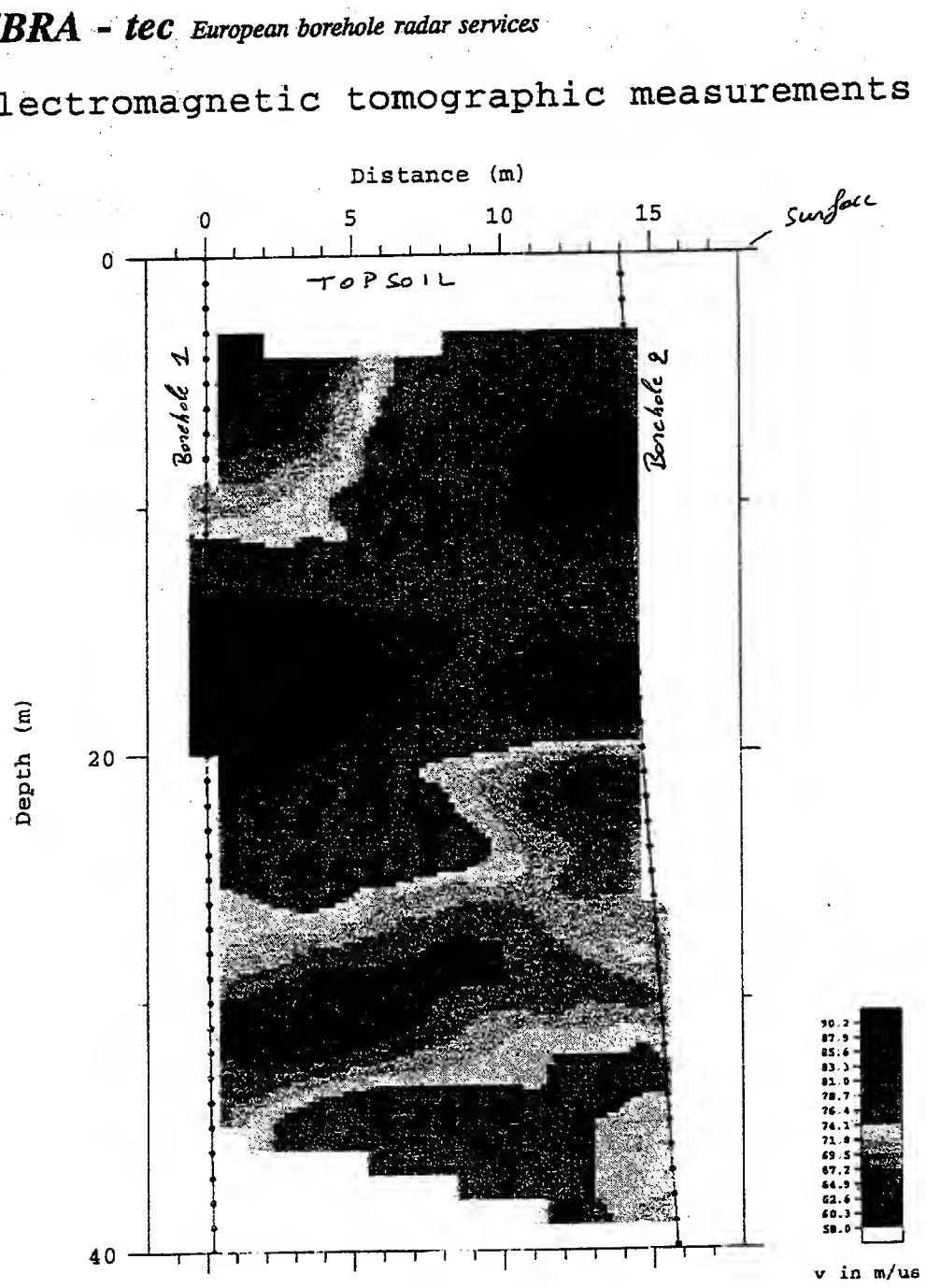
Fig. 14a: Radar tomographic measurements

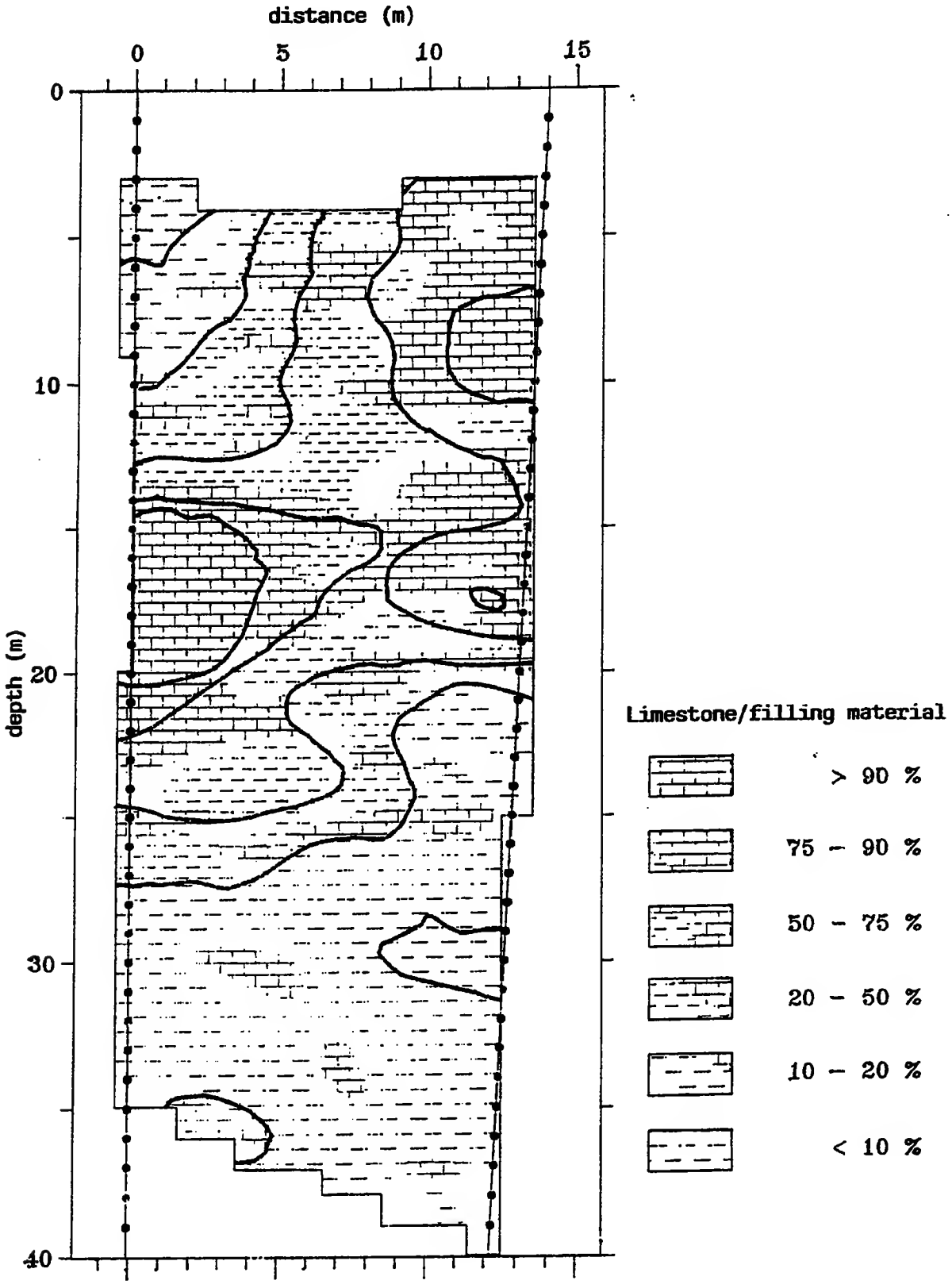


Electromagnetic tomographic measurements for the exploration of the foundation soil of a HST bridge in Belgium

Radar velocity tomogram

Center frequency: 22 MHz

Fig. 14b: Geological interpretation



Geological interpretation of radar velocity tomogram

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Chapter 5

Overall Raw Material Prospecting

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Overall Raw Materials Investigations

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Overall investigations of raw materials

1. INTRODUCTION

The overall raw materials exploration serves the purpose of acquiring sufficient data on a raw material deposit in order to establish a **feasibility study** for a project. It is important that the investigation program is well specified and that the responsibilities are clearly defined. This incorporates methods, responsibilities, information flow, cost controls, schedule and milestones.

The investigations of the quality and the quantity of 1 to 3 deposits (which were shortlisted in the preliminary field investigations) are appraised. The following work needs to be carried out in order to obtain accurate information about the deposits:

- Geological mapping and cross-sections.
  - Purpose : Distribution of the lithology / formations and geological structure
- Drilling :
  - 1'000 to 2'000 m of core drilling for the main limestone deposit
  - 500 to 1'000 m for the secondary deposits.
  - Purpose : Description of the lithology, sedimentology, structure, moisture content, porosity, density, compressive strength, sampling for chemical analyses
- Chemical analyses :
  - LOI, SiO_2, Al_2O_3, Fe_2O_3, CaO, MgO, SO_3, K_2O,Na_2O,TiO_2, P_2O_5, Mn_2O_3, Cl^-
- Calculation of reserves
- Geophysics if necessary
- Raw mix design
- Preliminary concept of quarrying
- Bulk samples for technological tests

The results of these investigations must be sufficiently precise in order to decide on the feasibility of the project with regard to the raw material aspects.

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2. DRILLING

During the overall exploration program, the deposit is investigated by drilling. In order to obtain the most reliable knowledge with respect to the chemical composition and the structure of the rocks, a comprehensive drilling programme is established. In this step, the drilling tool is generally a diamond core drill. This is the finest manner in which the lithology can be precisely described (defined), geological structures can be measured on the cores, and the core recovery of a drill hole can be accurately measured. It is very important to have a maximum core recovery, as the accuracy of the chemical character and the reserves determination of the deposit depends on core recovered.

2.1 Diamond core drill

These involve driving a fast rotating annular bit through the rock and so obtaining a solid undisturbed sample. The bits are either diamond impregnated or surface set. Occasionally other materials such as tungsten may be used. In terms of sample weight or lineal metreage this is at least three times as expensive as any other drilling methods for similar depths.

The drilling equipment is mounted on a frame or on a truck. This includes the drill, mast and water/air circulation pump. This frame may be mounted on skids to be towed or fitted with jack-legs (hydraulic rams), Alternatively it can be mounted on a truck or crawler track device as a permanent configuration.

2.1.1 Bits

The bits consist of a steel blank, to the lower end of which a metallic alloy matrix is sintered. This matrix is encrusted or impregnated with diamond grains (natural or synthetic) or occasionally with chips of tungsten carbide or silicon carbide as special alternative.

In soft formations such as clay, larger teeth or blades are needed. The cutting edge being made of tungsten or silicon carbide or polycrystalline diamond.

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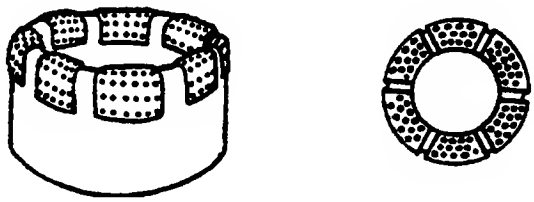
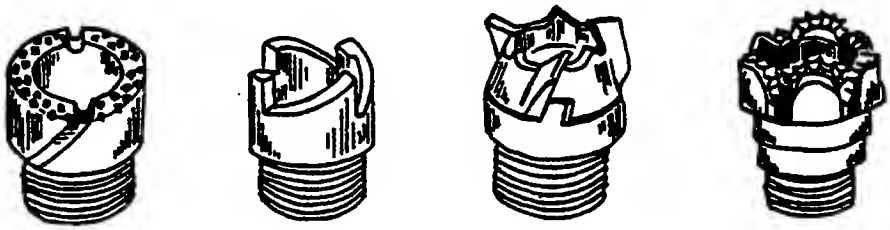
Fig. 1: Types of bits

Hard formations



Soft formations



2.1.2 Circulating fluids

The principal purpose of a circulating fluid is to remove the cuttings from the face of the bit as well as serve as a coolant to the bit. Water or an emulsion of water and special lubricants (barite, polymer oil) also known as drilling mud,is the common circulating fluid. The cuttings are washed up the borehole in the annular space between the drilling rods and the borehole wall or casing. Some drilling techniques (rotary and percussion) use air as the flushing medium. In special circumstances, air can be used to avoid dissolving minerals and losing the core, such as halite (NaCl) in the circulating water.

2.1.3 Core barrel

Core is collected at the bottom of the borehole in a core barrel and periodically. brought to the surface A conventional standard core barrel requires all the drilling rods to be removed from the borehole at the end of each run, before the core can be removed. In 1961, the wireline drilling system was introduced. This enables an innertube with the core (inner core barrel) to be withdrawn through the rods at the end of a cable or wire. This system obviates the need to withdraw all the drilling rods thereby saving time and money. Additionally the quality of the core and core recovery is improved when using wireline drilling.

A core barrel is that section of the drilling rods string in which the cylinder of rock is collected and can be brought to the surface. The simplest core barrel (single tube) consists of a reamer, a corelifter and the core barrel. The reamer serves two functions. One being to maintain the correct hole diameter to ensure sufficient clearance between the borehole wall and drilling rods, and the other, to provide seating for the corelifter. The corelifter consists of a tapered spring-steel ring, which allows the core to travel in one direction only (upwards). When the rods are lifted, the core is wedged in and clamps inside the core barrel.

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The corelifter not only prevents the cut core from dropping out, but also allows the core to be broken off at the bottom of the hole. The circulating fluid passes down through the center of the drilling rods past the collected core (in the core barrel) and out over the bit before returning to the surface on the ouside of the drilling rods.

In the double tube barrel, the drilling fluid is diverted away from the core to the bit by way of the annulus between the core barrel and the innertube. This system was devised for soft, broken or friable ground, where the core could be easily washed away by the driling fluid. In subsequent models, the innertube is fixed with a ballrace so that it cannot rotate with the rods thereby cause grinding and wearing of core. A further refinement is a split innertube to allow better transfer of undisturbed core into the trays.

In very soft, unconsolidated or porous rocks it is necessary to use a triple tube core barrel with split innertube to obtain a good recovery.

2.1.4 Recovery

The drilling should be carried out in such a manner that a maximum core recovery is ensured.

The minimum core recovery is 95% per drill hole. Below 80 %, the accuracy of the geological information derived and associated chemical analysis is doubtful.

According to Holderbank drilling specifications, the client has the right to reject the core and therefore the samples received from the drilling contractor if the core recovery is below 80%. Only if the insufficient core recovery is due to inadequate drilling. In which event, the contractor is obliged to redrill the borehole at no cost to the client. On the other hand the drilling contractor, cannot be held responsible if the geologist confirms that the poor core recovery is due to lithological features (e.g. cavities, caves, incompetent ground, etc).

Good core recovery is difficult to obtain from unconsolidated material for two reasons:

- large gravel pieces tend to either move about in the core bit and so prevent clean cutting or they jam in the bit
- fine sand, whilst feeding into the barrel, will not be rigid enough to be retained by the core lifter.

2.1.5 Core size

Core size depends on the bit and barrel combination used. In general for our purposes, the core size will have a minimum diameter of 45 mm (or NQ-3). In broken material, recovery will be better with a larger core size. In very hard material, recovery is invariably very good, at lower cost when using the smallest core which will be suitable for our purposes.

2.1.6 Core storage

The cores must be stored in wooden core boxes or trays, which are lockable. The core boxes must be adequately marked on the outside showing borehole co-ordinates (number), box number, depth of drilling (from... to) in meters. Generally, the inside length of the core box is 1.05m. whilst the width of the inside sections (usually 5 sections) should be +/- 5 mm larger than the core diameter. The inside of the core box must likewise, be adequately marked showing top and bottom depth of each section with an arrow pointing down hole. Every run of the core barrel must be 'flagged' using tags, labels or wooden blocks. Information on the flags indicates run number and depth. Pieces of wood of appropriate length must be inserted into the core boxes to indicate voids or cavities, as they are encountered in the actual borehole. The core boxes or trays, must be stored in suitable coe shed which is protected against atmospheric conditions, vandalism and competitor interference.

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2.1.7 Core sampling and preparation

The project geologist is responsible for samples selection of the cores according to acceptable geological practice. The sample widths are to be adequately marked on the cores and recorded in the borehole log.

Samples preparation procedure is important so that representative samples are supplied to the laboratory for analysis. The borehole cores are split (or divided) length-wise (from top to bottom) into two halves using a stone saw or a core splitter installed on site. One half is returned to the respective core box, and is adequately stored as the geological reference.

The other half is crushed and comminuted, sample for sample, according to the geologically defined intervals, to a grain size of max. 2 mm. The original sample of several kilograms is in a series of several steps of alternate crushing and quartering reduced to a final mass of about 200 grams.

The intermediate fractions not used for further processing will have to be adequately stored in labeled plastic bags.

The +/- 200 grams final sample material is ground with a laboratory mill to 0% residue on a 60 micron sieve (10'000 mesh per cm^2).

For both client and "Holderbank" one sample of approx. 50 g each should be prepared for chemical analysis.

2.1.8 Drilling problems

The main problems encountered during a drilling campaign are as follows:

Access

Sometimes, the access to the drillpoint is very difficult. Access roads have to be built or the drilling equipment will have to be dismantled and carried.

Recovery

A good recovery depends on the experience of the man in charge of the drilling. In case of poor or soft rock, which leads to bad recovery, a double or triple tube core barrel is recommended.

Others problems consist of loss of water, deviation of the drill holes, stabilility of the hole/casing, etc...

2.2 Other types of drilling

2.2.1 Auger drill

The principal component is a slowly rotating rod. The material from the hole is forced upwards by the wedging action of the cutting bit tips and by a spiral flue (archimides screw) fitted to the rods.

These rigs are mainly used for post hole digging and sampling of soil, clay and shallow weathered rock, in situations where disturbed samples are acceptable. These rigs are unsuitable in unconsolidated alluvial (pebble, sand, clay) or hard rocks.

2.2.2 Rotary drill

This method uses a slowly rotating rod string at the end of which a bit scrapes and dislodges rock chips by point pressure. The cuttings are flushed usually flushed by air although water or drilling mud (emulsion of barite and water) is used in special circumstances. Circulation may be normal, i.e. cuttings are exhausted from the borehole between the rod and the borehole hole wall or casing, or may be reversed (RC), whereby cuttings are exhausted through the interior of the rods.

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Reversed circulating drilling may be used for geochemical sampling. The price of the RC drilling is much cheaper than the core drilling, but geophysical logging must be carried out to for correlation between the drill holes. The total costs (RC drilling plus geophysics) are still good, compared to those of core drilling (30 to 60% lower).

Rotary drilling methods are frequently applied in oil well drilling.

2.2.3 Percussion drill

As the term suggests the principle thereof is having a slowly rotating chisel impacting the rock. The method of application of force to the chisel allows a classification as follows:

- uphole hammer drill
- downhole hammer drill.

Normally applied in production drilling (blasthole) in open pit or quarry operations.

3. GEOLOGICAL MAPS AND CROSS -SECTIONS

3.1 Geological mapping

A geological map is a representation of the surface outcrop rocks, very often superimposed on on a topographical map. Geological mapping is therefore essential to study the spatial distribution of the different rock types, to determine their thickness and to investigate the geological structure of the deposit. Many folds or faults, for instance, can be discovered or confirmed (from areial photography) by geological mapping. If seen in outcrop, these features must be mapped over a large area to be understood. Geologic maps are the basis on which one can construct the cross sections. The shape and structure of the deposit is therefore better understood through detailed geological mapping.

The scale of a detailed geological / topographic map is generally 1 : 1000 to 1 : 2000 with a contour line of 1 m.

Each formation encountered in the field is characterised by a specific colour or symbol on the map and their relative age is indicated in a legend. The legend customarily has a specific sequence. The oldest rock formation is listed at the bottom followed by the others in order of their decreasing age, ending with the youngest formation at the top of the legend (Fig. 2)

The geologist must characterise each exposure or layer and attribute them to a formation. The fossil content, sedimentary structures and their lithological aspects allow one to characterise them.

Moreover, the geological map shows the structures of the deposit (Fig 3). The dip and the strike of the layers, as well as the faults and the folds are drawn on the map. Fig. 4 shows the conventional symbols.

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Fig. 2: **Statigraphical column with the thickness of the formations**

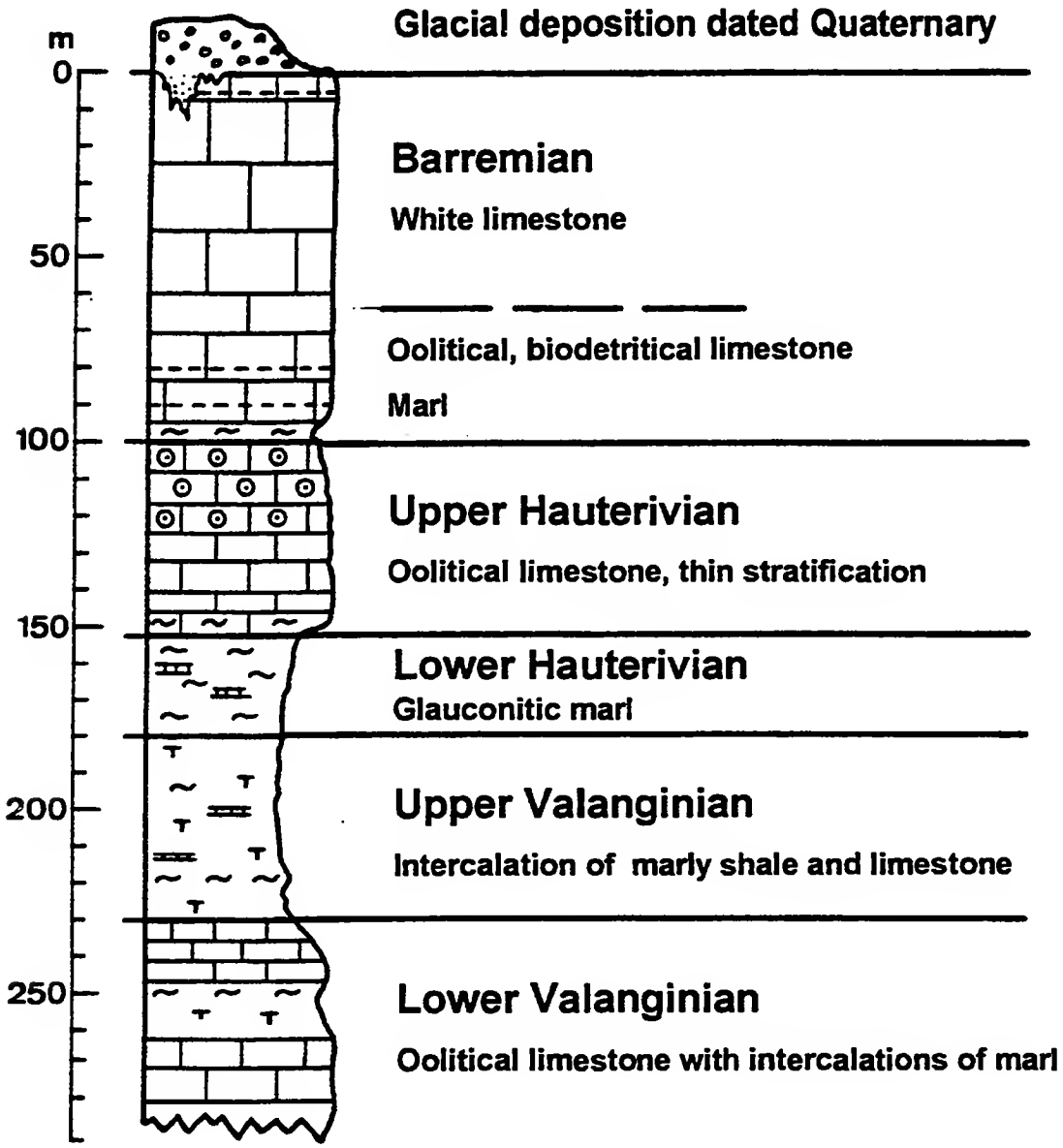

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Fig. 3: Geological map, scale 1: 2000

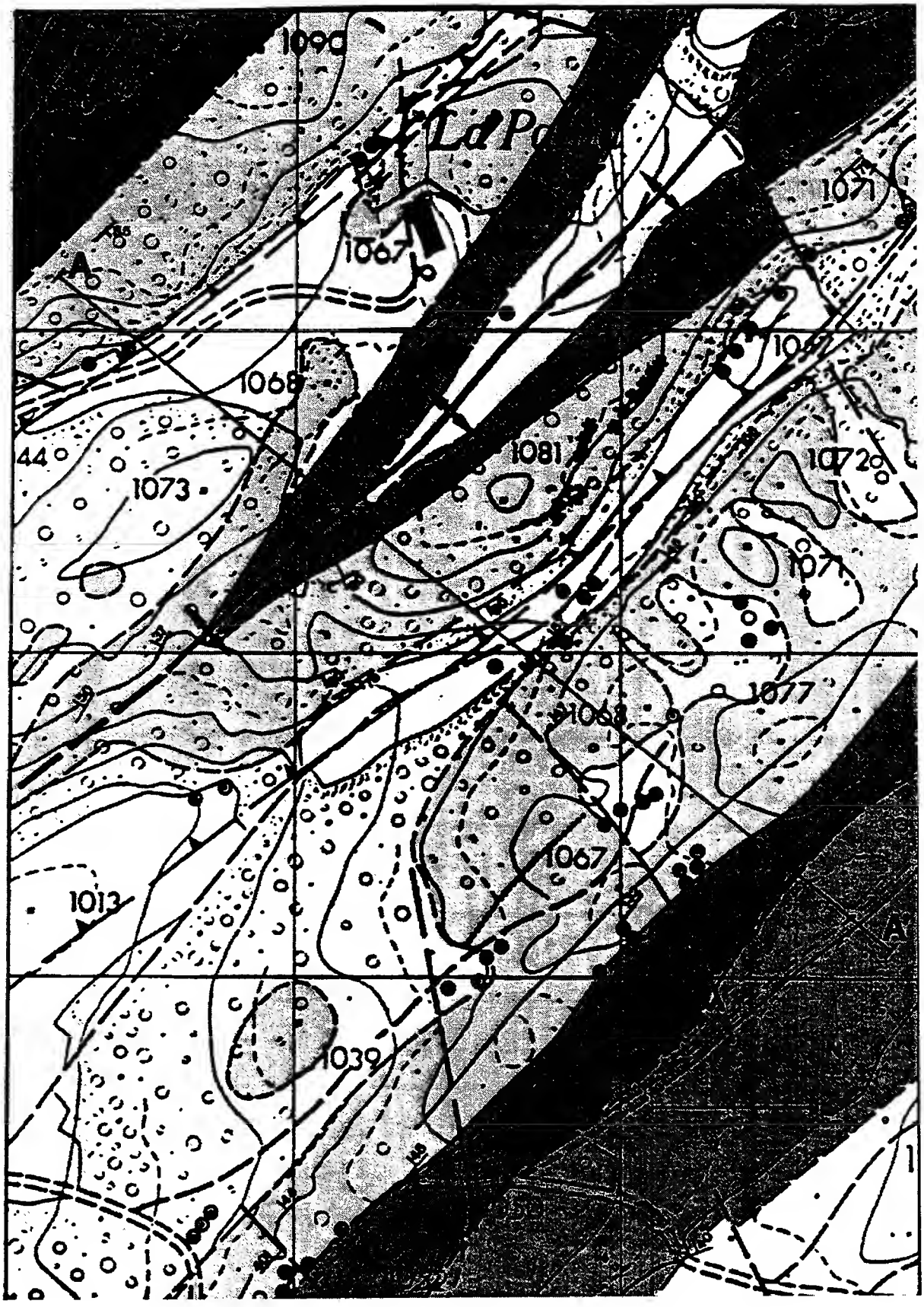

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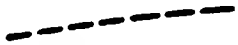
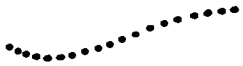
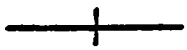
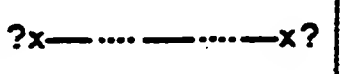
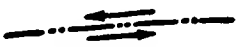
Fig. 4: Mapping symbols

Contacts of one formation to another		contact
		contact showing dip
		contact approximate position
		indefinite contact
		contact concealed
Bedding		bedding, strike and dip
		horizontal bedding
		vertical bedding
Faults (in red normally to distinguish from contacts)		fault
		fault approximate
		relative lateral movement
		thrust
Folds		trace of axial plane and plunge of axis

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Considering the discontinuity of the exposures due to the vegetation, overburden, alluvial deposits, etc. geological mapping is not easy and depends on the experience of the geologist. The geologist must use the knowledge available with regard to stratigraphy and tectonism of the region, in order to create a geological 'model'. This 'model' will be continuously compared with the field observations to get a more realistic image. This method of fieldwork allows one to obtain reliable knowledge very rapidly over a large area in spite of discontinuous exposures.

3.2 Geological cross-sections

Geological cross-sections show a section in a vertical plan (Fig. 5). They show a postulated distribution of the rocks at depth. Based on the geological situation observed at the surface (geological map) as well as information derived from drilling and geophysical surveys.

The geologist's structural interpretation can supported by the results of the drilling campaign. Correlation of the geological map and the results of the drillings give a reliable image of the sub-surface geological conditions. Refer Fig 6.

The significance of the geological interpretation of the deposit structures for the calculation of reserves is illustrated in Fig. 7.

Although the height of the outcrops of limestone is almost identical, the exploitable quantities are significantly different. Horizontally layered deposits (Fig. 7a) contain considerably larger quantities of exploitable materials than a deposit of inclined stratification. This is obvious for cases of where the strata dip outwards (Fig. 7b). Equally important also important in the opposite situation where the starta dip inwards(Fig 7c), because of increasing quantities of overburden to be removed.

Fig. 5: Geological cross-section, scale 1 : 2000

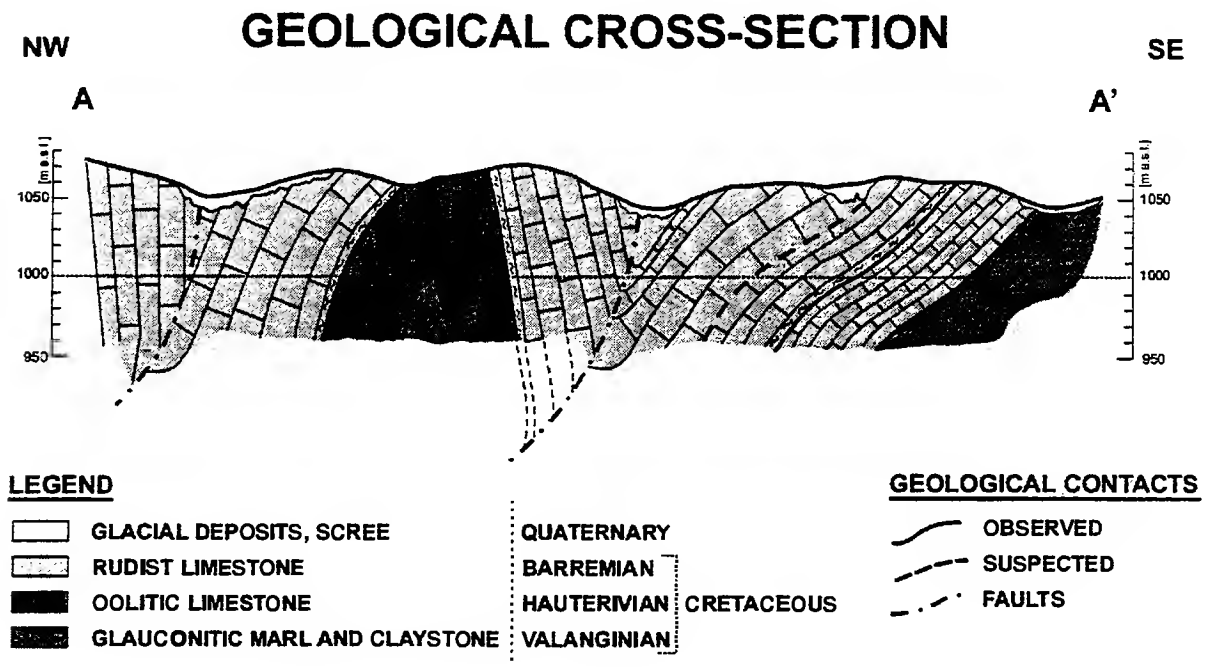

Fig. 6: Correlation between cross-section and drill hole data

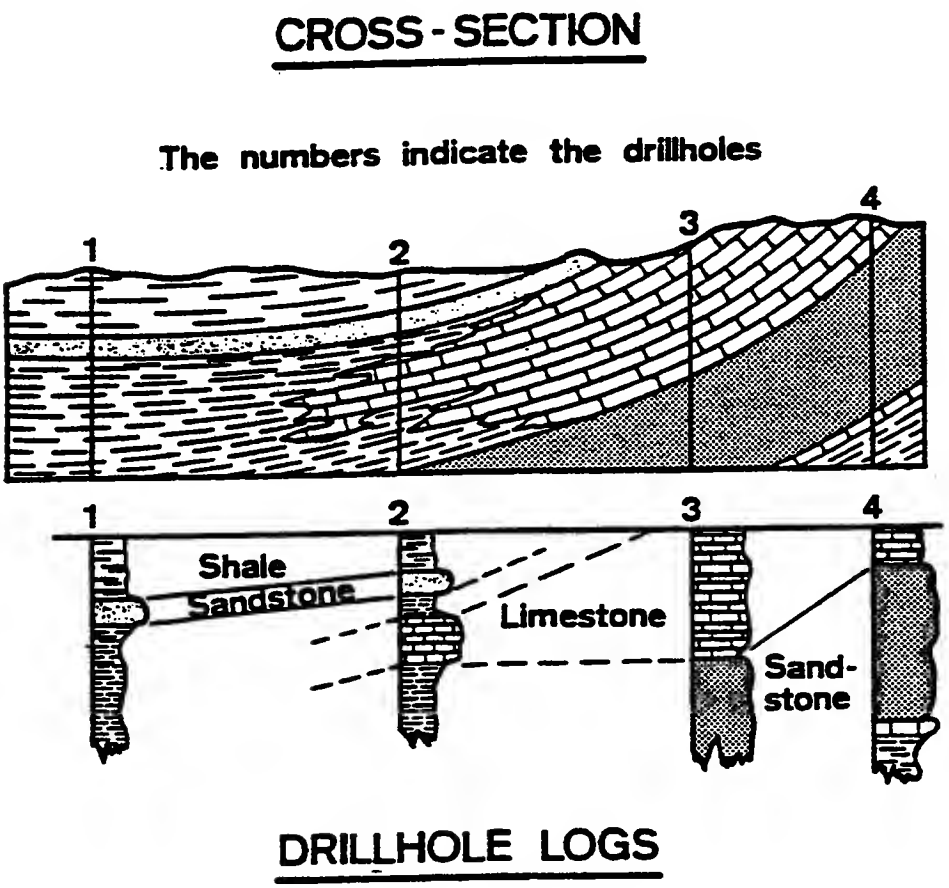

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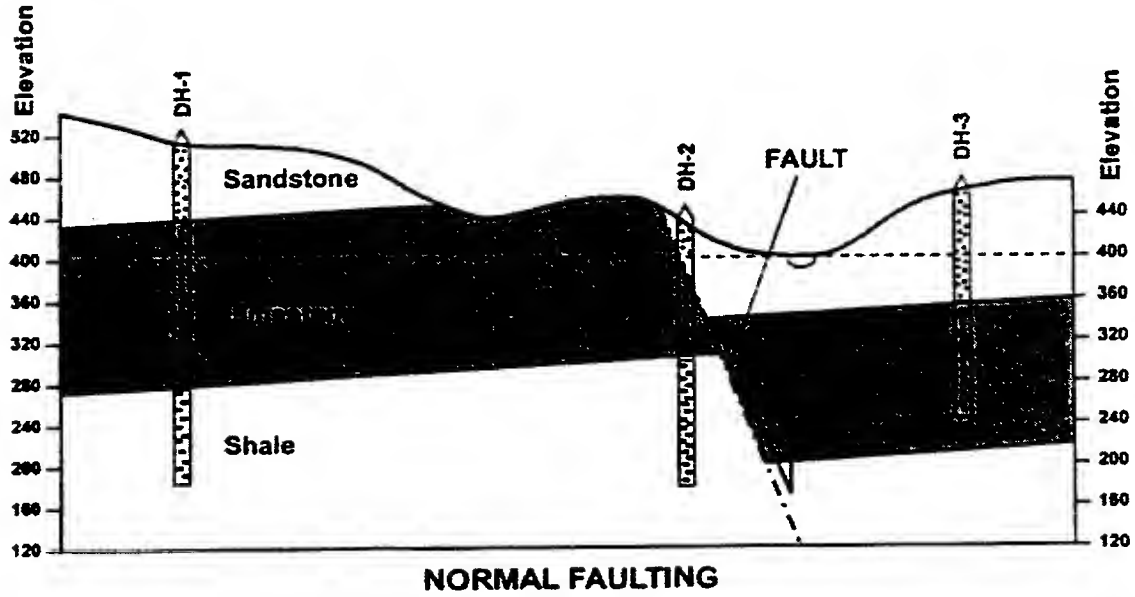
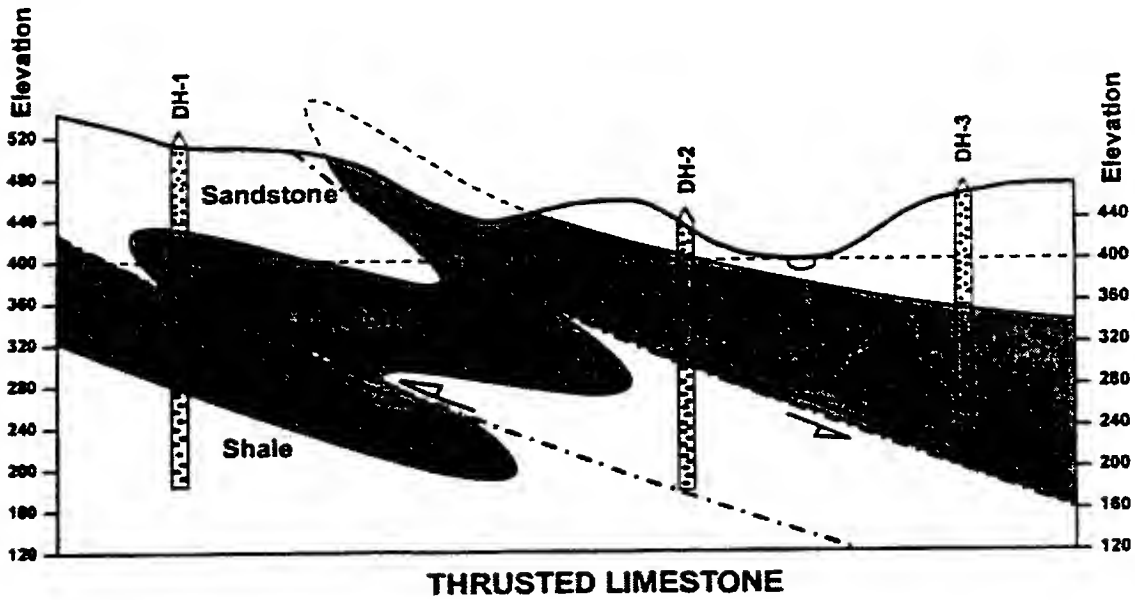
Fig. 6a Limestone Correlation - Alternative Solutions





Another aspect to consider is the intercalation of soft rocks (shale) between a formation consisting of hard rock (limestone) (Fig.8).

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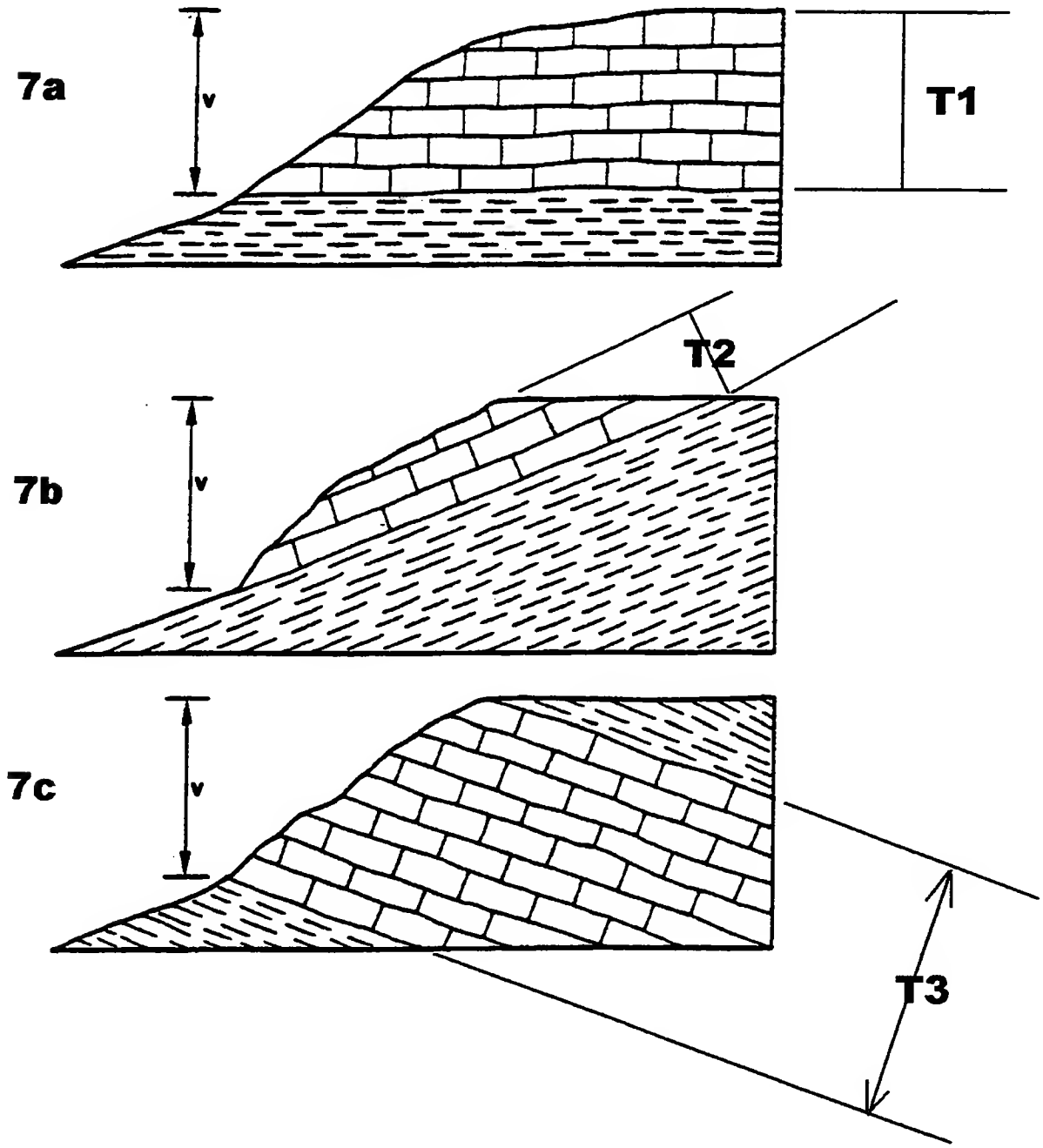
Fig. 7 Significance of the dip of strata for the reserve calculation



V - vertical height ≠ true thickness

T1 = V

T1 = 2 * T2

T3 = 1.5 * T1

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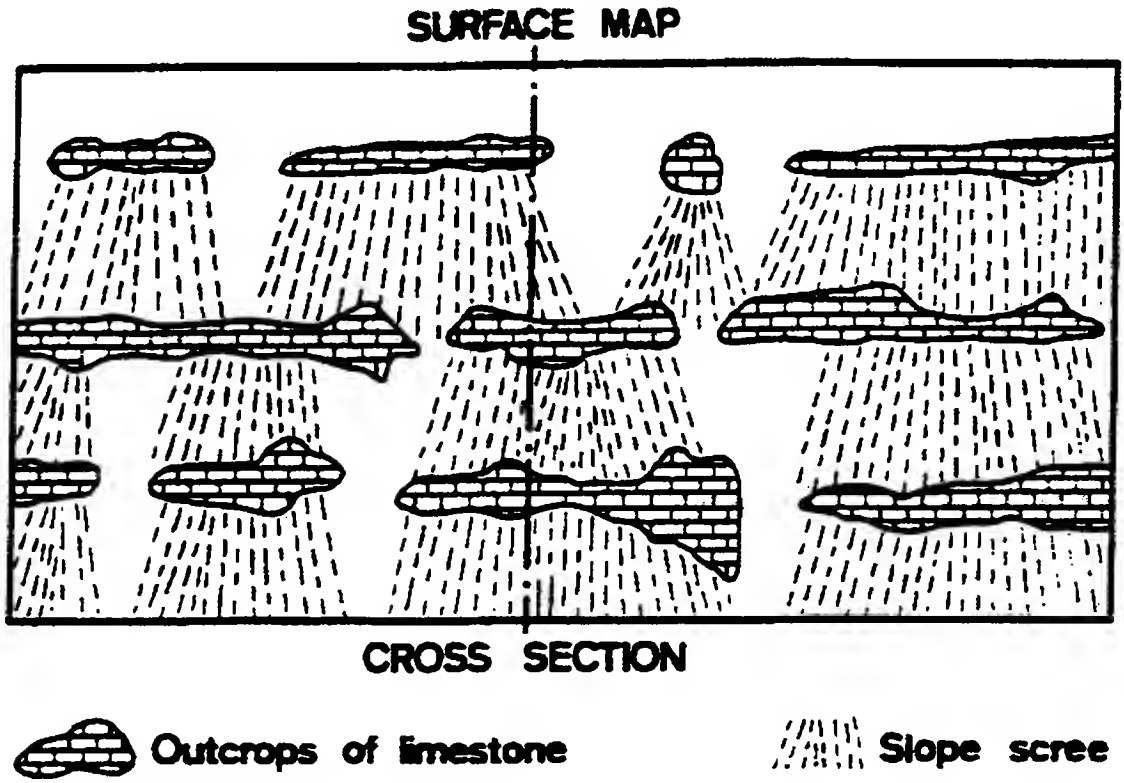
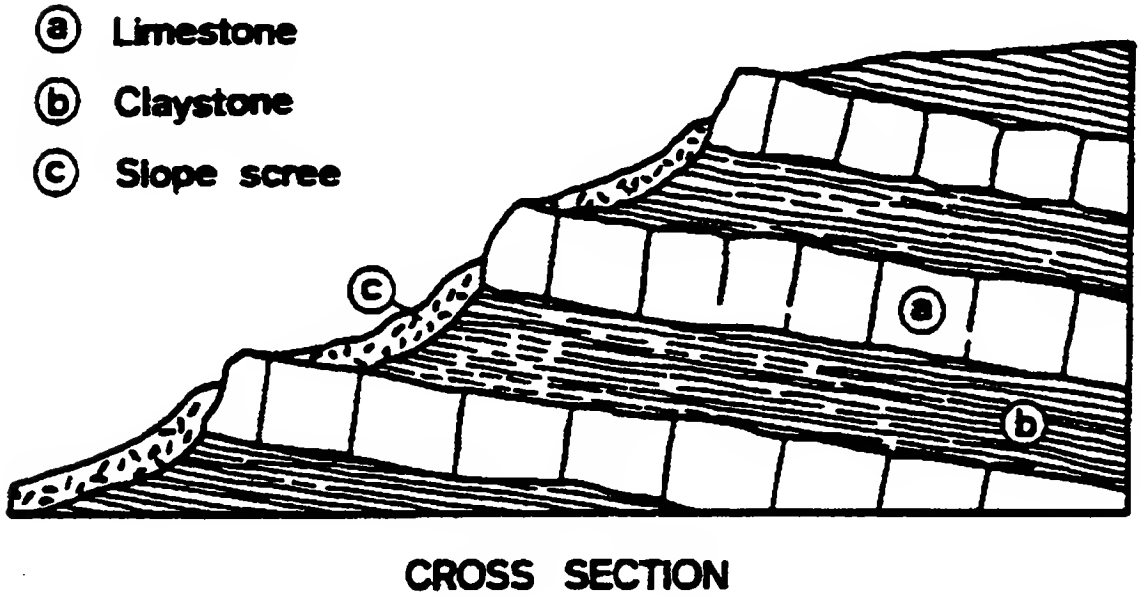
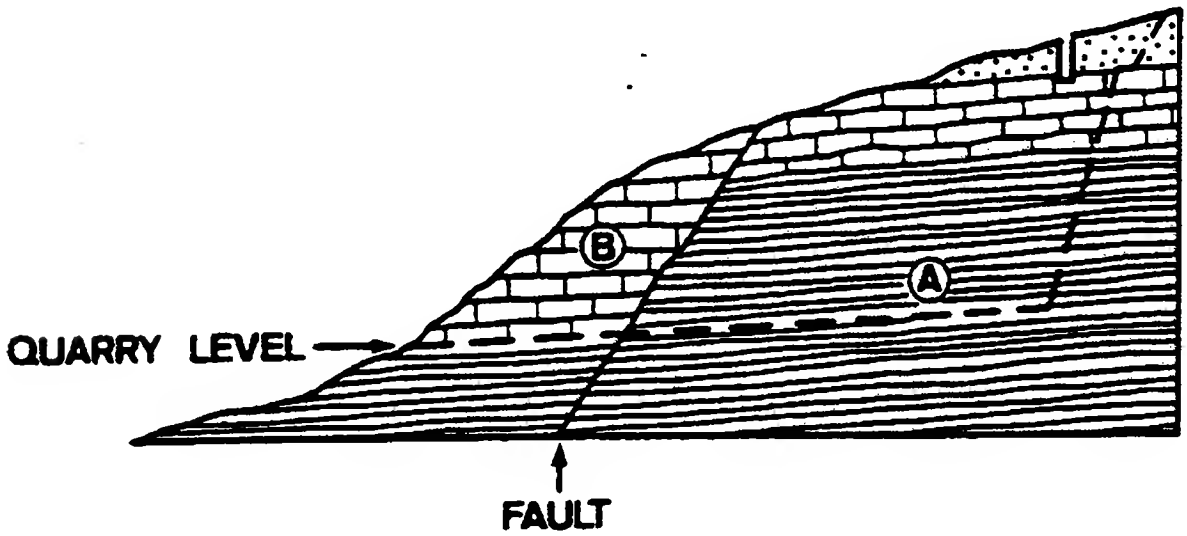
Fig. 8 Significance of outcrop situation





Faults may reduce the potential deposit considerably (Fig. 9).

Fig. 9 Reduction of limestone reserves by fault tectonics



Ⓐ Claystone Ⓑ Limestone

Folding may entail incorrect reserve estimates. In this case (Fig. 10), the claystone formation could not be recognised at the surface. By measuring dip and strike, the anticline structure of this deposit would in all likelyhood have been detected.

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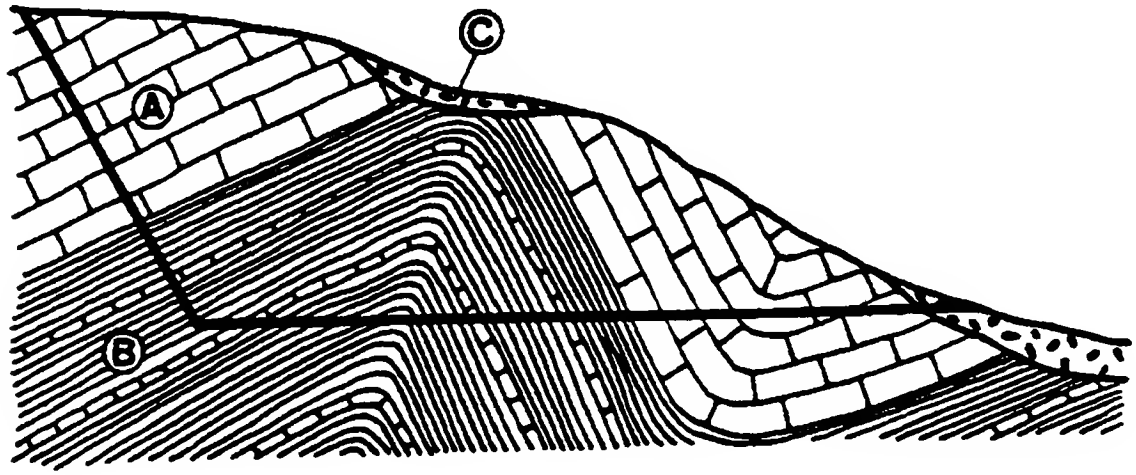
Fig. 10 Significance of Folding



Ⓐ Limestone
Ⓑ Claystone
Ⓒ Scree

Once the geological and geochemical limits are well defined within the deposit boundaries (physical, chemical, legal, etc.) the reserve calculations can be carried out.

Prior to calculating the reserves the following parameters must be defined.

- Geological structure of the deposit: limit of formations
- Structures: dip and strike of stratas, folds, fault
- Overburden
- Groundwater level, natural drainage
- Qualitative limits: chemical
- Mineralogical
- Base mining level and the final slope of the quarry
- Physical properties
- Economical aspect

The volume of exploitable materials can be calculated by a range of computer programs such Surfer, Surpac, ArcInfo, Vulcan, MOSS, Intergraph.

The reserve can be calculated by one of the geometric constructions shown in Fig. 11.

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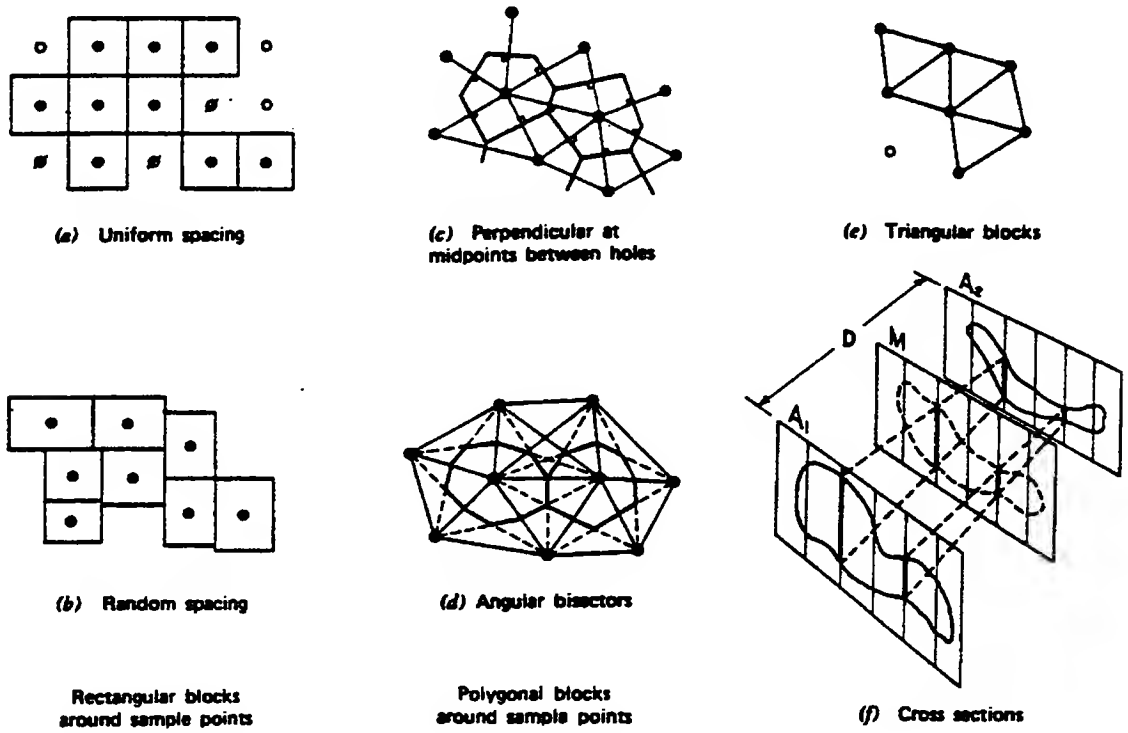
Fig 11 Geometric pattern for reserves calculations



The most suitable method to calculate the reserves is based on geological cross-sections and the distance between each parallel cross section (Fig. 11f).

$V = ((A_1 + A_2)/2) * L_{12} + ... + ((A_m + A_n)/2) * L_{mn}$

V = volume

A_m = surface area of the cross-section m

L_{mn} = distance between cross-sections m and n

For a deposit with horizontal stratification, the volume is quickly calculated by determining the area and multiplying it by the thickness of the useable rocks.

Calculation of the tonnage

The reserves are normally indicated in tons rather than in volume. The bulk apparent specific weight has to be determined on a few samples.

Reserves of raw material = V * r

r = density

3.3 Classification of reserves

Many attempts have been made to establish a worldwide-acceptable classification system for raw material deposits.

It is obvious that a potential deposit of cement raw materials comprises areas which are more thoroughly investigated than others, as well as zones which permit easier exploitation than others; i.e. according to the criteria:

- Degree of recognition

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- Degree of exploitability

The reserves of cement raw material deposits can be divided into different classes. In the interests of presenting a useable system, we at "Holderbank" apply the following classification.

Gelogical reserves

Raw mix reserves

Mineable reserves

Gelogical reserves.

Geological reserves refer to those geological mineral occurrences, which are identified with satisfactory reliability of prediction (certainty of existence and geological assurance, supported with geological maps, borehole information and detailed sampling for analysis). In addition to which the identified reserves meet the preconditions set for the feasibility study (locality, accessibilty, availablity, quantitative and qualitative).

Raw mix reserves.

Raw mix reserves are the total reserves of all components (calcareous, siliceous and corrective), which meet the chemical criteria set for the successful operation of a particular plant.

Mineable reserves.

Mineable reserves are those reserves which fullfil both the aforementioned paramaters and which are practically mineable. Losses due to mining activity may include 'frozen' reserves for haulage and access roads, government and saftey restrictions as well as losses due to practical mining problems.

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Table 2: US Bureau of Mines Classification System (modified)

Principle of a Resources/Reserves Classification for Raw Materials

Cumulative Prodiction (Already produced)	Identified Resources			Undiscovered Resources		
	Demonstrated					
	Measured [Proved] (Shape and contents of the resource is well established)	**Indicated** [Probable] (The degree of assurance is high enough to assume continuity between points of observation)	**Inferred** [Possible[1]] (Continuity is assumed beyond measured or indicated resources for which there is geological evidences)	Hypothetical (In known districts)	Speculative (In unknown districts)	
Economic (Recovrable)	Reserves (Recoverable reserves) (Extractable reserves)		Inferred Reserves	Resources hypothetical	Resources speculative	Towards top: increasing degree of profitable extraction
Error limit % :	± 10	± 20	±30	Similar to known resources that may exist in the same region	Other resources	
Reliability of existence % :	>90[2]	90 - 70	70			
Marginally Economic [Paramarginal] (Would be economic given changes in economic or technical factors)	Marginal Reserves		Inferred Marginal Reserves			
Subeconomic [Submarginal]	Demonstrated Subeconomic Resources		Inferred Subeconomic Resources			
	Towards left: increasing degree of certainity of existence, reliability of prediction and geological assurance					

Reserves Base [Geological Reserves. May include part of the Inferred Reserves]

Inferred Reserves Base

[ ] Old terms or terms used in other classifications

( ) Short explanation of term

1) Some classifications divide "Possible" into "Indicated" and "Inferred"
2) Example: for an estimated reserve of 100 tons, the actual tonnage available lies between 90 and 110 tons with a probability of 90%.

4. OVERBURDEN

Overburden is defined as material of any nature, consolidated or unconsolidated which overlies the deposit of useful materials. Ideally, attempts should always be made to utilise the overburden material as a raw material component. Should utilisation be impossible due to chemical or physical character, the overburden is dumped in selected areas. These areas should be optimised with respect to potential rehabilitation and transportation costs. Moreover, the dumping areas should never obstruct the exploitation. Fig 12 shows few favourable and inappropriate dumping areas. Unsuitable areas of dumps may render further exploitation uneconomical and may reduce the potential reserves considerably.

In the overall investigations, the amount and the quality of the overburden must be defined accurately. An important consideration in the economics of the open pit mining is the **stripping ratio**, which is defined as the ratio of total waste removed to total suitable material mined. The **overburden ratio** is defined as the ratio of the vertical thickness of overburden to the vertical thickness of suitable material.

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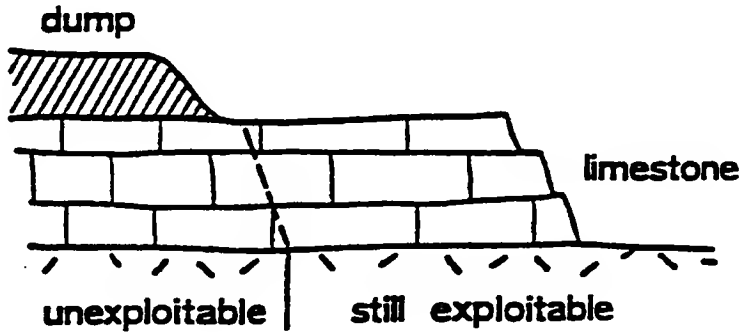
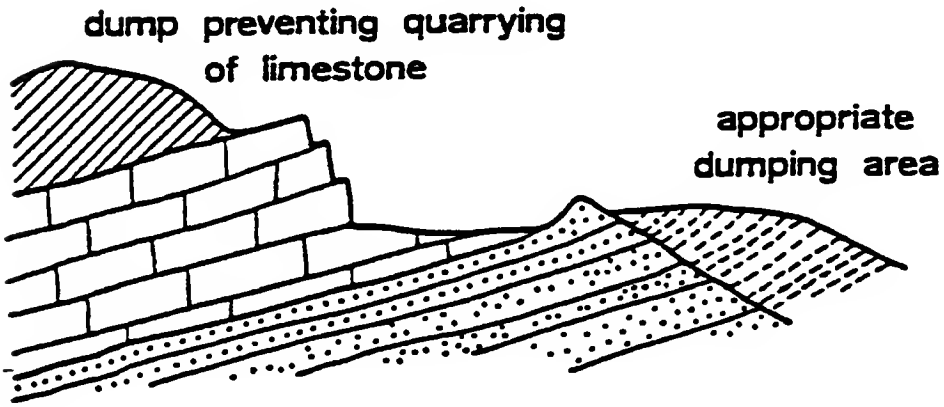
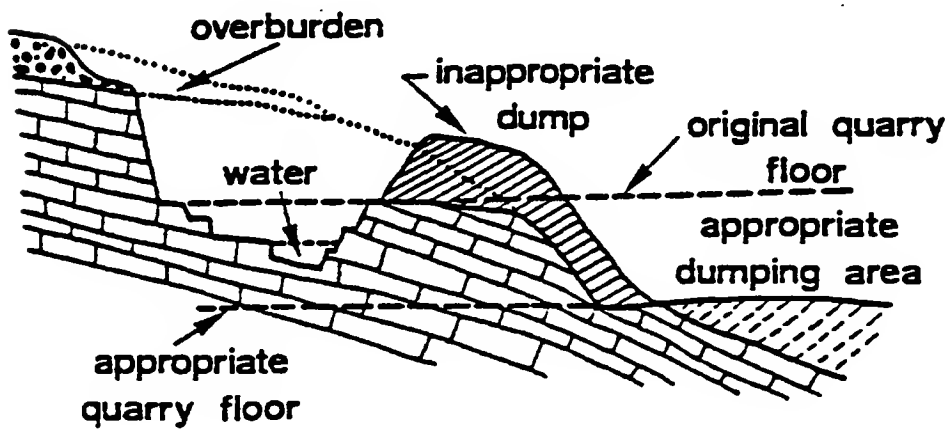
Fig 12: Dumping in quarry area







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The volume of and the contact betwenn overburden/useful materials is determined during exploration activities eg. geological mapping, drilling, geophysics etc.

5. BULK SAMPLING TECHNOLOGICAL TESTS

At the end of the overall drilling campaign, a bulk sample is taken. This bulk sample must be representative of the deposit and comes in general from the core collection. The purpose of the bulk sample is to carry out technological tests useful to dimensioning the machinery.

The following tests are carried out:

- Chemical analyses
- Mineralogy
- Burnability
- Abrasivness
- Grindability
- Moisture content

Other tests can be carried out such as :

Brazilian test :

Uniaxial compressive strength

Bulk density

Poisson's ratio

Young's coefficient.

Crushing test:

% of fines

etc...

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Chapter 6

Assessment of Cement Raw Materials

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ASSESSMENT OF CEMENT RAW MATERIALS

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1. ASSESSMENT OF CEMENT RAW MATERIALS

1.1 Definition of qualitative requirements of clinker and cement

Ordinary Portland Cement (OPC) - by far the most frequently used type of cement -consists mainly of the four major chemical components:

Calcium oxide	64 % CaO	"Lime", Calcium, C
Silicon oxide	22 % SiO_2	"Silica", S
Aluminium oxide	6 % Al_2O_3	"Alumina", A
Iron oxide	3 % Fe_2O_3	"Iron", F

The remainder is composed of:

Na_2O, K_2O	"Alkali"
SO_3	"Sulphur"
MgO	"Magnesium
TiO_2	"Titanium"
etc.	

It should be noted that nature only seldomly provides a raw material of the desired chemical composition. Normally, mixtures of various components are necessary to obtain the raw mix suitable for the production of OPC.

1.2 Raw materials for the cement industry

1.2.1 General

Basically, the following raw material classes can be distinguished:

- Main components
  They contribute CaO, SiO_2, Al_2O_3 and Fe_2O_3. CaO is supplied by the calcareous (carbonatic) component

  and

  SiO_2 Al_2O_3 and Fe_2O_3 by the argillaceous (siliceous) component (s).
- Corrective Materials
  If there is a deficiency of either CaO, SiO_2, Al_2O_3 or Fe_2O_3, respective "corrective" Materials providing the deficient component(s), have to be utilised.
- Additive Materials
  Materials which are added to the raw mix (i.e. to the mix consisting of main components and possibly corrective Materials or to the clinker in order to obtain special effects, are designated as "additive" Materials.

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1.2.2 Calcareous Component

As demonstrated in chapter 1.3, the calcareous component is usually classified according to its chemical Composition i.e. according to its carbonate content ("titration value").

The carbonate content - expressed as $CaCO_3$ content is normally higher than 75 %, the sought after $CaCO_3$ content of the cement raw mix.

The following classification system is employed in the cement industry (Table 19):

Table 19 Classification of calcareous rock (according to chemical composition)

% $CaCO_3$	Designation
95 - 100	high-grade limestone
85 - 95	limestone
75 - 85	marly limestone

However, by applying only chemical Information nothing is stated regarding the appearance, lithology, type of formation, etc. of such rocks. A great variety of rocks may fulfill the above classification (refer Table 20).

Table 20 Calcareous cement raw Materials

"limestone"
marly
limestone
chalk
coral
limestone
marble
"lime-sand"
shell deposits
etc.

The above can all have an identical titration value!

According to specifications for a cement raw mix, a perfectly suited raw material would contain approx. 75 % $CaCO_3$ and 25 % clay and sand components. Such raw Materials very seldom occur in nature as e.g. natural cement stone".

The economic value of a Potential industrially processed cement raw material depends largely on its conformity with the above specifications, i.e. conformity means favourable economy. It is wrong to assume that an increased carbonate content of a calcareous raw material would result in a higher economic value. For example, a calcareous component with 80 - 85 % $CaCO_3$ is often preferable to a high-grade limestone with 95 - 100 % $CaCO_3$ because fewer other raw Materials have to be added to meet the specifications.

Note

It should be pointed out that the titration method for the carbonate determination includes all carbonates, i.e. the "titration value" is not identical with the $CaCO_3$-content but includes $MgCO_3$, $FeCO_3$, Na_2CO_3, etc.

Titration thus produces only guide values which have to be verified by means of total chemical analysis.

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1.2.3 Argillaceous Component

The term "argillaceous" covers the field of cement raw materials with those chemical constituents which are contained in insufficient quantities in the calcareous component, i.e. argillaceous materials basically supply SiO_2, Al_2O_3 and Fe_2O_3 ("silica", "alumina" and "iron").

Because of the various mechanisms in the formation of argillaceous rock, a large number of these rock types are considered as potential cement raw materials (Table 21).

Table 21 Argillaceous materials

clay	"clay" loess	unconsolidated, loose
	claystone mudstone siltstone (slate) shale	consolidated, compact
marl		$CaCO_3$
	calcareous marl	65 - 75 %
	marl	35 - 65 %
	clayey marl	25 - 35 %
	marly clay	15 - 25 %
others	tuff ash	volcanic origin
	phyllite slate amphibolite	metamorphic origin

1.2.4 Corrective Materials

If the main components (calcareous and argillaceous component) do not permit the proportioning of a mix which conforms to the specifications, corrective materials have to be utilised.

Contrary to the main components, the concentration of the desired constituent (SiO_2, Al_2O_3 or Fe_2O_3) in the corrective is of utmost importance. Since corrective Materials (Table 22) are usually expensive (bought from special suppliers) and occasionally have to be transported over long distances, the corrective should be of high quality and used in small quantities.

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Table 22 corrective Materials

To correct the deficiency of	corrective material	minimum concentration
"iron" (Fe_2O_3)	Iron ore Pyrites ash	60% Fe_2O_3 60% -90%
"alumina" (Al_2O_3)	Bauxite Laterite Kaolinite	50% Al_2O_3 25% " 50%
"silica" (SiO_2)	Quartzsand Quartzite	80% SiO_2 80% "
"lime" (CaO)	Limestone Marble	90% $CaCO_3$

In rare cases, where the abundant calcareous materials do not contain a sufficient amount of calcium carbonate, a high-grade limestone (CaO) - corrective has to be supplied.

1.2.5 Additive Materials

Additive materials (Table 23) include, in most cases, only sulphate-bearing components which are added to the clinker by inter-grinding for control of the setting of cement. In earlier times, another additive material (fluorspar) was frequently added to the kiln feed in order to improve the burnability. Today, it is only employed in very rare cases (manufacture of white cement).

A great variety of special cements is produced by blending OPC with either waste or by-products of other industries (steel industry, power plants) or with naturally occurring materials, yielding latent hydraulic properties (pozzolana).

Table 23 Additive materials

Additive	minimum concentration	purpose of addition
Gypsum $CaSO_4 . 2 H_2O$ Anhydrite $CaSO_4$	35% SO_3 50% SO_3	production of OPC (95% clinker + 5% gypsum)
Fluorspar	30% F	improvement of burnability
Blastfurnace slag Pozzolana Fly ash	none	special cements

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1.2.6 Combination of Components

Criteria for the selection and combination of raw material components are:

- chemical composition
  (overall, homogeneity, variability, etc.)
- technological properties
- Transport and costs
- material casts
- operation (handling, preparation, etc.)

Basically, it is desirable to reduce the number of components to a minimum - taking of course the above i.e. two-component mixes instead of three or four component mixes. For instance:

limestone clay	instead of	limestone sand iron ore bauxite

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Potential cement raw mixes with a large number of components (more than three or four) may be subject to difficulties regarding dosage, control of components and of cement raw mixes, etc. Industrially produced cement raw mixes are compiled in Table 24.

Table 24 Examples of combinations of components ("Holderbank"-Group plants)

Mississauga 98 - 100 % marly limestone 0 - 2 % shale	Guayaquil 82 - 85 % limestone 12 - 15 % clay 1 - 3 % iron ore
Apasco 77 - 85 % limestone 8 - 22 % clay 0 - 4% sandy clay 1 -3 % iron ore	Geelong 55 % high-grade "limestone 44 % low-grade "limestone" 1% pyrites ash
Atocongo 27 % limestone 73 % marl	Hardegsen 92 % limestone 7 % sand 0.5 % iron ore 0.5 % gypsum (SO_3)
Darra 91 % coral 5 % sand 3 % clay 1 % iron ore	Obourg 89 % limestone 6 % sandy marl 1 % pyrites ash (Fe_2O_3) 1 % coal mine waste 3 % coal mine waste containing combustibles
Dudfield 88 % limestone 6 % clay 6 % boiler ash (Al_2O_3) 0 - 1 % magnetite (Fe_2O_3) pyrites ash (Fe_2O_3)	
Wunstorf 91 % limestone / marl 1.5 % clay 4.54 % sand 3 % iron ore	Gmunden 68 % limestone 29 % marl 3 % iron oxide

1.3 Characteristics of cement raw material

1.3.1 Chemical Characteristics

Any assessment of potential cement raw Materials is predominantly based on their chemical composition The following system (Table 25) for the presentation of chemical results has been developed:

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Table 25 Form for presentation of chemical results

Constituents	Designation
* SiO_2 AL_2O_3 Fe_2O_3 CaO	"main elements" (oxides)
MgO SO_3 K_2O Na_2O	"minor elements"
TiO_2 Cr_2O_3 Mn_2O_3 P_2O_5 Cl F	"Trace elements"

* plus loss on ignition

It should be mentioned that the assessment of each raw material component has to be made with consideration of a potential raw material combination; i.e. it should not be assessed in isolation but take into account the other components involved and the specifications for the cement raw mix.

For example:

A limestone yielding 98 % carbonate content may contain 3.5% MgO. An isolated assessment would result in the rejection of this Materials If the raw mix consists of 20 % clay (say without MgO) and 80% limestone to achieve a carbonate content of 75%, a MgO content of approx. 2.8% (4.5 % resp. when loss on ignition free) results, which is still tolerable for most markets.

The following tables (26 - 30) display a series of analytical results of cement raw materials and are to be interpreted according to the above criteria.

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Table 26 Analyses of calcareous cement raw materials

Designation	lime-stone	high-grade limestone	chalk	sandy limestone	marble	calcareous marl	phosphatic limestone	dolomitic limestone	dolomite
Ignition loss	42,90	42,90	42,26	35,23	43,14	30,60	42,0	37,5	45,63
SiO_2	5,19	0,70	2,38	15,74	1,10	13,80	2,0	16,7	3,24
Al_2O_3	0,81	0,68	1,57	0,88	0,51	7,00	0,37	2,1	0,17
Fe_2O_3	0,54	0,08	0,56	0,51	0,25	4,55	0,56	0,27	0,27
CaO	48,61	54,54	52,48	44,89	54,33	38,35	52,1	33,6	29,58
MgO	1,90	0,59	0,59	2,54	0,48	1,32	0,76	9,3	20,84
SO_3	0,05	0,25	0,01	0,61	0,04	0,43	0,23	0,08	0,02
K_2O	0,33	0,01	0,02	0,21	0,01	0,86	0,08	0,29	0,11
Na_2O	0,05	0,16	0,02	0,08	0,01	2,61	0,36	0,40	0,02
TiO_2	0,06	0,01	0,01	0,05	0,02	0,21	0,01	0,07	0,01
Cr_2O_3	0,01	0,01	0,01	0,01	0,01	0,02	0,01	0,01	0,01
Mn_2O_3	0,05	0,01	0,01	0,03	0,08	0,29	0,06	0,06	0,01
P_2O_5	0,04	0,01	0,01	0,10	0,03	0,25	0,98	0,20	0,02
Cl	0,02	0,01	0,03	0,11	0,01	0,04	0,04	0,12	0,03
F	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01
Total	100,11	99,97	99,97	101,00	100,16	100,34	99,57	100,71	100,97

Theoretical composition of dolomite:

MgO 21.9 %, CaO 30.4 %, CO_2 47.7 %

Table 27 Rock Analysis of argillaceous cement raw materials

Designation	Shale	Shale	Marl	Claystone	Clay	Calcareous sandstone	Sandy shale	"Slate"	"Cement-stone"
Ignition loss	3,71	5,3	32,24	13,3	12,50	23,8	3,9	5,58	34,44
SiO_2	61,1	64,1	22,7	48,6	45,90	46,3	70,8	56,29	11,2
Al_2O_3	16,4	13,6	3,9	19,6	23,90	1,2	10,9	19,22	4,02
Fe_2O_3	7,1	6,1	2,4	9,1	15,50	0,64	5,2	4,39	1,68
CaO	1,1	1,8	32,9	2,5	0,72	28,1	1,3	0,09	44,01
MgO	2,4	2,9	3,32	1,6	0,33	0,48	2,2	1,65	1,30
SO_3	0,01	0,03	0,95	1,78	0,03	0,04	0,03	0,72	0,13
K_2O	4,8	2,7	0,59	0,54	0,01	0,06	2,2	10,85	0,96
Na_2O	1,6	1,9	0,19	1,00	0,11	0,10	2,0	0,19	0,42
TiO_2	0,76	0,67	0,39	1,20	0,86	0,06	0,71	0,64	0,68
Cr_2O_3	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,02
Mn_2O_3	0,06	0,05	0,04	0,04	0,20	0,04	0,05	0,22	0,45
P_2O_5	0,13	0,18	0,03	0,18	0,06	0,03	0,17	0,15	0,12
Cl	0,02	0,01	0,01	0,22	0,04	0,03	0,01	0,06	0,04
F	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01
Total	99,21	99,36	99,68	99,68	99,88	100,80	94,49	100,07	99,53
Igneous and metamorphic equivalents	Granite, Rhyolithe Quartzdiorite Gneiss		Calcsili-cate fels	Diorite, Basalts Andesite Gneiss		Calcsili-cate fels	Granite Gneiss	Syenite Trachite Calcsili-cate fels	Sili-ceous marble

Table 28 Analysis of corrective materials

Corrective for	Aluminium			Iron				Silicon	
Designation	Bauxite	Bauxite	Bauxite	Pyrites ash	Hema-tite	Umber	Iron slag	Sand	Sandstone
Ignition loss	10,5	29,35	15,4	n.d.	3,41	13,99	1,6	0,91	3,69
SiO_2	4,3	3,3	3,1	7,8	11,0	32,1	21,5	94,70	80,2
Al_2O_3	56,0	56,6	57,6	1,0	3,1	4,3	8,8	2,90	8,37
Fe_2O_3	22,1	9,4	15,8	84,3	77,7	37,5	57,6	0,24	3,26
CaO	n.d.	1,15	4,1	0,8	1,2	1,42	3,1	0,35	3,29
MgO	n.d.	0,08	0,16	0,45	0,32	1,70	3,6	0,13	0,45
SO_3	n.d.	0,24	0,29	11,73	2,10	0,04	2,3	0,01	0,08
K_2O	n.d.	0,15	0,08	0,09	0,48	0,69	n.d.	1,3	1,90
Na_2O	n.d.	0,10	0,08	0,07	0,07	0,30	n.d.	0,20	0,11
TiO_2	3,0	n.d.	2,9	0,16	0,19	3,24	n.d.	0,12	0,48
Cr_2O_3	n.d.	n.d.	0,02	n.d.	0,01	0,24	n.d.	0,01	0,01
Mn_2O_3	n.d.	n.d.	0,03	0,04	1,31	5,75	n.d.	0,03	0,01
P_2O_5	n.d.	n.d.	0,21	n.d.	0,02	0,16	n.d.	0,01	0,02
Cl	n.d.	n.d.	0,01	n.d.	0,02	0,08	n.d.	0,01	0,01
F	n.d.	n.d.	0,01	n.d.	0,01	0,02	n.d.	0,01	0,01
Total	n.d.	100,37	99,79	n.d.	100,94	101,53	98,50	100,93	98,89

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Table 29 Analysis of attitive materials

Design.	Gypsum	Gypsum	An-hydrite	Fluor-spar	Pozzo-lana	Fly ash	Blast fur-nace slag
L.o.i.	22.3	12.8	0.3	4.6	8.8	1.5	0.18
SiO_2	1.3	9.2	4.8	9.2	54.1	50.1	36.3
Al_2O_3	0.71	2.6	2.6	0.73	17.1	29.0	9.2
Fe_2O_3	0.3	0.95	2.1	1.8	5.5	7.9	0.2
CaO	30.6	30.3	36.2	ca.39.9	4.1	4.5	32.9
MgO	0.05	0.5	2.5	0.11	1.6	2.2	10.4
SO_3	45.5	37.2	51.3	3.1	0.15	0.24	3.4
K_2O	0.08	1.18	0.10	0.26	5.1	2.6	1.3
Na_2O	0.2	0.97	0.02	0.14	2.3	0.37	0.3
TiO_2	0.01	0.88	0.01	0.26	0.78	0.91	0.3
Cr_2O_3	0.01	0.16	0.01	n.d.	0.01	0.02	0.01
Mn_2O_3	n.d.	0.64	0.02	0.02	0.14	0.05	1.5
P_2O_5	n.d.	1.52	0.12	0.05	0.22	0.16	0.01
Cl	n.d.	0.16	0.02	n.d.	0.04	0.01	0.01
F	n.d.	0.35	0.01	37.8	0.01	0.01	0.01
Total	101.06	99.61	100.11	97.99	99.95	99.47	96.13

Theoretical composition of pure gypsum ($CaSO_4 \cdot 2H_2O$)

SO_3 = 46.5 %
CaO = 32.5 %
H_2O = 21.0 %

Theoretical composition of pure anhydrite ($CaSO_4$)

SO_4 = 58.8 %
CaO = 41.2 %

Theoretical composition of fluorspar (CaF_2)

F = 48.7 %
Ca = 51.3 %

Carbon not included

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Table 30 Typical analyses of cement raw materials

n.loss	3.01	19.17	3.21	8.05	26.89	42.56	41.35	15.4	1.9	13.45
O_2	78,70	43,28	66,55	58,11	27,86	1,65	8,36	3,1	3,4	48,15
$_2O_3$	7,53	9,31	16,93	15,40	3,64	0,45	1,77	57,6	0,91	12,6
$_2O_3$	4,57	3,90	3,89	6,47	1,54	0,02	1,08	15,8	90,0	5,2
O	1,29	17,55	1,07	3,10	31,39	54,23	29,03	4,1	0,66	10,15
O	1,32	1,62	0,96	2,44	4,44	0,11	16,68	0,16	0,20	5,25
$_3$	0,03	0,02	0,26	0,65	0,09	0,06	0,14	0,29	1,5	0,45
O	2,24	2,40	3,60	3,24	2,48	0,36	1,08	0,08	0,15	2,5
$_2O$	1,14	1,22	2,05	1,30	0,78	0,28	0,06	0,04	0,09	1,03
O_2	0,51	0,68	0,80	0,65	0,20	0,04	0,01	2,9	0,10	0,61
$_2O_3$	0,02	0,02	0,01	0,02	0,02	0,01	0,01	0,03	0,03	0,01
$_2O_3$	0,08	0,11	0,04	0,16	0,02	0,02	0,03	0,03	0,03	0,07
$_2O_5$	0,09	0,75	0,45	0,17	0,20	0,05	0,06	0,17	0,05	0,14
	0,06	0,01	0,04	0,08	0,06	0,02	0,01	0,01	0,01	0,01
	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01
otal	100,60	100,05	99,87	100,30	99,62	99,87	99,68	99,72	99,04	99,63

1.3.2 Crystallographic Characteristics

1.3.2.1 The Crystalline and Amorphous State

Most materials, e.g. rocks and soil, metals and alloys, concrete and organic substances, are "crystalline", i.e. the atoms, ions and-molecules which form the crystals are arranged in strict geometrical order.

"Amorphous" solids on the contrary are substances with no strict geometrical order, i.e. which are not "crystalline". Glasses and certain resins belong to this class.

1.3.2.2 Definition of a Crystal

Crystals are chemically and physically homogeneous, regularly shamed polyhedral bodies.

1.3.2.3 Space Lattice and Unit Cell

Geometrically, a space lattice (Fig. 33) is defined as a regular and unlimited Arrangement of identical points in space. The lattice of a crystal is merely the repetition in three dimensions of the positions occupied by its atoms or molecules. As Fig. 33 shows, a crystal is built up of a multitude of "unit cells" having edges a_o, b_o, c_o and angles, α, β, δ,. The length of the edges, called lattice constants, is of the order of a fraction of a nanometer (1 nm = 10^{-9} m; equivalent to 10 Ångströms as an old unit).

The unit cell may be a regular cube, a prism, a rhombohedron or, generally, a parallel epipedon. The lattice constants a_o, b_o. and c. may be equal to one another or different, as may also the angles α, β, δ,., According to the combination, seven different crystal systems occur which are summarised in Table 31.

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Fig. 33 Crystal lattice

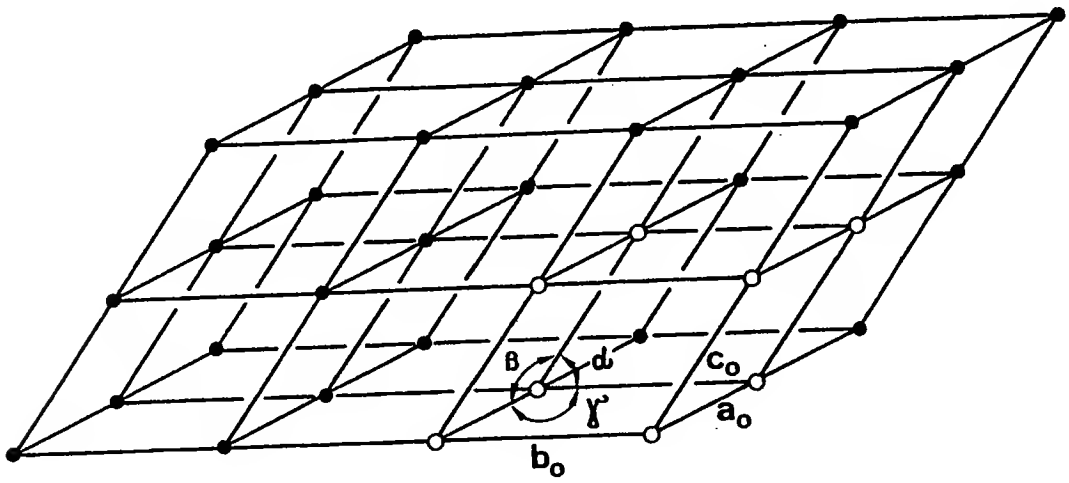

Table 31 Definition of crystal symmetry systems

System	Axial relationships	Angles between crystal axes
Triclinic	$a \neq b \neq c$	$\alpha \neq \beta \neq \gamma \neq 90^\circ$
Monoclinic	$a \neq b \neq c$	$\alpha = \gamma = 90^\circ$; $\beta \neq 90^\circ$
Orthorhombic	$a \neq b \neq c$	$\alpha = \beta = \gamma = 90^\circ$
Trigonal (Rhombohedral)	$a = b = c$	$\alpha = \beta = \gamma \neq 90^\circ$
Hexagonal	$a_1 = a_2 = a_3 \neq c$	$\alpha_1 = \alpha_2 = \alpha_3 = 120^\circ$; $\gamma = 90^\circ$
Tetragonal	$a = b \neq c$	$\alpha = \beta = \gamma = 90^\circ$
Cubic (isometric)	$a = b = c$	$\alpha = \beta = \gamma = 90^\circ$.

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According to the positions of atoms, ions and molecules (on the edges or on the faces or in the centre of the unit cell) fourteen space lattices are possible (Fig. 34). Every crystal belongs to one of these crystal systems and is composed of an infinite repetition in the three dimensions by one of the 14 space lattices.

A simpler presentation and characterisation of the point lattice is achieved by the introduction of "lattice planes". Equispaced planes, i.e. parallel planes with same distance represent identical two-dimensional arrangements of atoms and molecules (Fig. 33).

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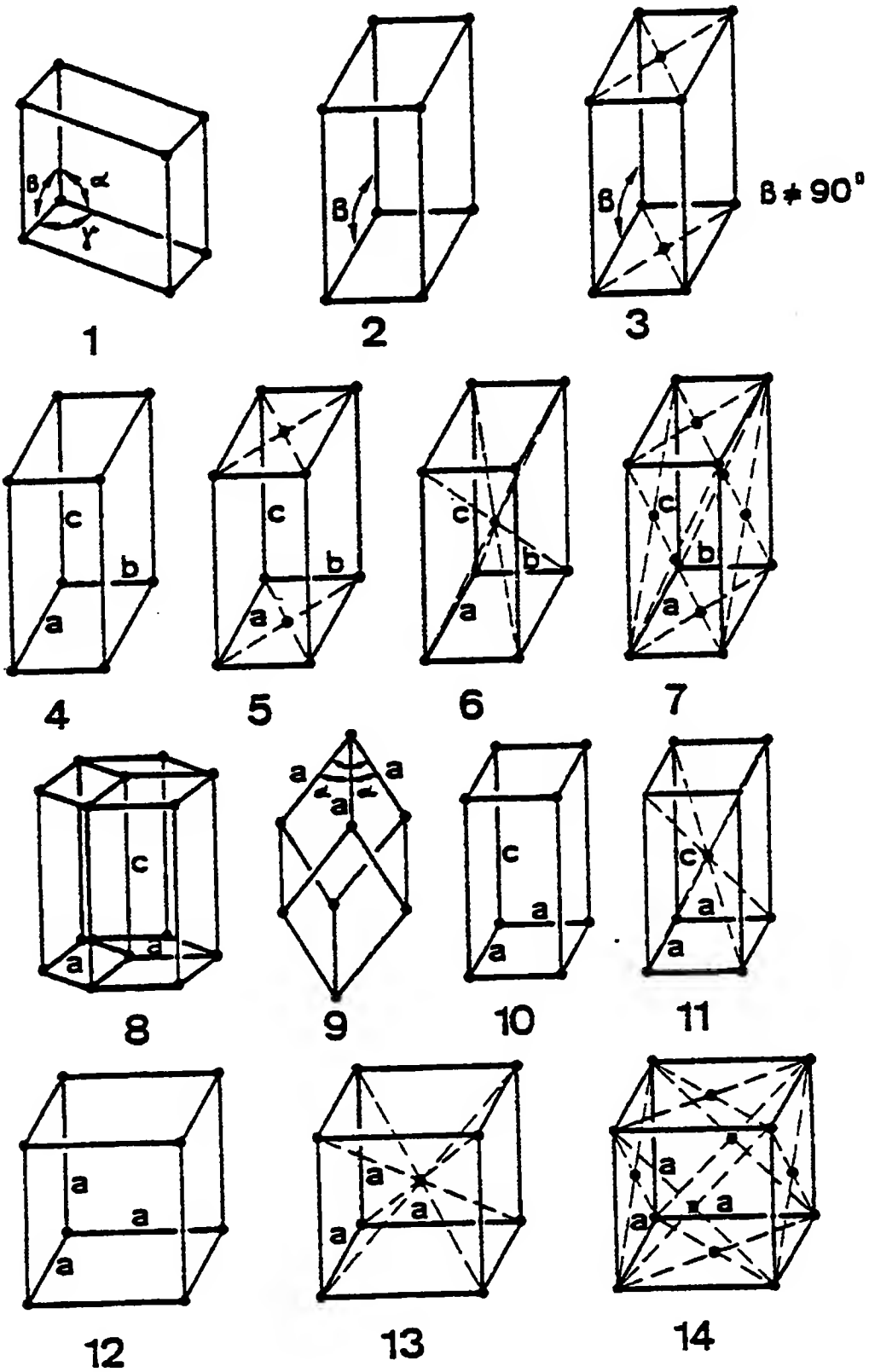
Fig. 34 Space lattice



1.3.2.4 Symmetry Elements of Crystals and Crystal Symmetry Classes

Symmetry is an essential feature of crystals. The symmetry elements are simple and few in number, viz. axes, planes and centre of symmetry.

The symmetry properties of all crystals are combinations of these axes, planes and centre of symmetry. The total number of possible combinations of symmetry elements leads to 32 crystal symmetry classes.

1.3.2.5 Definition of Minerals

A mineral is defined as a solid characterised by its chemical composition and crystal structure (lattice constants and symmetry).

1.3.2.6 Ray Diffraction for Determination of Crystal Structure and Identification of Minerals

X-ray diffraction is a tool to supply data and information on the crystal structure not obtainable by other techniques. In the cement industry it is applied for the identification of the mineralogical composition of raw Materials clinker, rings, refractory Materials cement and hydration products. It is based on the determination of the distance "d" between equispaced planes of the crystal lattice (4.3.2.3). A set of characteristic d - values represent the "fingerprint" of a mineral.

When a monochromatic beam of X-rays of wavelength (X of the same order of magnitude as the distance between lattice points) falls on a crystal, the characteristic fingerprint of the corresponding mineral is produced by reflection. This phenomenon is not a surface reflection as with ordinary light. Rather it can be compared with the diffraction of a monochromatic light-beam on an optical lattice. This phenomenon normally called "X-ray diffraction" is explained as follows (Fig. 35):

Parallel to the crystal face is an infinite series of equispaced atomic planes, spaced at a distance d. The X-rays penetrate to a depth of several million of such planes, whereby they may undergo the same reflection as on the surface plane. If these various reflections arrive in phase, they form a composite reflected X-ray beam. If the reflection on the multitude of identical planes occurs out of phase, the X-ray in the crystal is absorbed and no reflection occurs.

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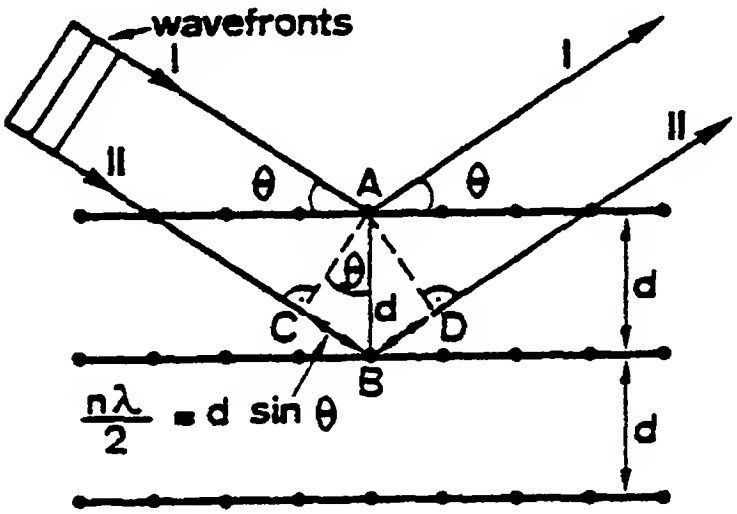
Fig. 35 Reflection of X-rays on atomic planes



θ = incident angle of the X–Ray beam

d = distance between the parallel atomic planes

λ = wavelength of the monochromatic X–Ray beam

Geometrical considerations (Fig. 35) demonstrate that reflection occurs only under the following, so-called Bragg conditions:

$$n\lambda = 2d \sin\theta \quad \text{or} \quad d = \frac{n\lambda}{2\sin\theta}$$

n = integer, order of diffraction

For the practical application in the cement industry, x-ray diffraction analysis is carried out on finely ground samples ("powder method"). This means that with in the powder sample an infinite number of minute crystals is randomly distributed and oriented. Therefore, all possible series of equispaced identical lattice planes with all the different d values are recorded.

Fig. 36 shows in a schematic way the principle of powder-x-ray diffractometry from which an x-ray diffraction diagram results (Fig. 37).

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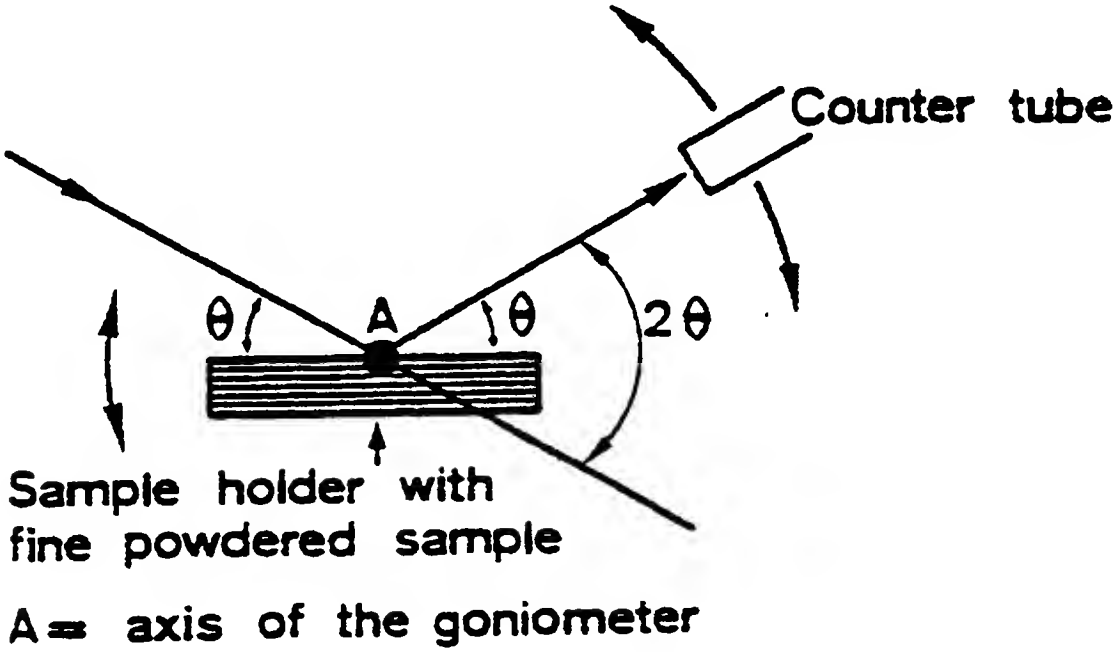
Fig. 36 Diffractometer (Goniometer type)



The diffraction method requires only small amounts (a few grams) of material.

The main application of X-radiation in the cement industry is that of X-ray fluorescence which is used for the determination of the chemical Composition of materials (see section 15, quality assurance, p. 15/33ff). The differences in principle (figure 38) and application between fluorescence and diffraction are briefly summarised in Table 31a.

Table 31a Difference between x-ray diffraction and x-ray fluorescence

	Diffraction (XRD)	Fluorescence (XRF)
Application	Analysis of mineralogical composition	Analysis of chemical composition
Principles		
- Bragg-equation 2d sin θ = nλ	determination of d-values according to 2 θ ("fingerprint" of mineral) 2d : variable sin θ : variable λ : constant	determination of λ according to 2 θ 2d : constant sin θ : variable λ : variable
- radiation (λ)	primary and diffracted radiation monochromatic with the same λ	primary radiation polychromatic, emitted radiation consisting of distinct characteristic wavelengths
- crystal structure (d-value)	identification of mineral according to d-value	special single crystal with defined d-value and orientation (analyzing crystal is used to disperse distinct characteristic wavelengths)

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Fig. 37 X-ray diffraction diagram

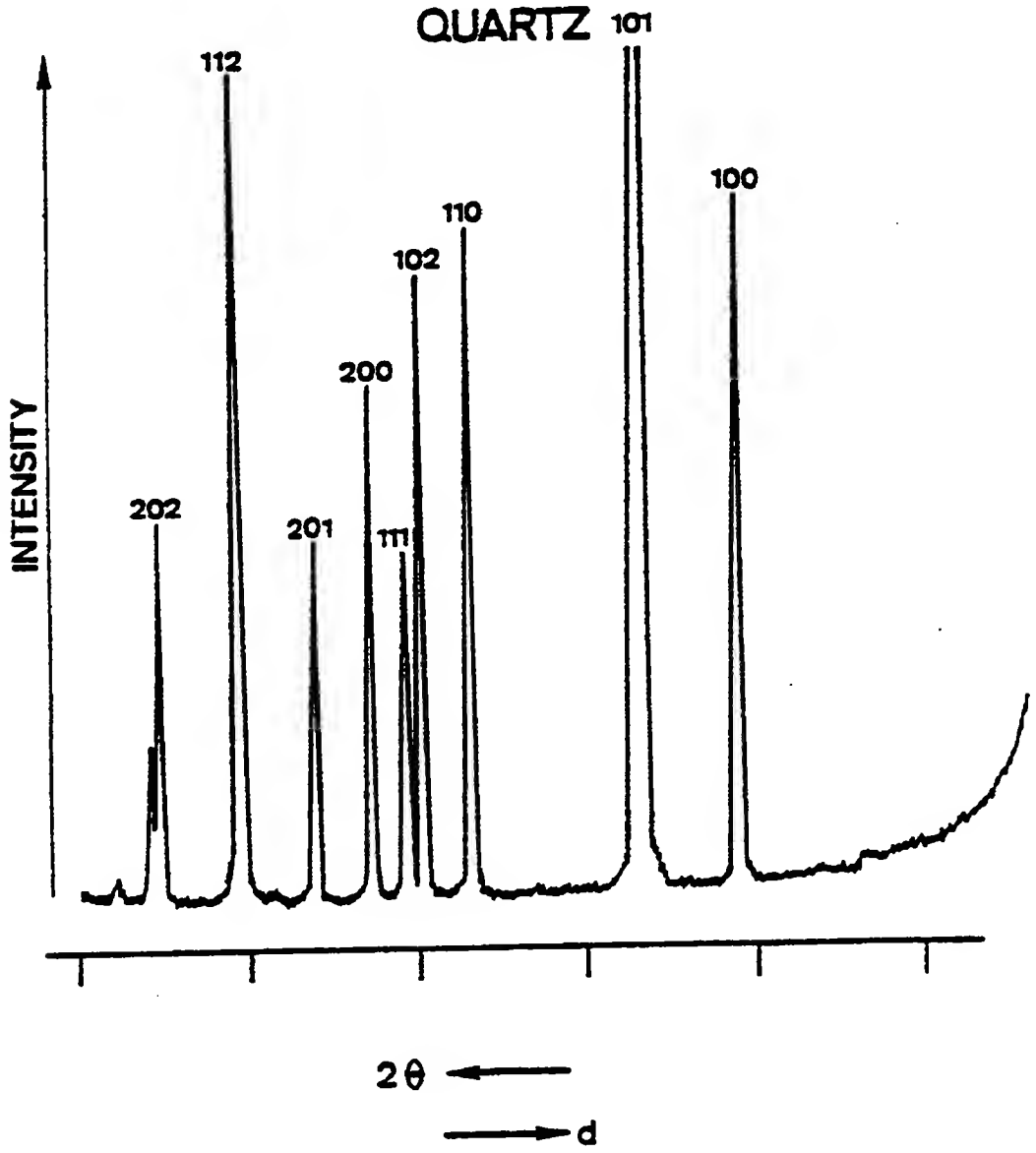
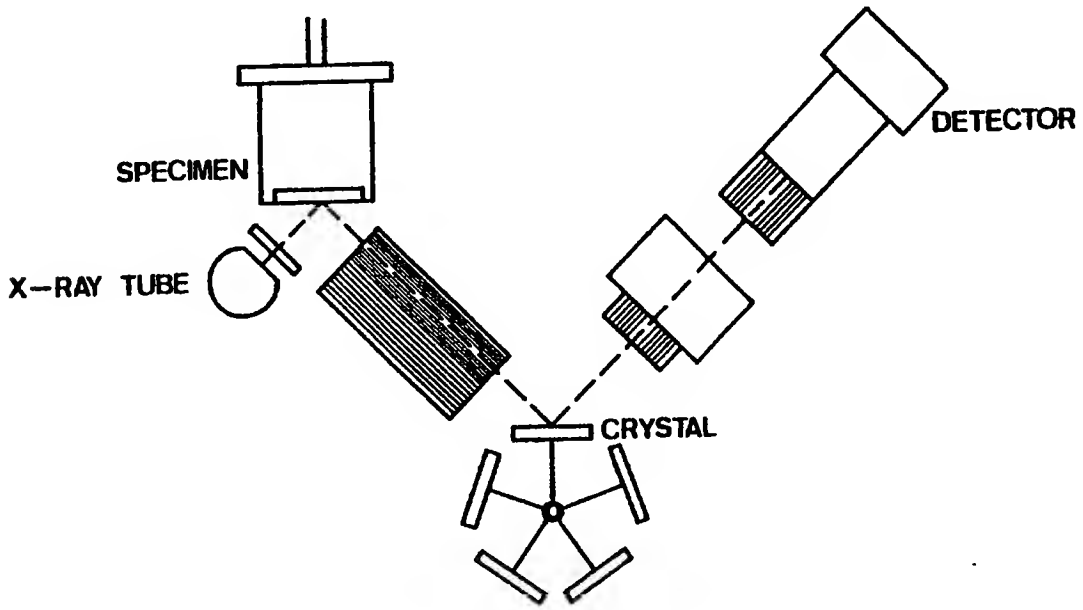


Fig. 38 X-ray fluorescence spectrography



1.3.2.7 Polymorphism

An element or compound is defined as polymorphic if it forms two or more crystalline solid phases differing in atomic Arrangement with identical chemical Composition The polymorphism depends on pressure and temperature, i.e. a particular modification is only stable under defined pressure and temperature conditions.

As a consequence of polymorphism it occurs that materials with identical chemical Composition may exhibit different physical and chemical properties. A good example is pure carbon in graphite and diamond form. In the graphite form the carbon is black, soft, and is used, for example, as a lubricant or for electrodes. In the diamond form it is clear, transparent and hard a much coveted precious stone. The lattice structures of these two polymorphic crystalline forms are shown in Fig. 39.

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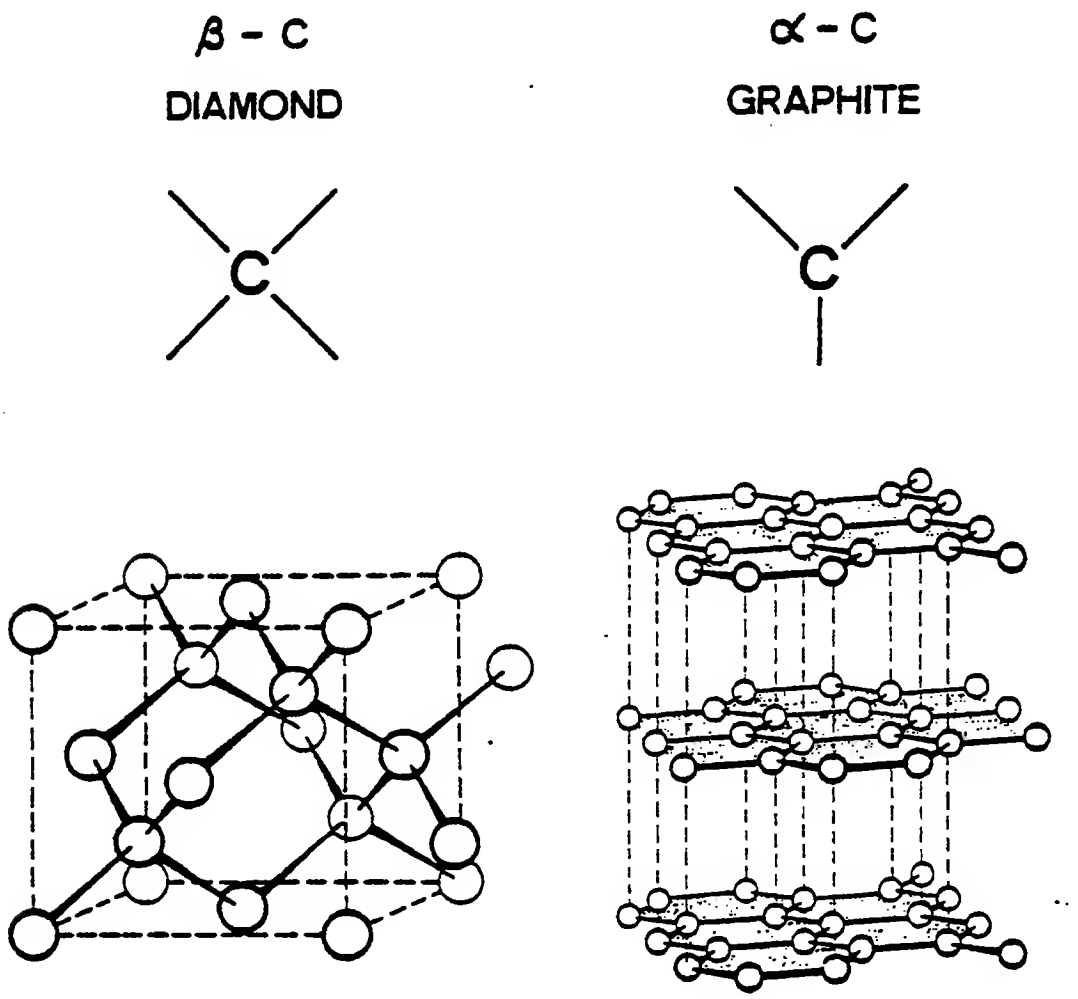
Fig. 39 Lattice Structures of Carbon



Further examples of polymorphism are:

- $CaCO_3$ as aragonite and calcite
- The crystalline forms of silica (SiO_2; see also p. 4/36.ff). At atmospheric pressure silica exists according to the temperature range in three crystalline forms (see fig. 40). Furthermore, each of the three forms exists in a low- and high-temperature modification (fig. 41 for quartz).

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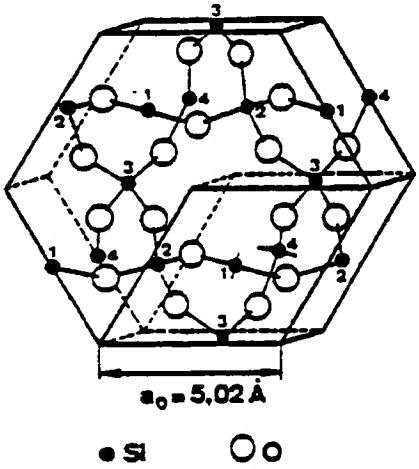
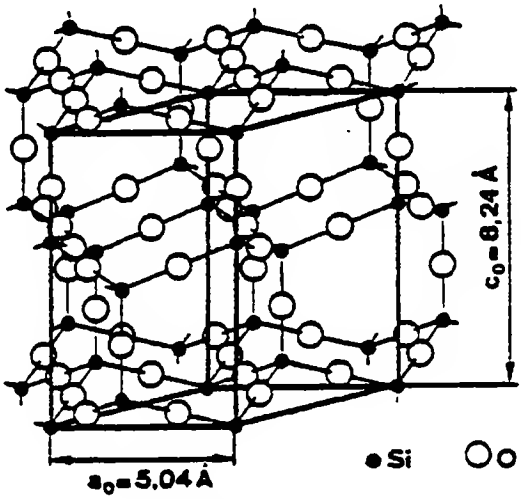
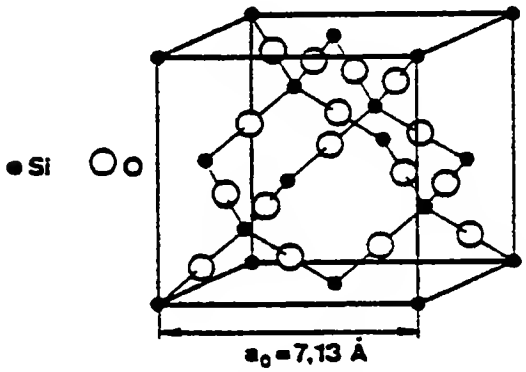
Fig. 40 Polymorphism of SiO_2



QUARTZ



TRIDYMITE



CRISTOBALITE

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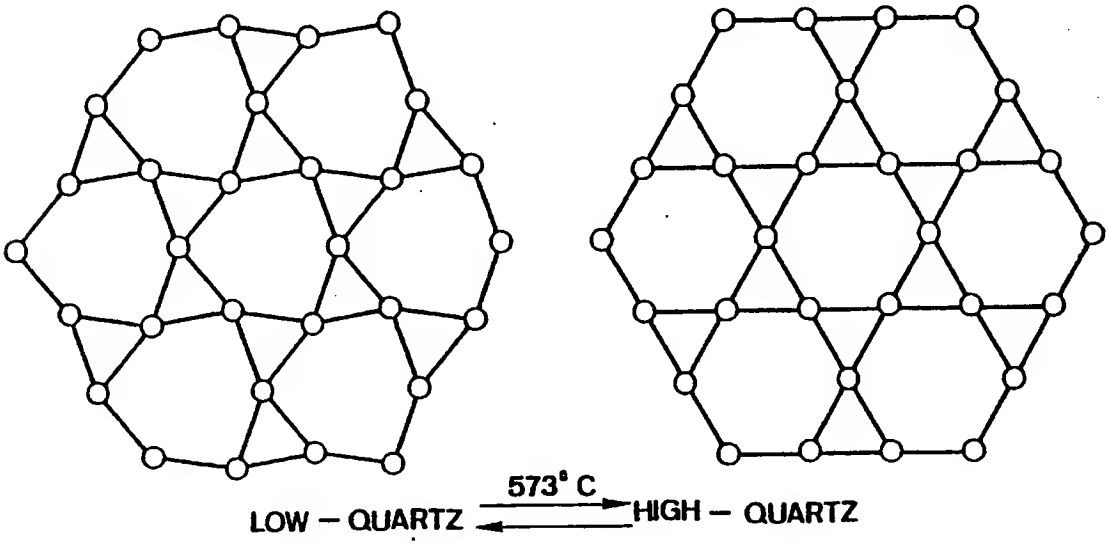
Fig. 41 Si-Atoms in Low and High Quartz



1.3.2.8 Temperature Influence

The lattice points joined with lines producing a lattice are illustrated in Fig. 33. In reality, the atoms, ions and molecules are tightly packed in the crystal lattice with distances close to their atomic radii. The lattice forces, holding together the geometrical Arrangement of atoms, ions and molecules are of a physical and chemical nature.

At absolute zero temperature(0°K = -273°C), the atoms are at rest with their centres in the equilibrium position. By raising the temperature, the atoms begin to oscillate: the higher the temperature, the stronger are their oscillations. With increasing temperature, the oscillation becomes so strong that the lattices may change their symmetry. This occurs at a defined, characteristic temperature. At even higher temperature, the atoms oscillate so violently that they leave the lattice places; the lattice literally falls apart; the crystal melts.

Conversion of the raw material into clinker in a cement kiln requires a high temperature which means high energy. This is the energy required to destroy the crystal lattices of the raw material minerals in order to transform them into crystal lattices of clinker minerals. The energy necessary to destroy the lattice depends an the lattice forces, i.e. on the crystal system, the lattice constants, the nature of atoms and ions, and on the perfection of the lattice. This explains why one particular raw material is more easily burned to clinker than another. Therefore, the type and state of the crystals present in the raw material determine burnability, i.e. the energy required to form the clinker.

Minerals with strong lattice forces which impair the burnability are :
quartz and feldspars

Minerals with weaker lattice forces and thus better burnability are :
clay minerals

Materials with a highly disordered structure, showing high reactivity are :
amorphous and glassy rocks such as tuff, lava, amorphous silica, etc.

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1.3.3 Mineralogical Characteristics

1.3.3.1 General

Mineralogy is concerned with crystalline solids all kinds, whether natural or industrial. Crystalline solids of all kinks, whether natural or industrial. Crystalline solids come in endless variety: the rocks of the earth's crust, the concrete of buildings and steel are a few examples of materials composed of crystals.

1.3.3.2 Minerals and their Properties

For the sake of practicality, minerals are classified into rock-forming minerals, ore minerals, clinker minerals, slag minerals, natural gemstones, synthetic gemstones, ceramic phases, etc. Whether these are natural or artificial is irrelevant; all are crystalline phases.

The properties of minerals depend upon the crystal structure (type of crystal lattice), lattice forces (arrangement and chemical binding of atoms) and lattice energy.

Apart from the chemical composition (4.3.1)the most important properties with regard to the cement-making process are:

- Reactivity and kinetic behaviour

In many cement raw mixes, the minerals are in a stable condition with low free energy, and therefore, a low reactivity. To produce clinker minerals much energy is needed to destroy the structure of the raw materials and to form new structures (clinker minerals).

The following phenomena improve the reactivity of minerals:

- impurities (atoms, ions, molecules) disturbing the ideal crystal lattice
- lattice defects

They give the crystal structure a higher degree of disorder which is equal to a lower degree of crystallinity:

ideal crystal (ordered lattice)	real crystal (partially disordered lattice)	glass (strongly disordered lattice)
———	reactivity	———→
←———	degree of crystallinity	———

At room temperature conditions, a mineral or a mineral combination of raw mix is in a stable condition To let a reaction take place, at least one condition pressure or temperature - has to be changed. The crystal lattices of the minerals present in the raw mix have to be disintegrated in the course of the clinkering process. This is best achieved by applying a high temperature. The minerals thus become unstable. As there is always a tendency from unstable towards stable conditions, reactions leading to a new equilibrium take place.

The equilibrium can be attained by e.g.:

- modification changes of minerals
- melting and evaporating reactions
- chemical reactions between the different minerals with the formation of new minerals

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While the correct chemical raw mix composition is a requirement for the clinker phases, the mineralogical characteristics of the mix determine how the reactions leading to these clinker phases take place. Because of the broad variability of mineralogical properties, many different paths of reaction exist (more details are given in section 10 concerning clinker manufacture).

Hardness

Hardness is defined as the resistance of a mineral to scratching. A scale of relative hardness was introduced by Mohs. The Mohs scale contains 10 hardness classes each characterised by specially selected minerals (refer Table 34, p.55 and Table 35, p.56

Other mineral properties are:

- Density
- Cleavage and fracture
- Electrical properties
- Heat conductivity and heat expansion
- Surface properties

1.3.3.3 Determination of Minerals

For identification and quantification of minerals, the chemical analysis is usually combined with:

- x-ray diffraction analysis
- thermal analysis
- light microscopy

1.3.3.4 Minerals in Cement Raw Materials

Basically, a preliminary grouping of the minerals is accomplished by the chemical Composition whereby the mineral formulae sometimes have to be idealised.

Carbonates

Calcite	$CaCO_3$ trigonal	) polymorphic
Aragonite	$CaCO_3$ orthorombic	)

Aragonite and calcite are polymorphic modifications with an identical chemical Composition but with different crystal lattices. Aragonite is formed in the warm and shallow sea. It is unstable and transforms readily to calcite and can thus be found only in recently deposited unconsolidated sediments.

Other carbonates are:

Magnesite	$MgCO_3$
Siderite	$FeCO_3$
Dolomite	$CaMg(CO_3)_2$
Ankerite	$Ca(Mg, Fe)(CO_3)_2$

Calcium sometimes may be substituted by significant amounts of other elements such as Mg, Mn and Fe; for example: Mg-bearing calcite $Ca_{1-x}Mg_xCO_3$ (not identical with dolomite).

Carbonates have an excellent cleavage and a Mohs-hardness of 3 - 4.

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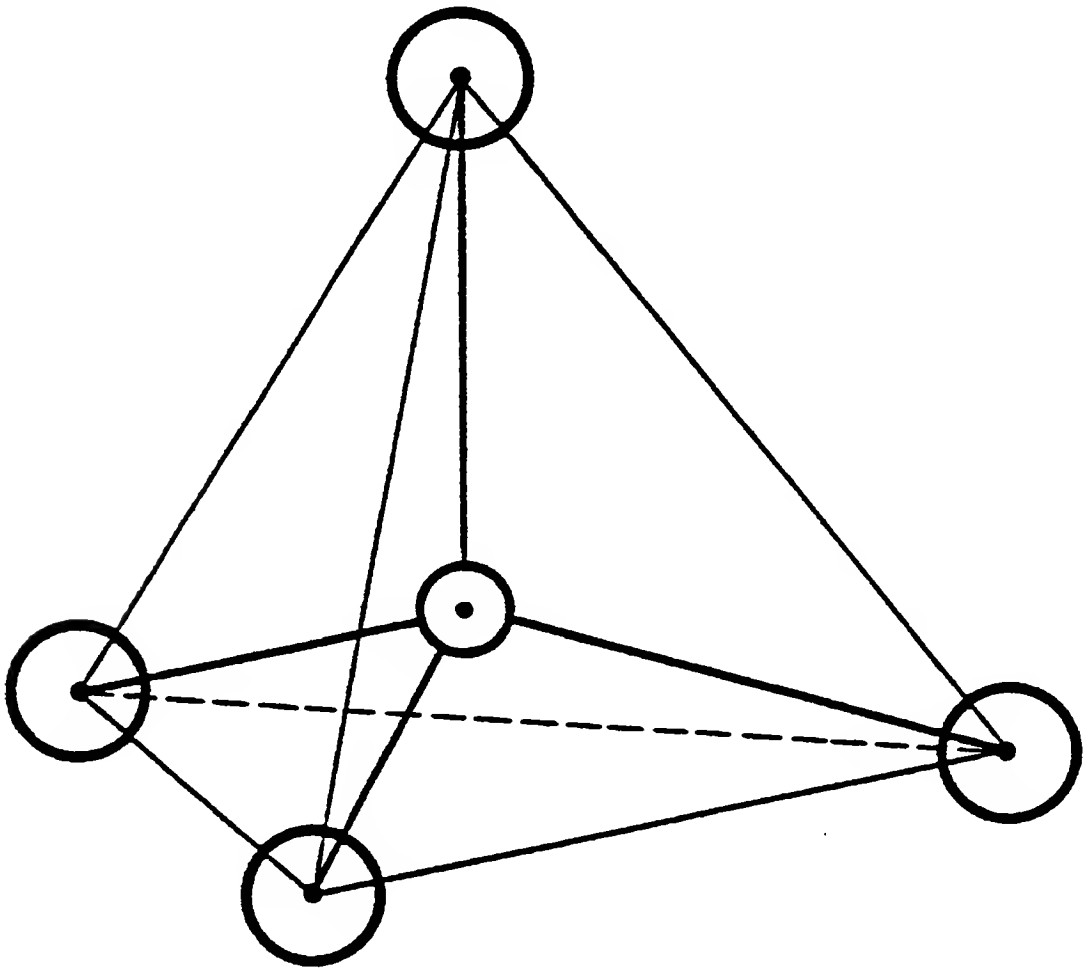
Silicates

The lattice of all silicates are based on the SiO_4-tetrahedron (Fig. 42). One Si-atom is situated in the centre of the tetrahedron, whereas the four corners are occupied by oxygen atoms.

Fig. 42 Structure unit of Silicates: SiO_4-Tetrahedron.



SiO_4-tetrahedra can be combined in different ways (Fig. 43 - 46) leading to different silicate types:

single tetrahedron	- nesosilicates :	olivine and garnet groups
double tetrahedra	- sorosilicates. :	epidote group
rings	- cyclosilicates :	tourmaline, beryll
single and double chains	- inosilicates	pyroxene and amphibole groups
sheets	- phyllosilicates:	mica and clay mineral groups
framework	- tectosilicates:	feldspar group

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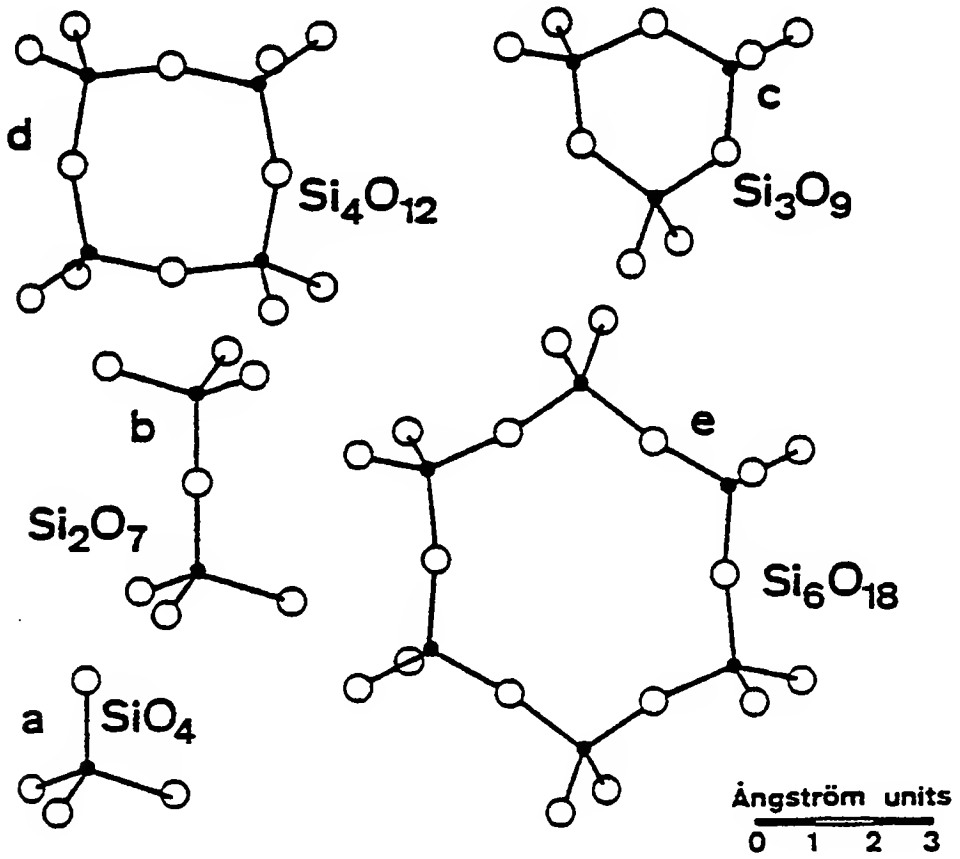
Fig. 43 Types of linkage of SiO_4-Tetrahedra (single/double/rings).



a) SINGLE TETRAHEDRON

b) DOUBLE TETRAHEDRA

c), d), e) RING STRUCTURES

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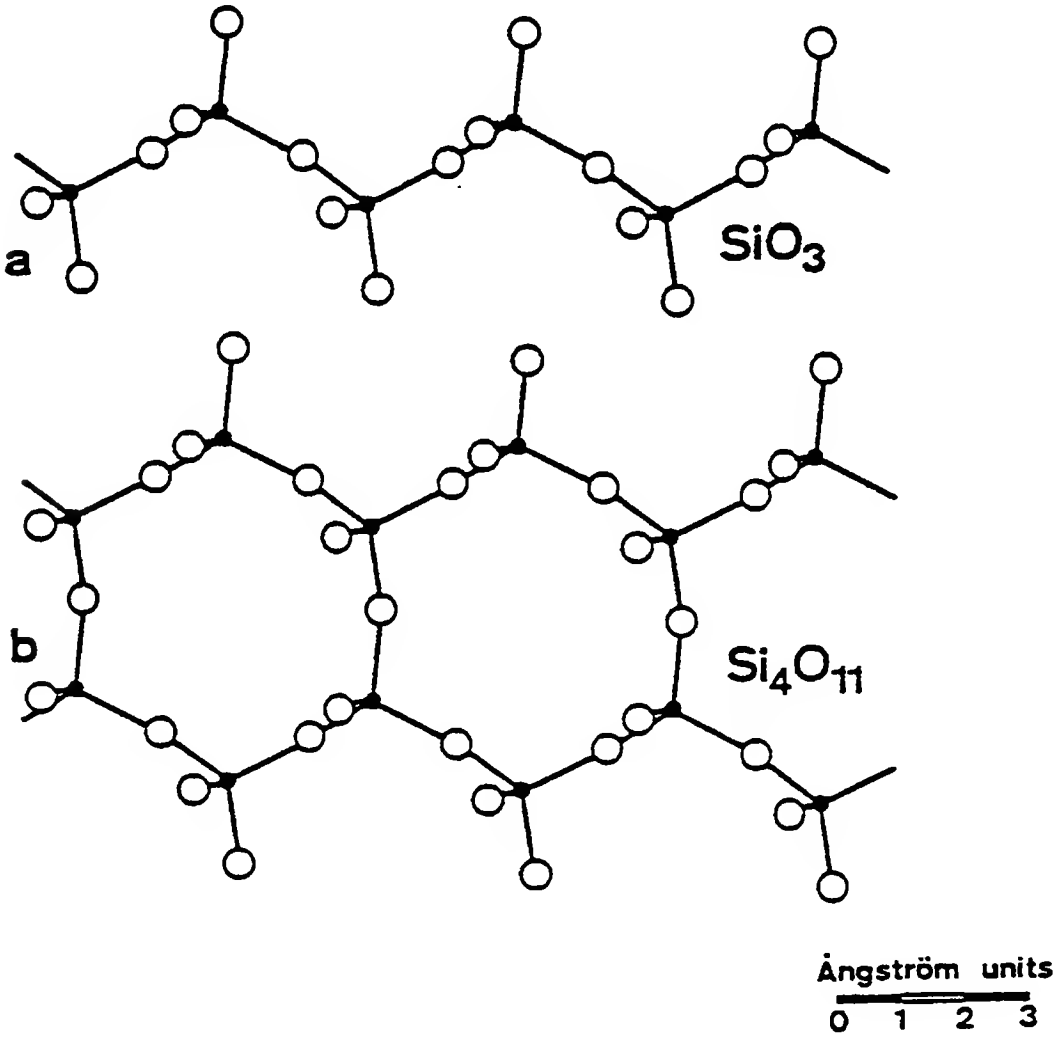
Fig. 44 Types of linkage of SiO_4-Tetrahedra (chains - single/double).



a) SINGLE CHAIN STRUCTURE

b) DOUBLE CHAIN STRUCTURE

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Fig. 45 Types of linkage of SiO_4-Tetrahedra (Sheets).

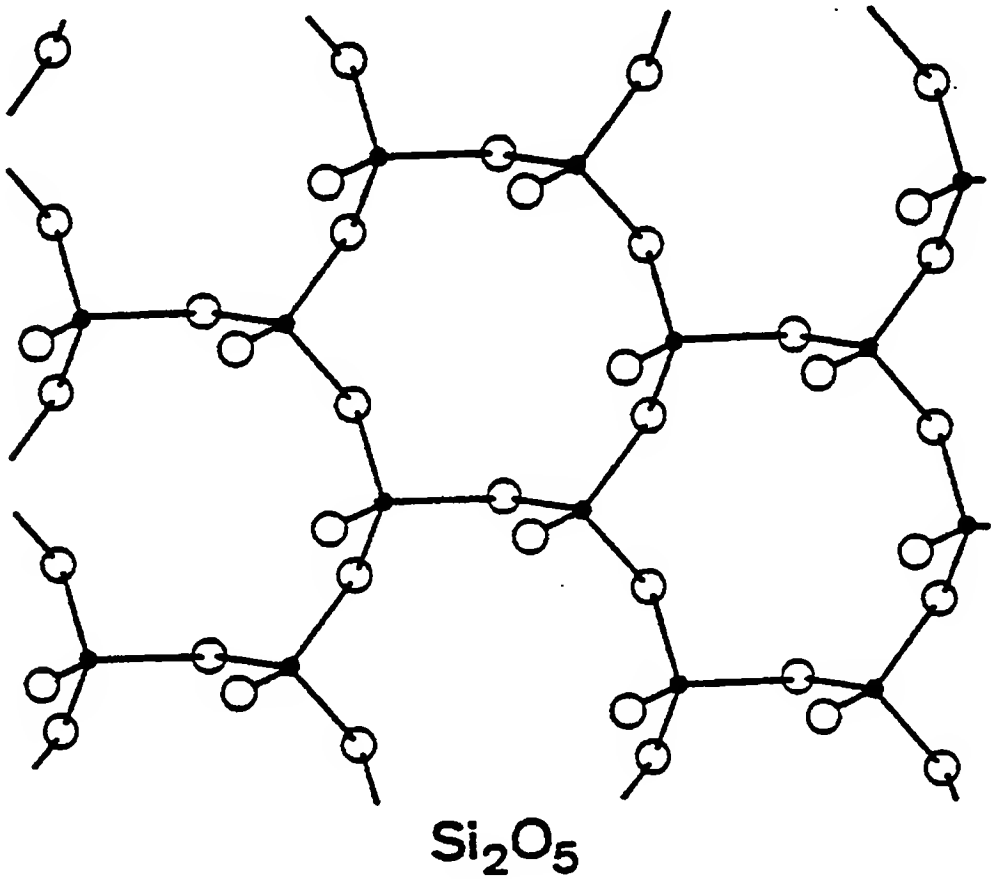
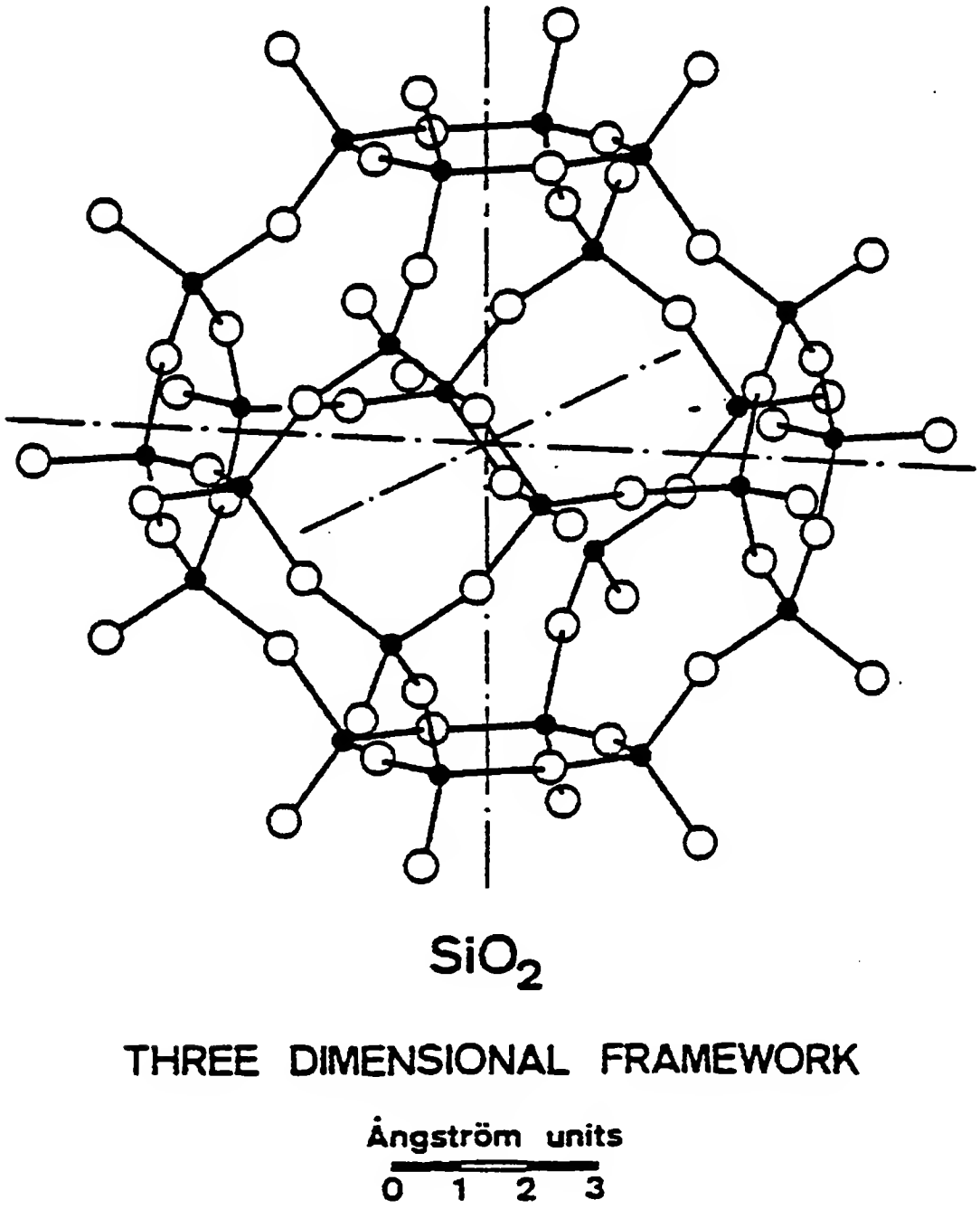


Fig. 46 Types of linkage of SiO_4-Tetrahedra (Framework).



The fact that so many possibilities to combine the structural unit exist, may explain why the silicate group contains a large number of different minerals (about 500). The Si of the SiO_4-tetrahedron may be replaced by Al; and nearly all silicate structures show the ability of exchanging cations (Na, K, Ca, Mg, Fe, Al, etc.). Within the structure of many of the resulting lattices, there is free space left for different cations (e.g. Na, K, Ca, Al, Mg).

The amount of silicates in the earth's crust is estimated to be about 90 %.

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As indicated on page 4/36, the group of silicates consists of several structural types such as:

Feldspar Group

Feldspars are the most abundant of all minerals. They occur in almost all siliceous components used in cement raw mixes. By alteration and weathering feldspars change into clay minerals.

Most abundant feldspars are:

Alkali feldspar series (K, Na) $AlSi_3O_8$	orthoclase	$KAlSi_3O_8$
	Albite	$NaAlSi_3O_8$
Plagioclase series (Na, Ca)(Si, Al)$AlSi_2O_8$	Anorthite	$CaAl_2Si_2O_8$

All feldspars show good cleavage and a Mohs-hardness of 6.

Clay Minerals and Micas

Clay minerals usually are the most important siliceous compounds of cement raw mixes. Crystal lattices of clay minerals can be characterised as shown in Figs. 47 to 49.

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Fig. 47 Clay Mineral Structures (two-layer structures).

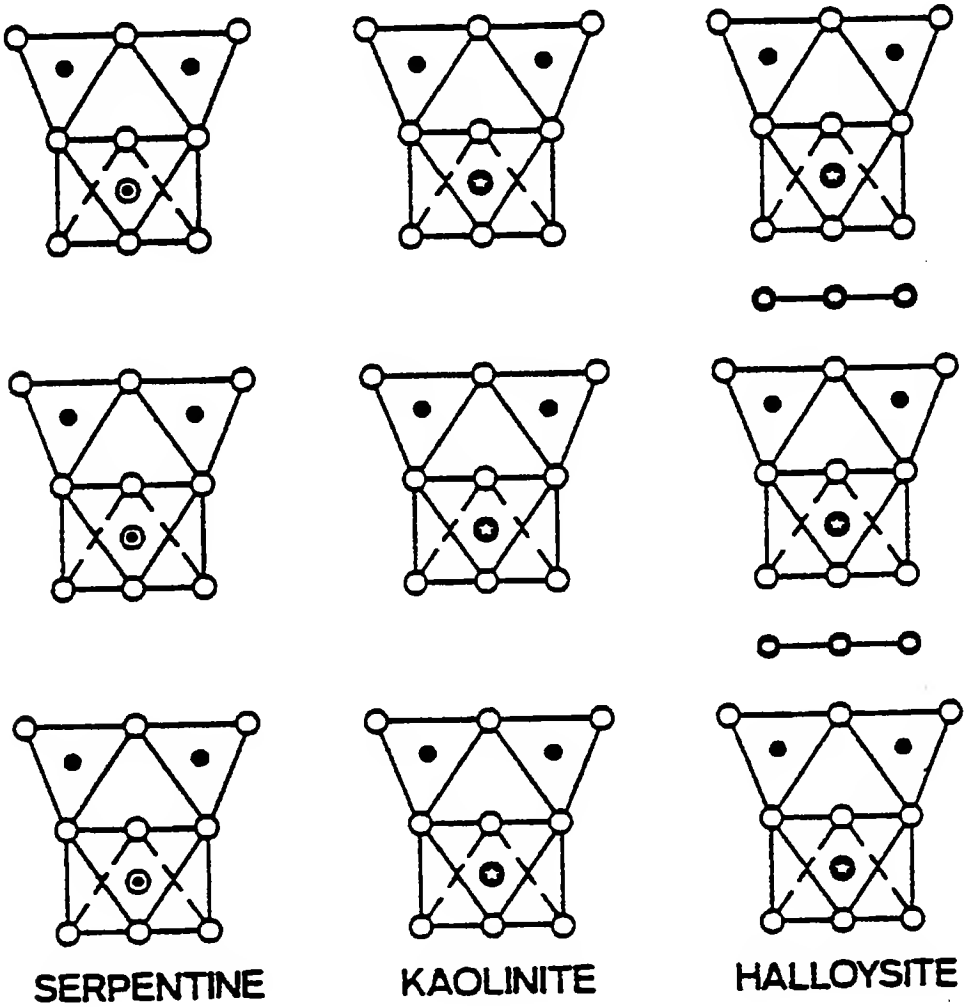

Fig. 48 Clay Mineral Structures (three-layer structures).

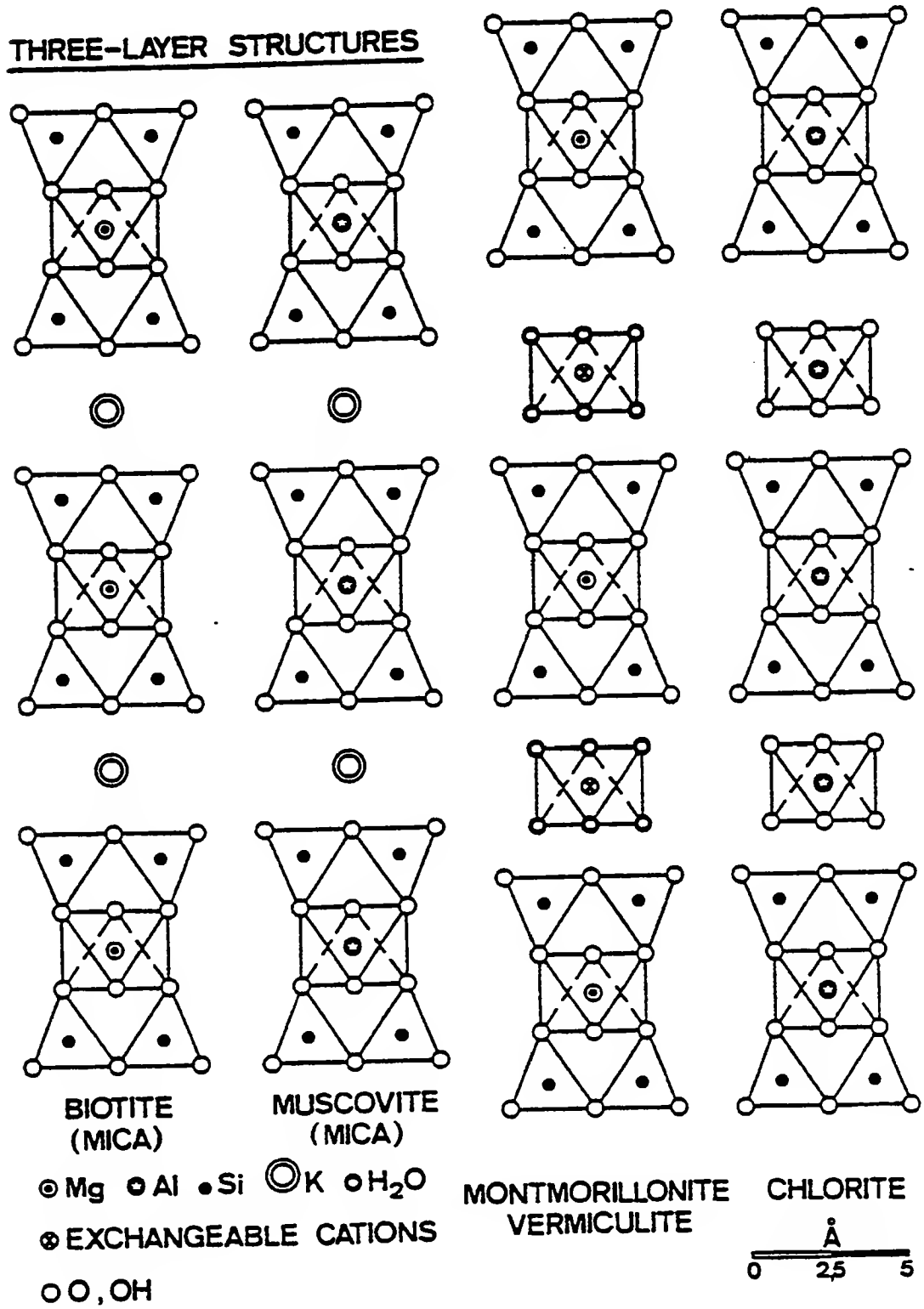

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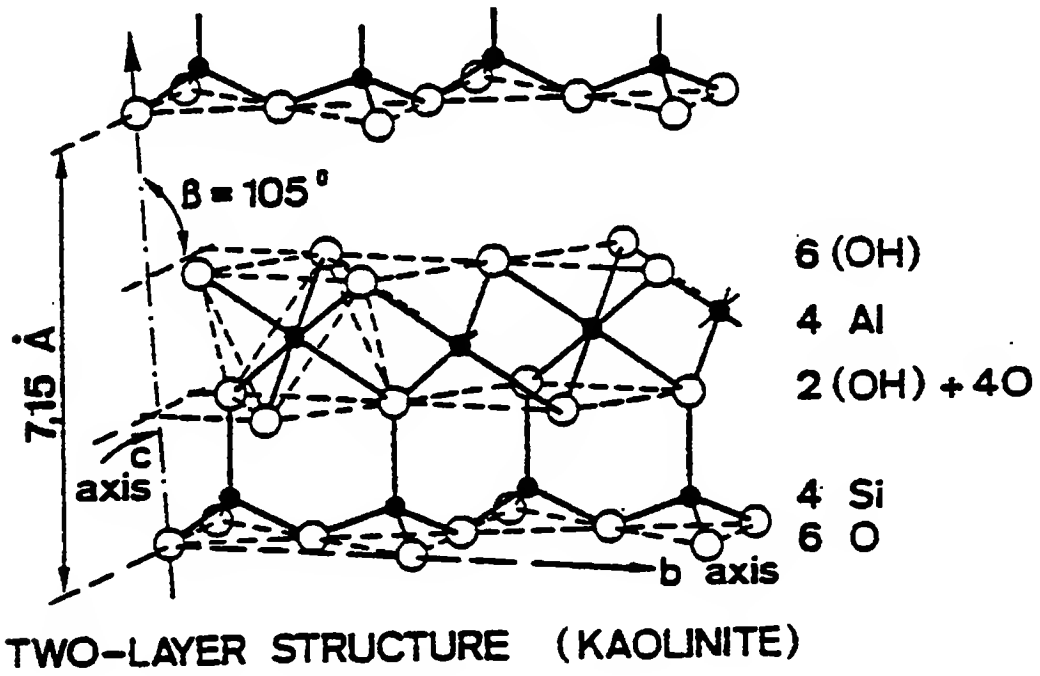
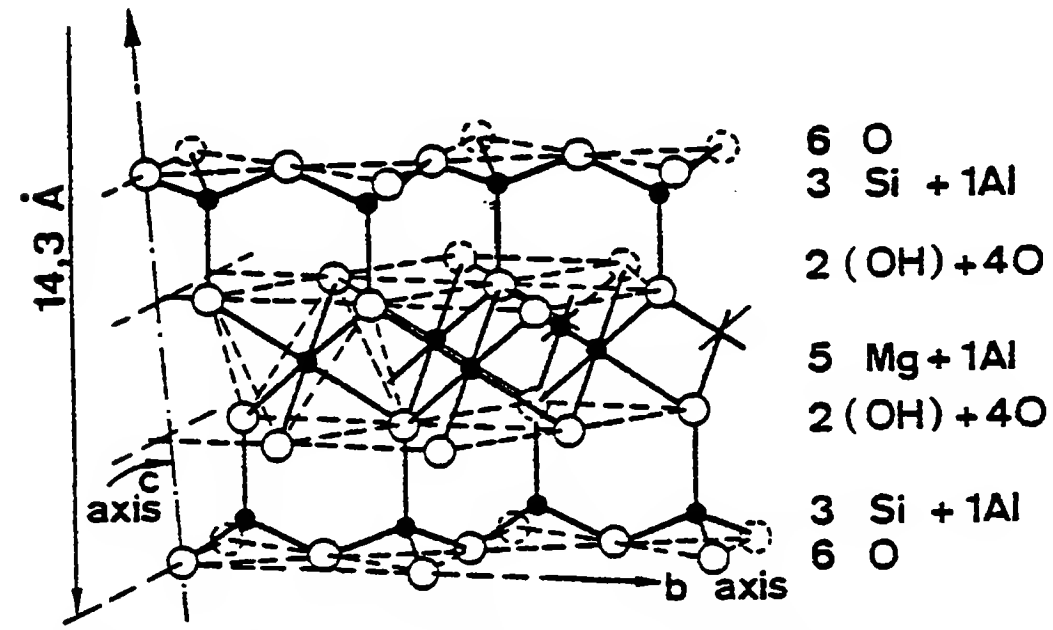
Fig. 49 Clay Mineral Structures (two and three-layer structures).



TWO-LAYER STRUCTURE (KAOLINITE)



THREE-LAYER STRUCTURE (CHLORITE)

The main structure unit is again the SiO_4-tetrahedron. In addition, a new unit having the shape of an octahedron appears. The centre of the octahedron is occupied by a cation (e.g. Al, Mg), the corners by O or OH-Groups (structurally bound water). The structure of a clay mineral may contain layers of Tetrahedra and octahedra and combinations of both structural elements. The dimensions of the tetrahedral and the octahedra layers are very similar. Composite tetrahedral-octahedral layers are readily formed leading to a two-layer structure. Similarly, an octahedral layer sandwiched between two tetrahedral layers forms a three-layer structure. Between these two-layer or three-layer packages different cations and/or H_20-molecules can be situated.

The many possibilities of combining the structural units are responsible for the existence of numerous clay minerals.

The macro and microscopic appearance of clay minerals corresponds to the internal layer structure (phyllosilicates). Properties which are very typical for clay minerals are explained by the characteristics of their crystal lattice.

Relevant features of clay minerals are:

- Water adsorption capability : the water is absorbed in three different ways:
  - in pores between single crystals and crystal Aggregates
  - directly on the surface as "adsorptive water" due to the dipole character of the H20-molecule; this water particularly causes clay mineral properties such as plasticity and thixotropy.
  - as "interlayer water" between the two- or three-layer packages in the crystal lattice, causing expansion of clay minerals.
- Capability of ion exchange : the ions placed between the layers of the crystal lattice e.g. Na^{+}" K^{+}, NE^{+}, SO_4^{-}, Cl^{-}, are easily exchangeable.
- Thermal behaviour : of special interest are the temperatures of loss of water. Generally, pore water, adsorbed water and interlayer water are set free at temperatures between 100 ° and 300 +C. The structurally bound water (OH-groups) is driven off in the range of 400º - 900ºC. Depending on the type of clay mine -1, the loss of the 'water types" mentioned above occurs at defined temperatures or continuously in a certain temperature range.

Important clay minerals and micas (see also Fig. 47 to 49) are:

Kaolinite group:

Kaolinite $Al_2Si_2O_5\ (OH)_4$
Halloysite $Al_2Si_2O_5\ (H_2O)_4$
low plasticity, without inlayer cations

Palygorskite:

$Mg_{2.5}\ (H_2O)_2(CH)Si_4O_{10}\ .\ 2\ H_2O$
properties partially similar to montmorillonite and vermiculite with ability to take up and retain large quantities of water

Montmorillonite:

$Al_2Si_4O_{10}\ (OH)_2$ - x H_2O
varieties with Na, K, Fe, Mg, etc., especially capable of exchanging ions, large content of interlayer water possible, optimum plasticity therefore with water contents from 80 % up to 250 %, thixotropic behaviour of clay suspensions or slurries, capability of expansion

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Vermiculite:

$Mg_3Si_4O_{10}(OH)_2$ - x H_2O
properties partially similar to those of montmorillonite, shows rapid and large expansion on heating.

Illite (Hydromica):

$(K, H_2O)Al_2AlSi_3O_{10}$ $(H_2O, OH)_2$ usually poorly crystallised, plasticity can be high depending on degree of crystallinity.

Micas:

- Biotite
  $K(Mg, Fe)_3 AlSi_3O_{10}$ $(OH)_2$
- Muscovite
  $KAl_2AlSi_3O_{10}$ $(OH)_2$ usually well crystallised, no plasticity.

Chlorite series:

$(Mg, Fe, AL)_6$ $(Al, Si)_4O_{10}$ $(OH)_8$

Mixed Layers are more or less regular combination of lattices of different clay minerals, e.g. of illite-montmorillonite by combination of two- and three-layer structures. The properties are usually close to those of montmorillonite.

Silica

SiO_2 occurs in several polymorphic forms (see 4.3.2.7) each of them being stable in different temperature ranges under atmospheric pressure:

Low-Quartz	< 573°	triconal
High-Quartz	573° - 870°c	hexagonal
Tridymite	870° - 147° c	hexagonal
Cristobalite	1470°- 1715°c	isometric
Liquid	>1715° c	

Amongst the crystalline forms of SiO_2, low-quartz is the most abundant modification with a Mohs-hardness of 7, without cleavage and with conchoidal fracture. With the transformation of quartz to tridymite and cristobalite, an increase in volume occurs. SiO_2 also exists in a more or less amorphous modification, which has been formed by gradual dehydration of water containing silica-gel. This modification is known as:

Opal or Silex: SiO_2 . x H_2O with Mohs-hardness of 5½ - 6½

SiO_2 also occurs in almost sub-microscopic and fibrous varieties which are known as:

- Chalcedony
- Agate
- Jasper
- Chert
- Flintstone

Because of their fibrous structure, these varieties are very hard and tough.

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Hydroxides

Hydroxides are formed at the earth's surface by weathering action under humid climatic conditions. They occur in many sedimentary rocks. Common hydroxides are:

- Gibbsite (Hydrargillite) $Al(OH)_3$
- Diaspore, Boehmite AlOOH
- Goethite, Lepidocrocite FeOOH
- Limonite $FeOOH - n\ H_2O$

These minerals usually have a low degree of crystallinity and are quite reactive.

Oxides

Some of the minerals of this group are present as minor components in many rocks. They predominate in certain ores which sometimes are used as correctives in cement raw mixes.

Abundant oxides are:

- Magnetite Fe_3O_4
- Haematite Fe_2O_3
- Ilmenite $FeTiO_3$

Sulphides

The most abundant sulphide is:

- Pyrite FeS_2

which is frequently employed as Fe-corrective after roasting to oxide (pyrite ash or pyrite cinders).

Sulphates

Two naturally occurring sulphates are important:

- Gypsum $CaSO_4 . 2\ H_20$
- Anhydrite $CaSO_4$

They are added to the clinker to produce cement.

1.3.4 Rock Characteristics

1.3.4.1 Definition

Rocks are combinations of one or more kinds of minerals being stable under given conditions of pressure and temperature.

Equilibrium conditions exist between the minerals of a rock as well with respect to the surrounding of the rock (conditions of low free energy). If the conditions change for some reason, reactions towards a new equilibrium occur. This is possible in the system itself by reactions between minerals present in the rock or by reactions with agents introduced to the system from outside.

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In nature, reactions of this kind are involved in rock metamorphosis and weathering. Metamorphosis can be schematised as follows:

Pressure	P_1	H_2O	Pressure	P_2
Temperature	T_1	CO_2	Temperature	T_2
		↑		
Rock	A	⇒	Rock	B
Minerals	a, b, c		Minerals	a, b, e, f
Chemical Composition I			Chemical Composition II	

In the case of rock weathering, agents from outside are added to the system (water, air);

Pressure	P_1	H_2O	Pressure	P_3
Temperature	T_1	O_2	Temperature	T_3
		↓		
Rock	A	⇒	Rock	C
Minerals	a, b, c		Minerals	a, b, e, f
Chemical Composition I			Chemical Composition II	

The process of clinker formation can be compared, to some extent, with rock metamorphosis.

1.3.4.2 Properties of Rocks

The more important rock characteristics to be considered in the course of cement production are:

- chemical composition
- type and condition of minerals building up the rock
- grain size, distribution and geometrical Arrangement of the minerals (or of the rock fragments) which can appear as rock compounds
- as well. These features are summarised by the expression "rack texture'.
- cleavage
- degree of consolidation

1.3.5 Physical Characteristics; Engineering Properties of Rocks

The physical properties of rocks are as important as the chemical and mineralogical characteristics because they are determining factors in the choice of method and technique of raw material exploitation, transportation, blending and comminution. They therefore affect the manufacturing process.

1.3.5.1 Density and Porosity

The term "specific weight" or density is used for the weight of a unit of volume of a specimen, dried at 105°C, excluding all pores.

The term "specific gravity" includes the-volume of the pores per unit of volume of a specimen (Table 32).

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"Bulk density" refers to comminuted material, i.e. means weight per unit of volume.

"Porosity" or "pore volume" expresses the hollow space of pores and voids in percent. Porosity (Table 32) as:

$$\frac{\text{volume of pores}}{\text{total volume}}$$

With regard to the sedimentary rocks, porosity depends largely on the amount of cementing material present and the size, grading and packing of the granular constituents.

Other physical properties (e.g. strength) also depend on the porosity.

Table 32 Density and porosity of rocks

Rock	Specific gravity (density) g/cm	Porosity %
Granite	2.5 -2.8	0.5 - 1.5
Diorite	2.6 -2.9	0.1 - 0.5
Gabbro	3.0 -3.1	0.1 - 0.2
Dolerite	3.0 -3.1	0.1 - 0.5
Rhyolite	2.4 -2.6	4 - 6
Andesite	2.2 -2.3	10 - 15
Basalt	2.8 -2.9	0.1 - 0.2
Tuff	2.8 -3.0	high
Sand (loose)	1.5 -2.0	25 - 50
Sandstone (compacted)	2.0 -2.6	10 - 30
Shale	2.0 -2.4	10 - 30
Limestone	2.2 -2.6	5 - 20
Dolomite	2.5 -2.6	1 - 5
Marl	1.6 -2.6	5 - 30
Clay	1.5 -1.9	30 - 50
Claystone	1.8 -2.3	15 - 35
Gneiss	2.9 -3.0	0.5 - 1.5
Marble	2.6 -2.7	0.5 - 2
Quartzite	2.6 -2.7	0.1 - 0.5
Slate	2.6 -2.7	0.1 - 0.5

1.3.5.2 Water Absorption

Phenomena, such as consistency, plasticity, swelling, etc. are functions of porosity or water absorption. Loose material absorbs considerably higher quantities of water than compact material. The water absorption capability is increased significantly in the presence of clay minerals. Three kinds of absorption are distinguished: pore water, water accumulation on clay mineral surfaces ("adsorption water"), and water penetrating the clay mineral structure ("structural water"). The latter kind may lead to swelling. Water absorption of clay minerals increases

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from	illite	high crystallinity
	Kaolinite	high crystallinity
⇓	Kaolinite	low crystallinity
	illite	low crystallinity
to	Montmorillonite	

1.3.5.3 Rock Strength

Compressive strength increases with:

- increasing content of minerals with high compressive strength potentials
- increasing content of cementing matter (increasing degree of intergrowth of minerals)
- decreasing grain size

and decreases with:

- increasing amount of minerals with good cleavage
- increasing porosity
- increasing content of micro-joints
- increasing degree of weathering
- increasing moisture absorption capability

Table 33: shows same compressive strength for different rocks.

Table 33 Compressive strength of rocks

Rock type	Compressive strength kg/cm^2 / N/mm^2	Valuation (of compressive strength)
Basalt Quartzite	> 2800 / > 280	very high
Granite Diorite Limestone	1800 - 2800 / 180 - 280	high
Sandstone Limestone Marble Slate	800 - 1800 / 80 - 180	medium
Porous Limestone	400 - 800 / 40 - 80	low
Chalk, Tuff, Shale, Clay, Claystone	< 400 - < 40	very low

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1.3.5.4 Rock Hardness

There is no absolute Definition of the term "rock hardness". Today, the terms "hard rock" or "soft rock" are occasionally used in practical fieldwork. Rock hardness is not only determined by the mineral content but also by the degree of intergrowth of the individual minerals. It is commonly expressed by the compressive strength.

The mineral hardness can be measured by means of the scratch method according to Mohs (Tables 34 and 35).

Table 34 Hardness of most abundant minerals according to Mohs

Mineral	Mohs scale	Valuation
Quartz	7	
Feldspar	6	"hard"
Pyroxene	6	minerals
Amphibole	6	
Calcite	3	
Dolomite	3.5	"soft"
Mica	3	minerals
Chlorite	2-3	
Clay minerals	2	

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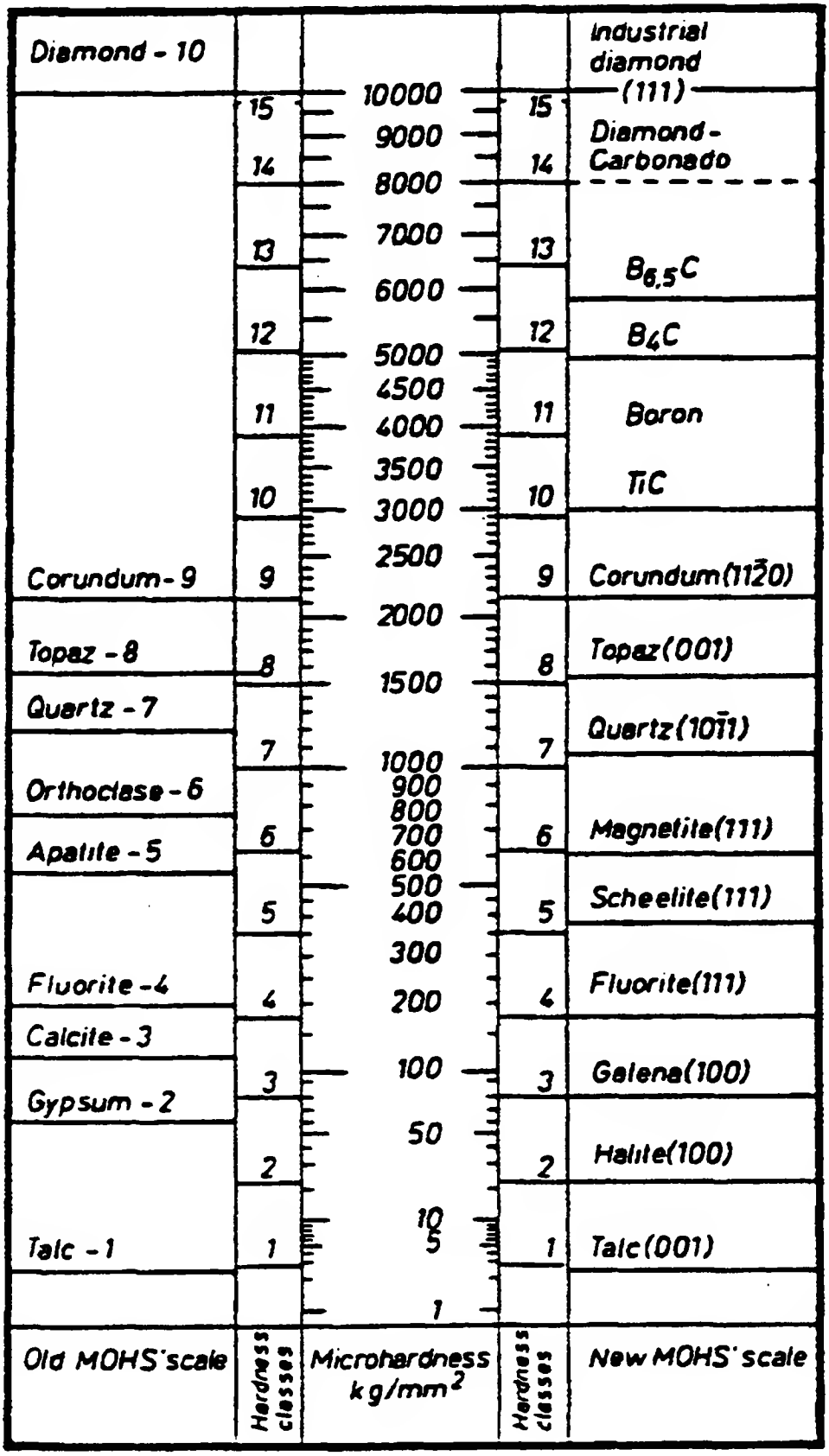
Table 35 Hardness Scale



1.4 PRINCIPLES OF RAW MATERIAL ASSESSMENT

1.4.1 Chemical Parameters

A check-list for the assessment of a Potential cement raw material is given in Table 36, comprising basic chemical features.

Table 36 Check-list for potential cement raw material (chemical)

Components	Parameters to be checked	Limits (guide lines)
Calcareous materials (limestone, etc.)	Carbonate titration as $CaCO_3$	> 80 %
	MgO content	< 4 %
	SO_3 content	< 1 %
	P_2O_5 content	< 1 %
	Cl	< 0.1 % *
Argillaceous materials (clay-marl)	Total alkali as Na_2O equivalent (Na_2O + 0.66 K_2O)	depending on cement type
	Al_2O_3 content	> 10 %
	SO_3 content	< 1 %
	CL content	< 0.1 %
Correctives for aluminium (bauxite, laterite) for iron ore (pyrites ash) for silicon (quartz sand) for calcium (high-grade limestone)		> 25 % (Tab. 22)
	Al_2O_3 content	> 60%
	Fe_2O_3 content	< 5 %
	SO_3 content	>80 %
	SiO_2 content	> 90%
	$CaCO_3$ content	< 2 %
	MgO	
Additives (tab. 23) (gypsum)	SO_3 content	> 35 %

* limit variable according to process

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In addition, the mineralogical characteristics have to be checked (Table 37).

Table 37 Check-list for Potential cement raw materials (mineralogical)

Components	Parameters to be checked	Limits (guide lines)
Calcareous materials (limestone, etc.)	Calcite Aragonite Dolomite	> 80% < 5% < 5%
Argillaceous materials (clay, marl)	Clay minerals	> 30%

1.4.2 Deleterious Constituents in Cement Raw Materials

"Deleterious constituents" are defined as components which:

- adversely affect the quality or marketability of cement
- appreciably increase production costs through difficulties in exploitation, reparation and processing in the kiln

In fact, the limits established in Tables 36 and 37 are already referring to these constituents. The major deleterious constituents (if occurring in higher amounts) are K_20 and Na_2O, MgO, SO_3, P_2O_5, CL, etc. In addition to these oxides, there are other, less important, deleterious constituents including Cr_2O_3, Mn_2O_3, F, etc.

They are always present in small amounts and are usually not considered for the mix calculation. If the concentration of certain minor and trace elements becomes too high, components otherwise suitable for cement manufacturing have to be rejected or limited in amount. Suddenly occurring difficulties in a plant very often are caused by an increase of the content of some minor and trace elements in the raw mix. Most minor and trace elements can have a deleterious effect on the production process as well as on the cement quality. On the other hand, it is known that a small amount of such elements is necessary to stabilise certain modifications of clinker minerals which are preferred to be present in an industrial clinker. The maximum content of some of these elements having a well known deleterious influence on the cement quality is limited by the specifications (e.g. MgO).

The following description gives a summarised survey of the most important minor elements and of the influence they have:

Alkalis

The term 'alkalis" as used in the cement industry" refers to the oxides of potassium (K_20) and sodium (Na_2O). Alkalis occur mainly in minerals (Tables 38, 39) which are components of igneous rocks or their weathered products, such as clay and sandstone.

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Table 38 Alkali sources in cement raw Materials

Rock type	minerals	Deleterious	constituents
Igneous rocks			
Granite	Feldspars:		
Rhyolite	Albite	11.8 %	Na_2O
Diorite	orthoclase	16.9 %	K_2O
	Andesite	5.9 -8.3 %	Na_2O
Lava	Micas:		
	Muscovite	1.8 %	K_2O
	Biotite	10 -11 %	K_2O
Tuff			
Sediments			
Clay	Feldspars	as above	
Shale	Micas	10 - 11.8 %	K_2O
Sandstone (marl)	Illite	5.5 %	K_2O

Alkalis may cause problems particularly in the kiln section, since they are volatilized quite easily. Together with other volatile constituents, such as chlorine and sulphur, they may cause build-ups in the form of rings, cloggings, etc. in kilns and preheaters. Furthermore, an alkali content of > 0.6% (total alkali as Na_2O equivalent) may react with the amorphous silica present in some concrete aggregates producing alkali silica gels which have a great affinity for water. The resulting osmotic pressures may cause disruption of concrete. On the other hand, the initial strength development of concrete.

MgO

MgO is predominantly supplied by carbonatic raw materials such as dolomite and dolomite limestone, but may also by present in argillaceous materials. Crystalline MgO, in the form of periclase present in cement, tends to hydrate after setting, a condition which causes excessive expansion and possible disruption of concrete. For this reason, MgO is limited to 5 - 6 % in cement. However, the activity of periclase is also a function of the cooling rate of the cement clinker: slow cooling allows a large migration of ions and the subsequent development of large periclase crystals; rapid cooling results in an significant proportion of the MgO being frozen in various inactive solid solution.

SO_3

SO_3 is generally associated with gypsiferous carbonate sources or with pyrite-containing materials. SO_3 inhibits the volatilisation of the alkalis. It also requires - in form of SO_2 - special attention with respect to air pollution. Furthermore, SO_3 (as gypsum) above a certain optimum percentage causes a decrease in strength and an increase in expansion.

P_2O_5

P_2O_5 is mainly found in carbonate components. Contents of >1 % P_2O_5 in cement reduce the cement strength by raising the free lime content and combining with CaO in solid solution at the expense of the C_3S. Setting times are considerably affected as well.

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Chloride (Cl)

Chloride is mainly supplied by sedimentary rocks containing or contaminated by salt. In wet preparation systems chloride attacks all metal installations; during burning it easily volatilizes and causes clogging and plugging in kilns and preheaters. Chloride in concrete jeopardises reinforcement by corroding.

Fluorine (F)

Fluorspar (CaF_2) was in the past added to many cement raw mixes as a fluxing agent. Although this material decreases the kiln reaction temperature, several undesirable effects - as deterioration of refractory lining in kilns, increase of dusting of the clinker through a tendency to convert β-C_2S to the γ-form, difficulties with setting times and a lowering of early strengths - led to a considerable reduction of consumption in the cement industry. Aside from those reasons, recent air pollution regulations prohibit the introduction of fluorides into the atmosphere.

Mn_2O_3

Manganese occurs normally as a constituent of corrective materials for iron deficiency. The presence of larger quantities (> 2 %) may lead to a dark coloured clinker.

Cr_2O_3

Chromium bearing cement may lead to the "chromium ekzema"; on the other hand it is apparently possible to improve the cement strength to a certain extent by adding Cr_2O_3 - bearing components to the raw mix ("doping of alite").

It is furthermore claimed that Cr_2O_3 could also affect the colour of cement.

TiO_2

Titanium reacts as fluxing agent decreasing the alite content and increasing the belite content in clinker. It produces a darker cement colour, increases strength and retards initial setting.

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Tab. 39 summarises the influence of some minor components.

Table 39 Mineralogy of major deleterious constituents in cement raw materials.

	$K_2O + Na_2O$		MgO		SO_3		P_2O_5		SiO_2
	Mineral	Alkali content	Mineral	MgO content	Mineral	SO_3 content	Mineral	P_2O_5 content	Mineral
	Feldspars								
	Albite	11.8 % Na_2O	Dolomite	21.7 %	Gypsum	46.6 %	Collophane	15-40 %	Chert
	Oligoclase	8.3-10.6 % Na_2O	Magnesite	47.6 %	Anhydrite	58.8 %	Apatite	41-42 %	Chalcedony
	Andesine	5.9- 8.3 % Na_2O	Periclase	100.0 %	Pyrite	53.4 %			Quartz
	Labradorite	3.5- 5.9 % Na_2O	Brucite	69.0 %					Opal
	Microcline	16.9 % K_2O	Montmorillonite	2.5-6.5 %					
	Orthoclase	16.9 % K_2O	Chlorite	30-37 %					
	Mica - Illite		Amphiboles, proxenes, micas	15-30 %					
	Biotite	10.0-11.0 % K_2O							
	Muscovite	11.8 % K_2O							
	Illite	5.5 % K_2O							
Reason for deleterious effect	Causes deterioration in concrete when combined with reactive aggregate		Expansion caused hydration of MgO to brucite-$Mg(OH)_2$		Inhibits volatilization of alkalis		Decreases early strength through reduction of C_3S		Increased mining, milling, and fuel costs
Limit in cement	0.6 % (Na_2O + .658 K_2O) for low-alkali cement		5.0 - 6.0 %		2.3 - 4.0 %		1.0 - 1.5 %		

1.4.3 Assessment of Physical Properties

1.4.3.1 General

The description of the relevant physical ("engineering") properties is given in 4.3.5. Basically, limits or guidelines for physical properties can hardly be given. In practice, appropriate process-technological methods have to be selected and adapted in order to permit optimum processing.

A series of physical properties and their effects on various stages of the cement manufacturing process are listed in Table 40. The influence of these properties is known only very generally; it is almost impossible to express them quantitatively, because only a few out of the total can be measured exactly.

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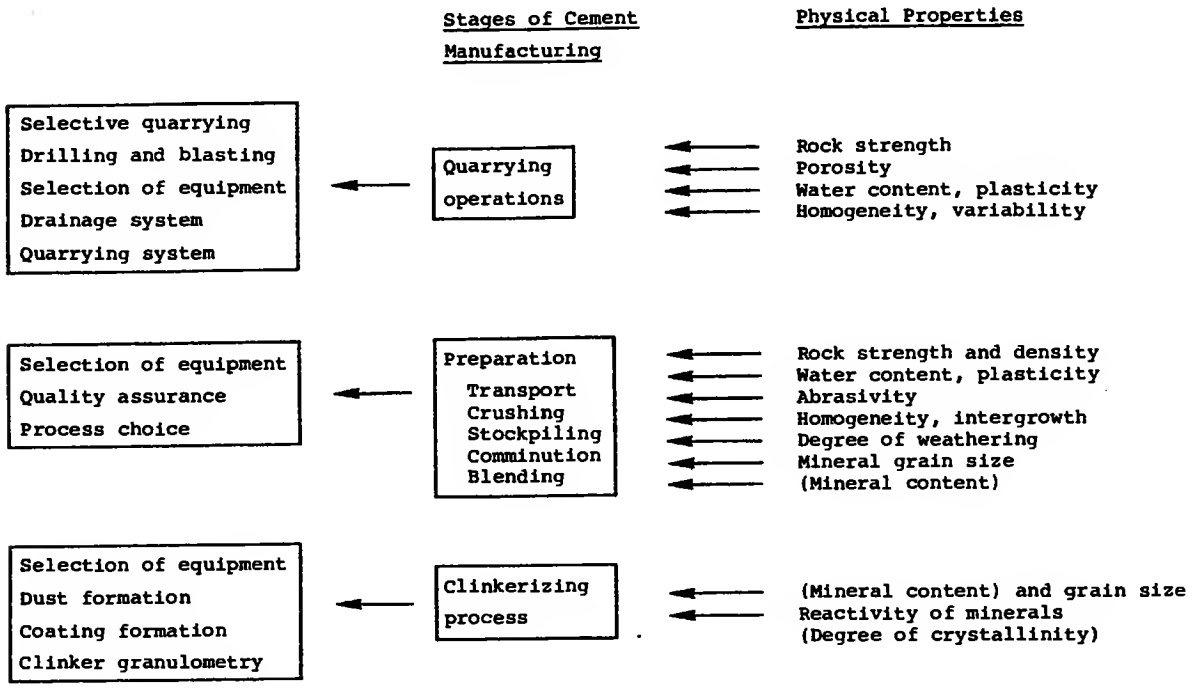
Table 40 Stages of Cement Manufacturing.



1.4.3.2 Examples of Practical Experience

Quarrying operations

The content of "hard" minerals is important. Quartz being harder than steel has an abrasive influence, mainly on drilling equipment. In cases of large quartz content and high rock strength, drilling will progress slowly and abrasion will be high. Such a rock has a high "rock hardness".

The water content of the material to be quarried depends partially on the minerals present. Clay minerals, especially montmorillonite, mixed layers, illite, absorb large amounts of water. Clays thus have a high plasticity, they become adhesive, with more water slurry-like and dry very slowly. Clogging of machinery results, and quarry drainage becomes a problem.

Preparation

For crushing the rock strength has to be considered in the first place. It is rather the impact resistance than the compressive strength which is important. Rocks having a high quartz content and a high compressive strength may behave as brittle Materials during crushing, whereas clay minerals containing rocks sometimes are better characterised to be elastic and thus more difficult to crush.

As in all disintegration processes the formation of dust is a severe problem. This is especially the case when quartz, feldspar and micas are present in large amounts (e.g. in siliceous and related rocks), whereas the tendency of dust formation is much lower with clay bearing rocks. Quartz dust of a certain fineness (<5 µ) is the main cause of silicosis, a disease that attacks people working for longer periods in a dust environment (quarries, galleries, crusher buildings).

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Again rock strength and mineral hardness play a determining role for grinding. In addition, the mineral grain size as compared to the raw meal fineness is influencing the grindability. Raw meal particles may either be mineral or rock fragments. It is assumed that coarser single minerals are disintegrated more easily, especially when the minerals have a good cleavage (e.g. calcite), than rock pieces consisting of different very finely grained and densely intergrown minerals. Quartz will contribute to the abrasion of the grinding equipment-, but can also take part in the disintegration of softer minerals. Thus a difference in mineral hardness can be an advantage.

Usually, the quartz content of coarser raw meal fractions is higher than in the fine fractions. This fact can be utilised in cases where a limestone with a too high quartz content, e.g. silex inclusions, has to be used.

Coating sometimes becomes a problem during grinding. Among many different reasons the properties of micas and clay minerals - flaky shape, water content, plasticity - possibly contribute to this problem. From experience it is known that coating also may appear when aragonite is present in the raw material instead of calcite.

For raw mix preparation, homogeneity and variability of the raw mix components are of paramount importance. Specific physical parameters become impotent when the most suitable process has to be selected. The wet process is considered when the raw Materials have a high water content. That can be especially the case in the presence of clay minerals such as montmorillonite and illite absorbing large amounts of water. The viscosity of a slurry depends strongly on the type of clay mineral present. If montmorillonite and other clay of high plasticity are predominant, much more water is needed to get a slurry of sufficient flow properties.

To produce nodules of sufficient strength for the Lepol process, plastic clay minerals have to be present. Raw meals containing mainly quartz, feldspar and mica are hardly nodulizable with a reasonable water content.

In the case of a dry kiln there are no severe restrictions to be mentioned in this stage of the process. The question is always how much energy will have to be needed for raw material drying.

The clinkering process

The reactivity of raw mix depends on the intensity of the contact between calcareous and siliceous components which have to react together. In an argillacecus limestone both components are tightly intergrown within the rock texture itself. If at the same time the mineral grain size is well below the average grain size of the raw real, each particle contains both components needed for reaction. This results in a high reactivity which is not greatly influenced by the raw meal fineness. If the calcareous and the siliceous components occur in different rocks or in coarse grained minerals, a much finer grinding and a very effective homogenisation are essential for a goad reactivity.

Water bearing clay minerals react more readily than water free silicates such as quartz, feldspar and mica, because their mineral lattices are already partially destroyed when the carbonate starts to dissociate at about 600 - 700°C and therefore in a more reactive condition.

Poorly crystallised clay minerals and gel-like hydroxides having a low degree of crystallinity have a higher reactivity than minerals with well defined crystal lattices. The same is true for amorphous SiO_2 (i.e. silex, opal) as compared with quartz.

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The exact knowledge of the mineralogical raw mix composition shows in which form minor elements occur. Feldspar, mica and illite e.g. are the principal alkali bearing minerals. From an illite raw mix the alkalis evaporate at lower temperatures than from a mix rich in feldspar. Under certain conditions this fact should be considered in practice.

The tendency to form dust depends also on the mineralogy. Mixes with water free and well crystallised silicates tend to develop more dust than clay mineral containing mixes.

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Jura Wildegg
Limestone Quarry

Drill-rig

Make	Hausherr HBM 12
Drive	Hydraulic
Feed rate (m/min)	0 - 6.0
Drilling rate (m/hr)	30.0
Bench height (m)	5.0 - 25.0
Burden (m)	3.6
Spacing	3.0

Drill bit - 2 stage

Centre (mm)	65
Reamer (mm)	80

Mobile Crusher
Weserhütte

Apronfeeder

Apronfeeder - angle (°)	30
- width (m)	2.0
Installed Power (kW)	40
Velocity (m/s)	0.05 - 0.15

Mobile Crusher
Weserhütte

Double Shaft Hammer Crusher
Titan - Miag

Tonnes/hr nominal	500
Max feed size (mm)	1 500
Final product size (mm)	0 - 20
Grate size (mm)	25
Rotor dimensions (mm)	1 800 * 2 110
Installed power (kW)	2 * 400

Hammers

Number / row	17
Number / rotor	34
Weight (kg)	90

04:95:RP
CRUSHJW2

Jura Wildegg Cement
Marl Crusher
Hazemag Compound

	Marly Limestone	Marl
Tonnes/hr	650	650
Product Size (mm)	0-30	0-70
Max Feed Size (mm)	1 500	1 500
Moisture content (%)	13-15	
RPM Rotor 1	370 (36m/s)	360 (26 m/s)
Rotor 2	480 (45 m/s)	330 (35 m/s)
Power Cons. kW/t	0.5	0.4

04:95:RP
CRUSHJW1

ANNEX

ASSESSMENT OF RAW MATERIALS

ASSESSEMENT OF RAW MATERIALS

CRITERIA

. chemical composition to conform cement requirement

. handling and comminution properties

. burning behaviour

DETERMINED BY

. chemical composition

. mineralogical composition

. petrographical properties

COMPARISON
CHEMICAL ELEMENT REQUIREMENT CEMENT
V.S.
OFFER EARTH CRUST

ELEMENT	TYPICAL FOR CEMENT (%)	AVERAGE CONC. EARTH CRUST (%)
Ca	46	4
Si	10	28
Al	3	8
Fe	3	5

CHEMICAL COMPOSITION

MAIN ELEMENTS	MINOR ELEMENTS	TRACE ELEMENTS
SiO_2 (22%)	MgO	Tl
Al_2O_3 (5%)	SO_3	Cd
Fe_2O_3 (4%)	K_2O	Hg
CaO (64%)	Na_2O	etc
	TiO_2	
	Cr_2O_3	
	Mn_2O_3	
	P_2O_5	
(typical for OPC)	Cl	
	F	
	etc	

CLASSIFICATION OF CALCAREOUS COMPONENTS

$CaCO_3$	Designation	Petrographic desig.
95-100	high grade limestone	limestone
		marble
		chalk
		coral limestone
85-95	limestone	lime sand
		shell deposit
75-85	marly limestone	marly limestone

CLASSIFICATION OF ARGILLACEOUS COMP.

$CaCO_3$	Clay	Designation	Petr. design.
65-75	25-35	calcareous marl	calcareous marl
35-65	35-65	marl	marl
25-35	65-75	clayey marl	clayey marl
15-25	75-85	marly clay	clay loess
5-15	85-95	clay	claystone mudstone
0-5	95-100	high grade clay	siltstone shale

CORRECTIVE MATERIALS

TO CORRECT THE DEFICIENCY OF	CORRECTIVE MATERIAL	MINIMUM CONCENTRAT
"IRON" (Fe_2O_3)	IRON ORE	60 % Fe_2O_3
	PYRITES ASH	60 – 90 % Fe_2O_3
"ALUMINA" (Al_2O_3)	BAUXITE	50 % Al_2O_3
	LATERITE	25 % Al_2O_3
	KAOLINITE	40 % Al_2O_3
"SILICA" (SiO_2)	QUARTZSAND	80 % SiO_2
	QUARTZITE	80 % SiO_2
"LIME" (CaO)	LIMESTONE	90 % $CaCO_3$
	MARBLE	

EFFECTS OF MINOR ELEMENTS

Element	Concentr. raw meals %	Limits cement %	Effect
MgO	0.50-4.0	5-6 norms	Expansion
SO3	0.04-1.06	2.3-4 norms	Volatilisation Setting behaviour Emission
K20	0.01-1.40	0.6	Volatilisation
Na2O	0.03-0.60	low alk.	Alkali aggregate reaction

EFFECTS OF MINOR ELEMENTS

Element	Concentr. raw meals %	Limits cement %	Effect
P2O5	0.03-0.50	appr. 1.0	reduces strength
Cl	0.01-0.2	0.05-0.1 in several norms	volatilisation corrosion
F	0.00-0.24		flux problematic setting strength reduction emission

EFFECTS OF MINOR ELEMENTS

Element	Concentr. raw meals %	Limits cement %	Effect
TIO2	0.10-0.47		flux retards init. setting increases strength
Cr2O3	0.01-0.05		"chromium ekzema" colour
Mn2O3	0.03-0.83		colour

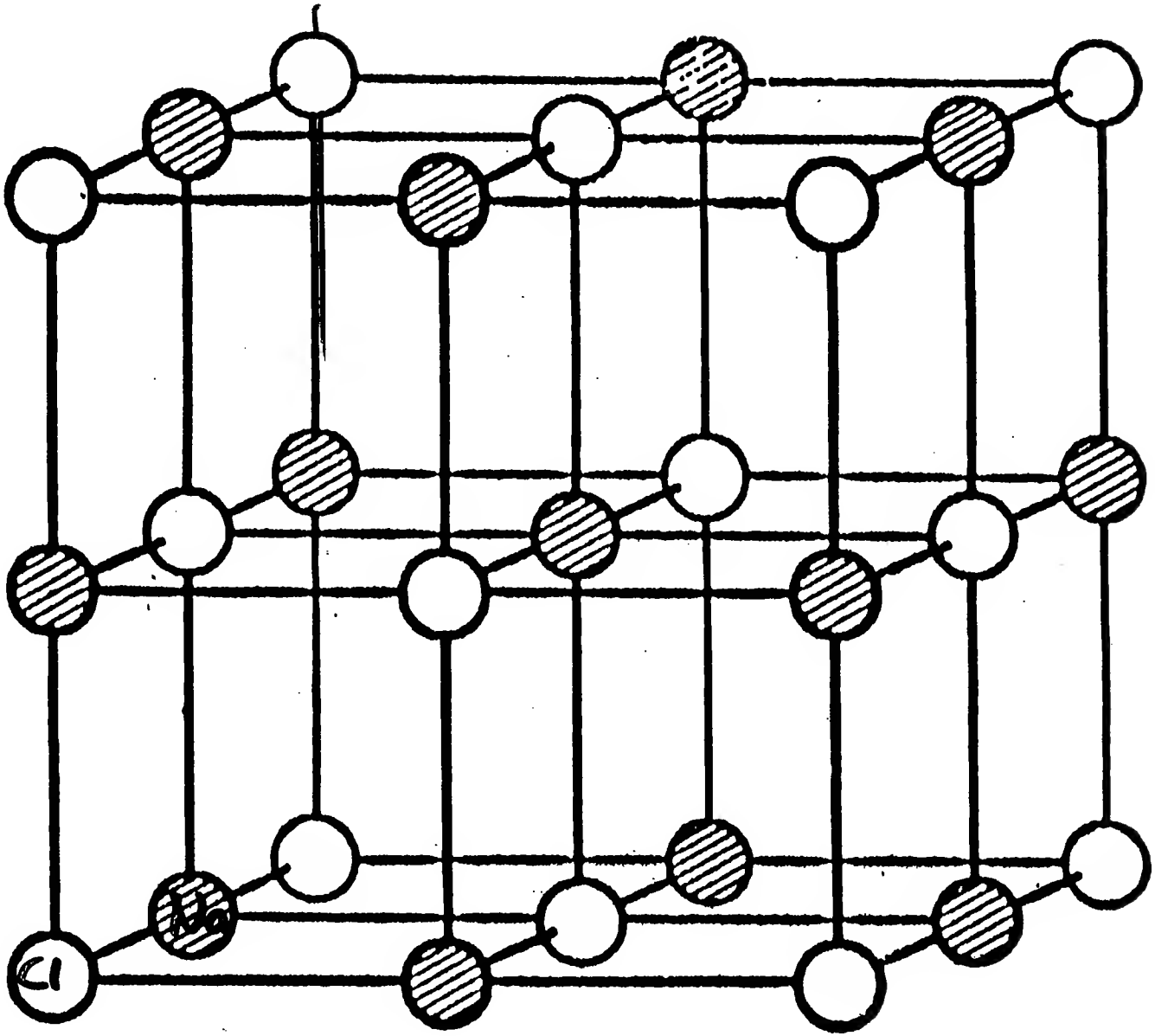


Bild 19. Die Elementarzelle (Würfel) der Steinsalzstruktur

MINERALOGICAL COMPOSITION OF RAW MIXES

CARBONATES

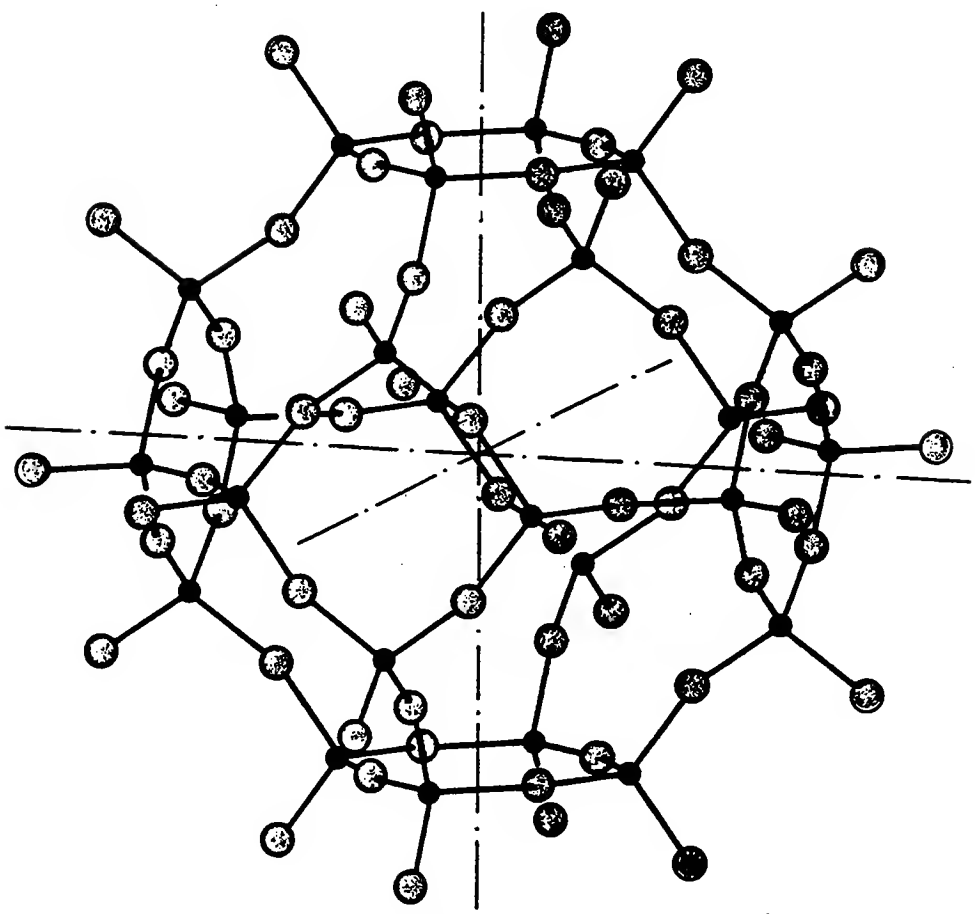
- calcite $CaCO_3$
- aragonite $CaCO_3$
- dolomite $CaMg(CO_3)_2$
- magnesite $MgCO_3$
- siderite $FeCO_3$

SILICATES

- quartz SiO_2
- feldspars eg. $(K,Na)AlSi_3O_8$
- micas
- clay minerals
- amphiboles
- pyroxenes
- olivines

Fig.46b

TYPES OF LINKAGE OF SiO_4 - TETRAHEDRA



SiO_2

THREE DIMENSIONAL FRAMEWORK

Ångström units

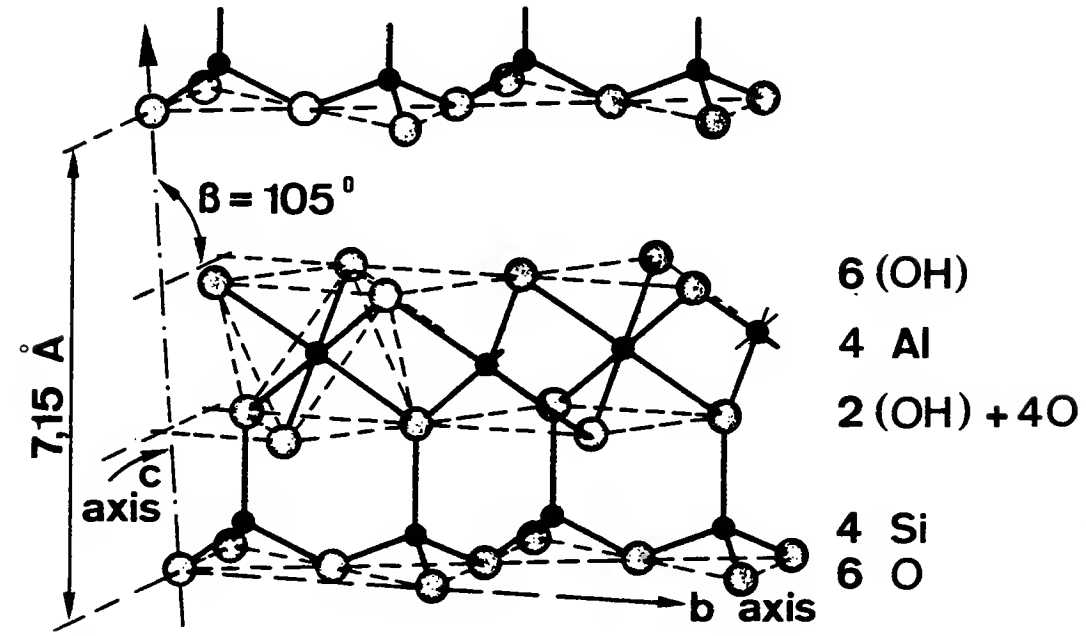
0 1 2 3

Table 34 : Hardness of most abundant minerals according to Mohs

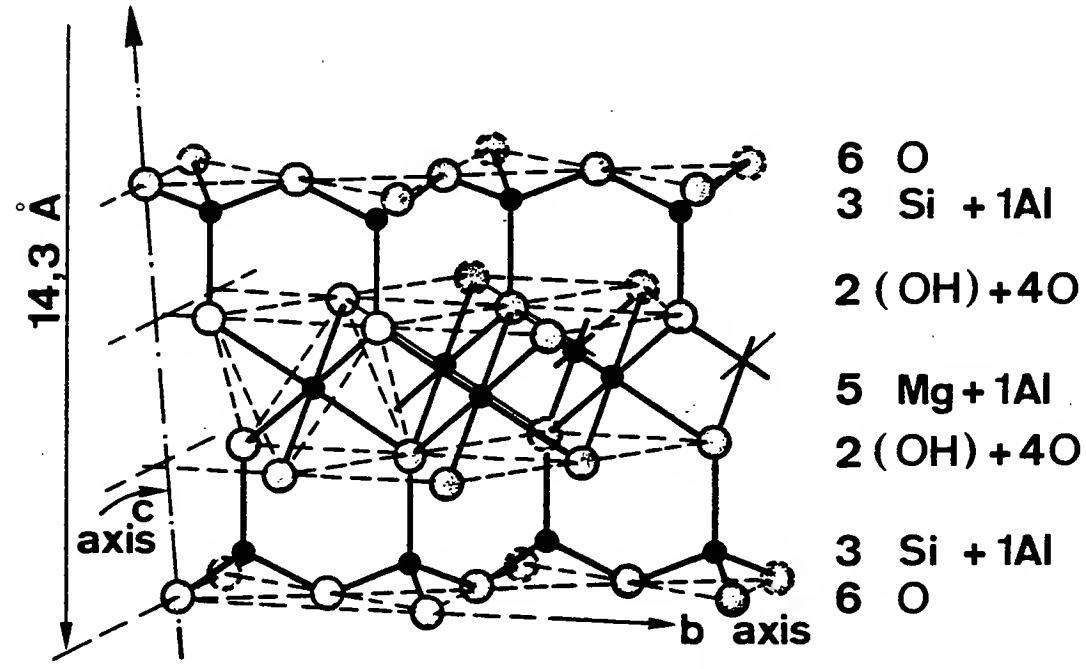
Mineral	Mohs scale	Valuation
Quartz	7	"hard" minerals
Feldspar	6	
Pyroxene	6	
Amphibole	6	
Calcite	3	"soft" minerals
Dolomite	3.5	
Mica	3	
Chlorite	2 - 3	
Clay minerals	2	

Fig. 48

CLAY MINERAL STRUCTURES



TWO-LAYER STRUCTURE (KAOLINITE)



THREE-LAYER STRUCTURE (CHLORITE)

ROCK CHARACTERISTICS

. mineral assemblage : types of minerals present

. texture and structure

- spacial distribution of minerals
- form of minerals
- size distribution of minerals and pores
- orientation of minerals and pores

. consolidation

PHYSICAL ROCK PROPERTIES

. density and porosity
. water absorption, moisture
. compressive strength
. elasticity

. cleavability
. hardness
. crushability
. abrasivity
. grindability

Table 32 Density and porosity of rocks

Rock	Specific gravity (density) g/cm^3	Porosity %
Granite	2.5 - 2.8	0.5 - 1.5
Diorite	2.6 - 2.9	0.1 - 0.5
Gabbro	3.0 - 3.1	0.1 - 0.2
Dolerite	3.0 - 3.1	0.1 - 0.5
Rhyolithe	2.4 - 2.6	4 - 6
Andesite	2.2 - 2.3	10 - 15
Basalt	2.8 - 2.9	0.1 - 0.2
Tuff	2.8 - 3.0	high
Sand (loose)	1.5 - 2.0	25 - 50
Sandstone (compacted)	2.0 - 2.6	10 - 30
Shale	2.0 - 2.4	10 - 30
Limestone	2.2 - 2.6	5 - 20
Dolomite	2.5 - 2.6	1 - 5
Marl	1.6 - 2.6	5 - 30
Clay	1.5 - 1.9	30 - 50
Claystone	1.8 - 2.3	15 - 35
Gneiss	2.9 - 3.0	0.5 - 1.5
Marble	2.6 - 2.7	0.5 - 2
Quartzite	2.6 - 2.7	0.1 - 0.5
Slate	2.6 - 2.7	0.1 - 0.5

Table 33 Compressive strength of rocks

Rock type	Compressive strength kg/cm^2 / N/mm^2	Valuation (of compressive strength)
Basalt Quartzite	> 2800 / > 280	very high
Granite Diorite Limestone	1800 - 2800 / 180 - 280	high
Sandstone Limestone Marble Slate	800 - 1800 / 80 - 180	medium
Porous lime-stone	400 - 800 / 40 - 80	low
Chalk, Tuff, Shale, Clay, Claystone	< 400 < 40	very low

GRINDABILITY OF RAW MIXES

"Holderbank" Grindability Test; 10 % Residue on 90 µm)

	A	B	C
GRINDABILITY KWH/T	12	17.5	22
ASSESSED AS	GOOD	MEDIUM TO POOR	VERY POOR
COMPONENTS	85.7 % LIMESTONE MARLY LIMESTONE POROUS, FINE GRAINED, HOMOGENEOUS 14.3 % CLAY	87.2 % LIMESTONE LIMESTONE RECRYSTALLIZED, DENSE, HARD 12.8 % CLAY	77.7 % LIMESTONE HIGH GRADE LIMESTONE 16.5 % SHALE 5.7 % SANDSTONE

Chapter 7

Detailed Investigations of Raw Materials

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Detailed investigations of raw materials

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1. INTRODUCTION

The overall investigations of raw materials have proved that one of the deposits investigated is suitable for the purposes of clinker / cement production. Therefore, the proposed cement plant is feasible. The last step of the investigations, known as detailed investigations of raw materials, is carried out in order to define the selected deposit accurately in quantity and quality. Once these detailed investigations have been completed, it is possible to 'construct' a computer generated block model (CADE) and implement QSO Expert systems for medium and long term planning.

The main tasks in this step are as follows:

- Detailed drilling campaign in order to obtain the most accurate information with respect to the geological structure, chemical behaviour and associated reserves of the deposit. The drilling volume usually is in the order of between 1000 and 2000 m, whilst 300 to 1000 samples will be selected for chemical analysis.
- Geotechnical data to assess the overall stability of the slopes, rock strength, fracture patterns, groundwater circulation
- Quarry layout and development concept is prepared by the mining engineer

2. DETAILED DRILLING CAMPAIGN: OPTIMAL DRILL HOLE PATTERN

A question asked frequently during the exploration stage of a deposit is: "how much drilling is required to accurately describe the deposit and to achieve the required precision of results ?"

The normal procedure is to apply geostatistics on the chemical data of the initial drilling campaign. With the help of variograms, which are a measure of the **spatial** interdependence of chemical characteristics between the drill holes, it is possible to determine the optimal drill hole pattern.

Fig 1 is a graphic display of the relationship between possible errors of estimation and number of drill holes, using a statistical and geostatistical approach. Two important conclusions can be derived from this figure:

- to achieve a required accuracy, the number of drill holes can be reduced significantly using geostatistical methods
- using geostatistical analyses on the same number of drill holes, the error of estimates is reduced.

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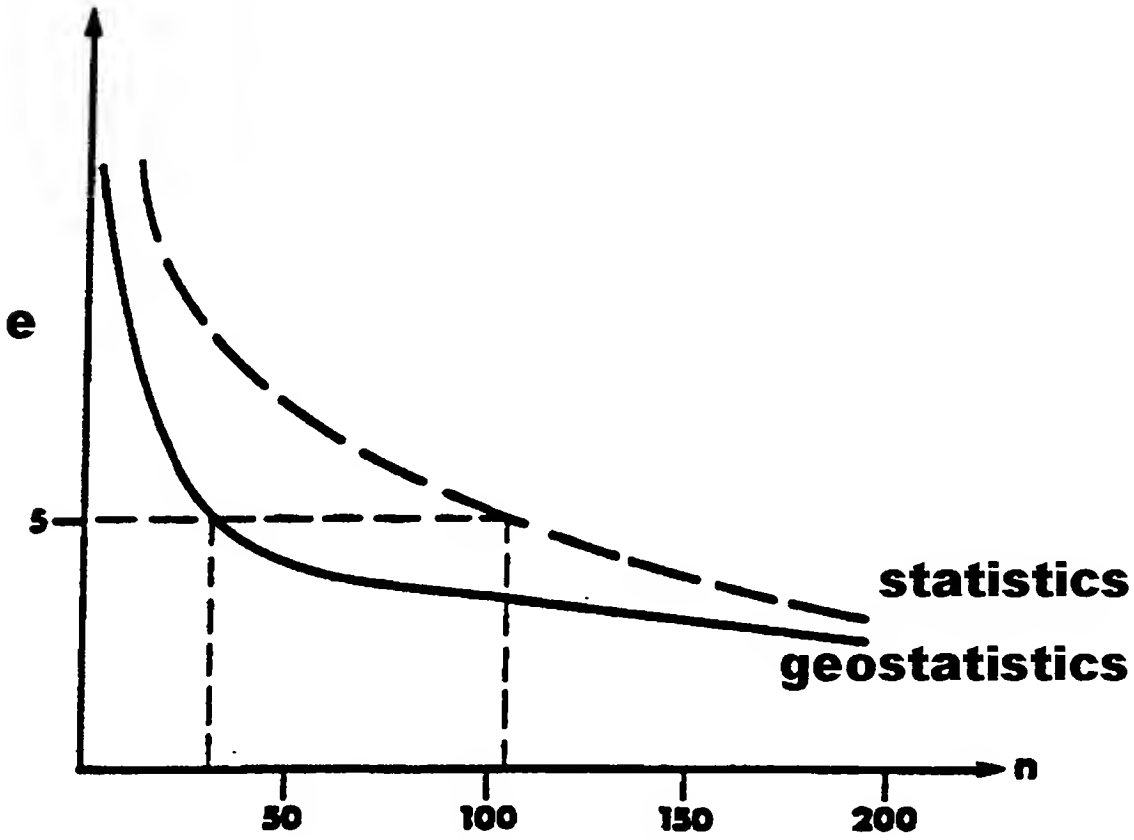
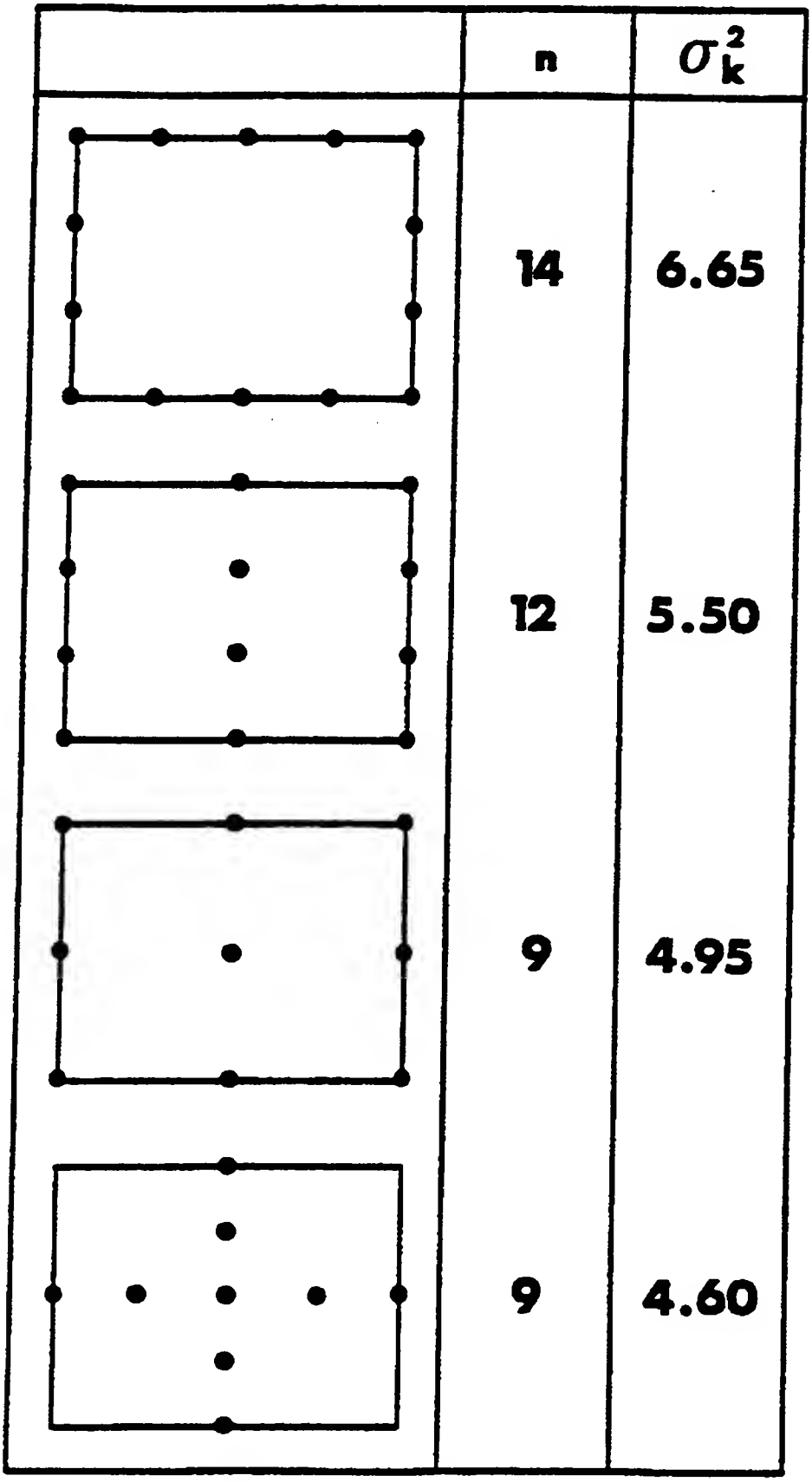
Fig. 1 **Relationship between number of drill holes n and estimation error using statistics (dashed line) and geostatistics.**



n = N° of drillholes e = error limit

Fig. 2 **Comparison of the errors associated to different sampling patterns.**



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Optimal grid design (spacing, regular, irregular, etc.) can be tested with the aim to minimise the estimation errors in the interpretation of the data by kriging (Fig. 2)

3. GEOTECHNICAL ASPECTS

In designing the quarry slopes, we are faced with two conflicting requirements. On the one hand by steeping the slope, the stripping ratio waste / ore to be excavated is reduced and recoverable reserves are increased. On the other hand, loss of life and serious damage to property can result from failures induced by excessive steepening of particular slopes. A compromise between steepness and economy has to be found for an economically safe quarry.

A practical solution is built up from the investigations of the following criteria:

- **Geological data**: lithology (presence of thin intercalations of clay), geological mapping, strike and dip of all types of planes and structures in the deposit
- **Physical data**: rock strength, cohesion, angle of friction
- **Groundwater conditions**: the presence of groundwater in the rock mass surrounding an open pit has a detrimental effect upon the mining programme for the following reasons:
  - water pressure reduces the stability of the slopes and increases the forces tending to induce sliding
  - high moisture content in shales decrease the stability

Freezing of groundwater during winter can cause wedging in water-filled fissures due to volume changes in the ice. Freezing of surface water on slopes can block drainage paths resulting in a build-up of water pressure in the slope with a consequent decrease in stability.

Discharge of groundwater into an open pit gives rise to increased operating costs because of: (a) the requirement to pump this water out, (b) also because of the difficulty of operating heavy equipment on very wet ground and (c) increased wear / maintenance of in-pit equipment.

Liquefaction of overburden or waste tips can occur when water pressure within the materials rises to the point where the uplift forces exceed the weight of the soil.

By far the most important effect of the presence of groundwater in a rock mass is the reduction of stability resulting from water pressures within the discontinuities in the rock.

3.1 Slope failures

The following stability problems could be expected in an open pit quarry:

Plane failure occurs when a geological discontinuity, such as a bedding plane, strikes parallel to the slope face and dips into the excavation at an angle greater than the angle of friction.

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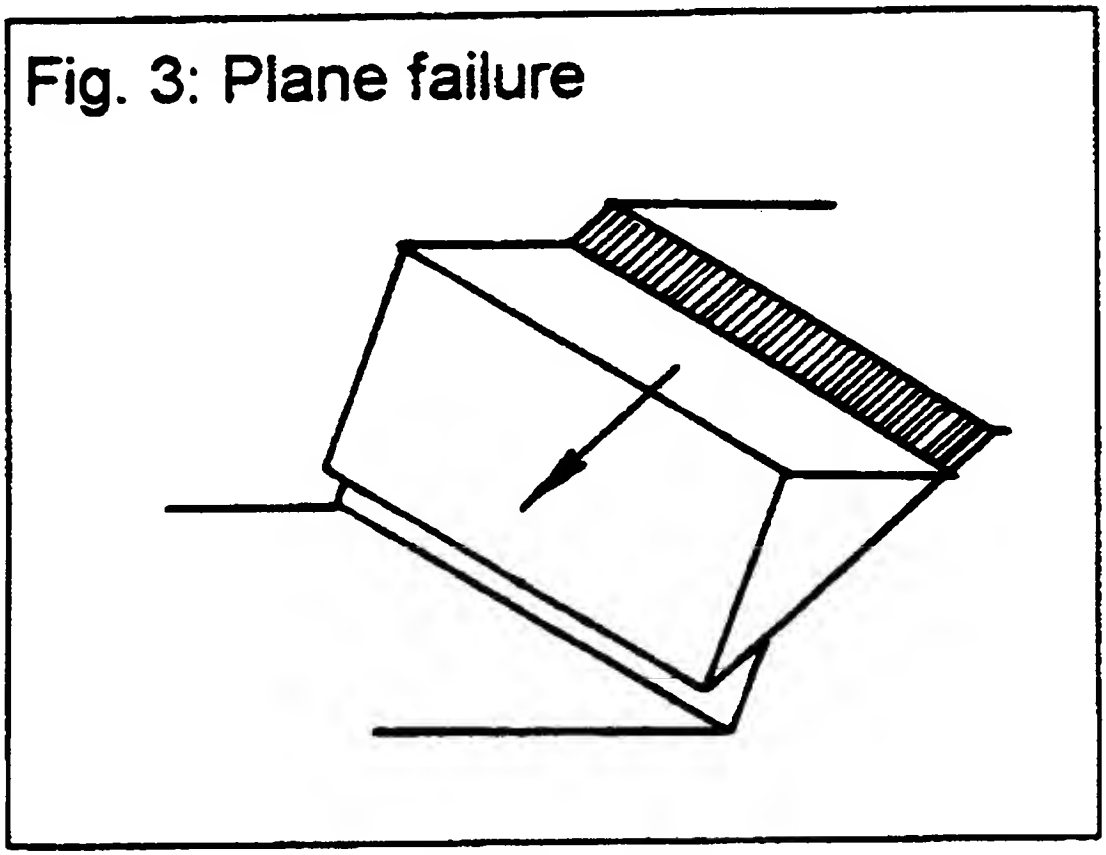
Fig. 3 Plane Failure



Wedge failure (Fig. 4) occurs when two discontinuities strike obliquely across the slope face and their line of intersection daylights in the slope face. The wedge of rock resting on these discontinuities will slide down along the line of intersection, if the inclination of this line is greater than the angle of friction.

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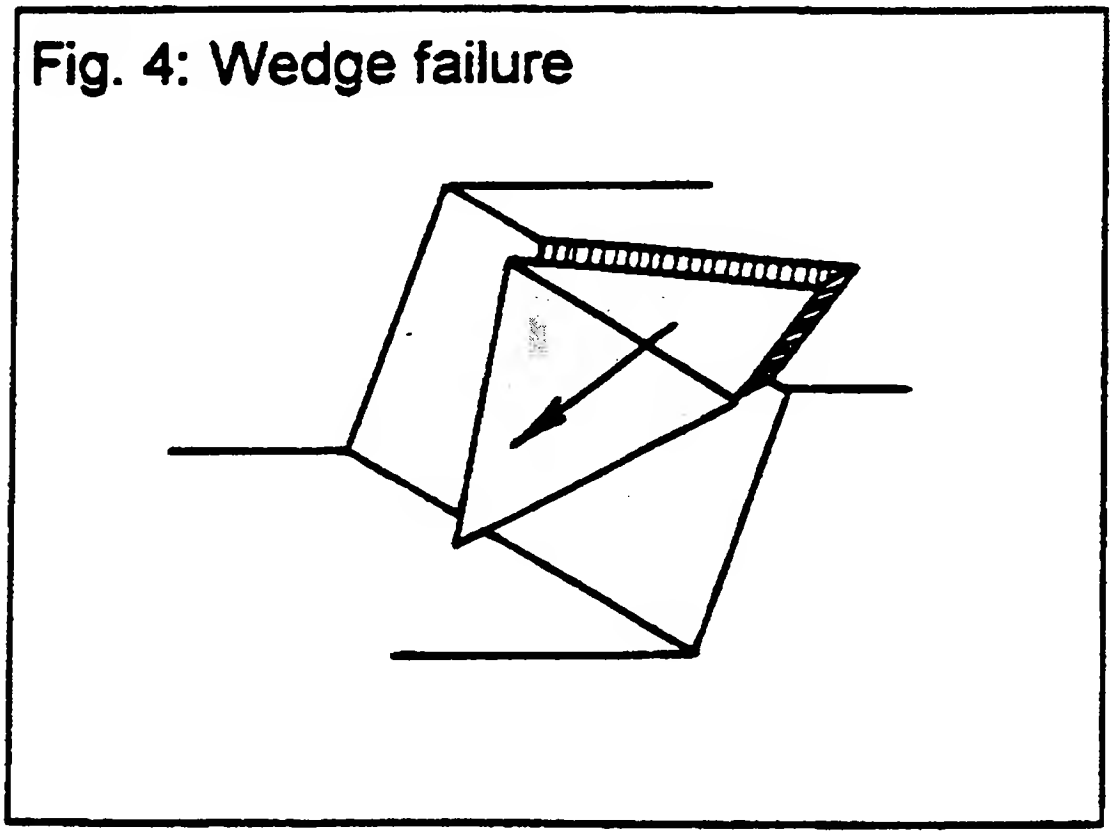
Fig. 4 Wedge Failure



Circular failure (Fig. 5) occurs when the materials are very weak. For example in a soil slope, or when the rock mass is heavily jointed or broken, (waste rock dump). The failure will be defined by a single discontinuity surface and will tend to follow a circular failure path.

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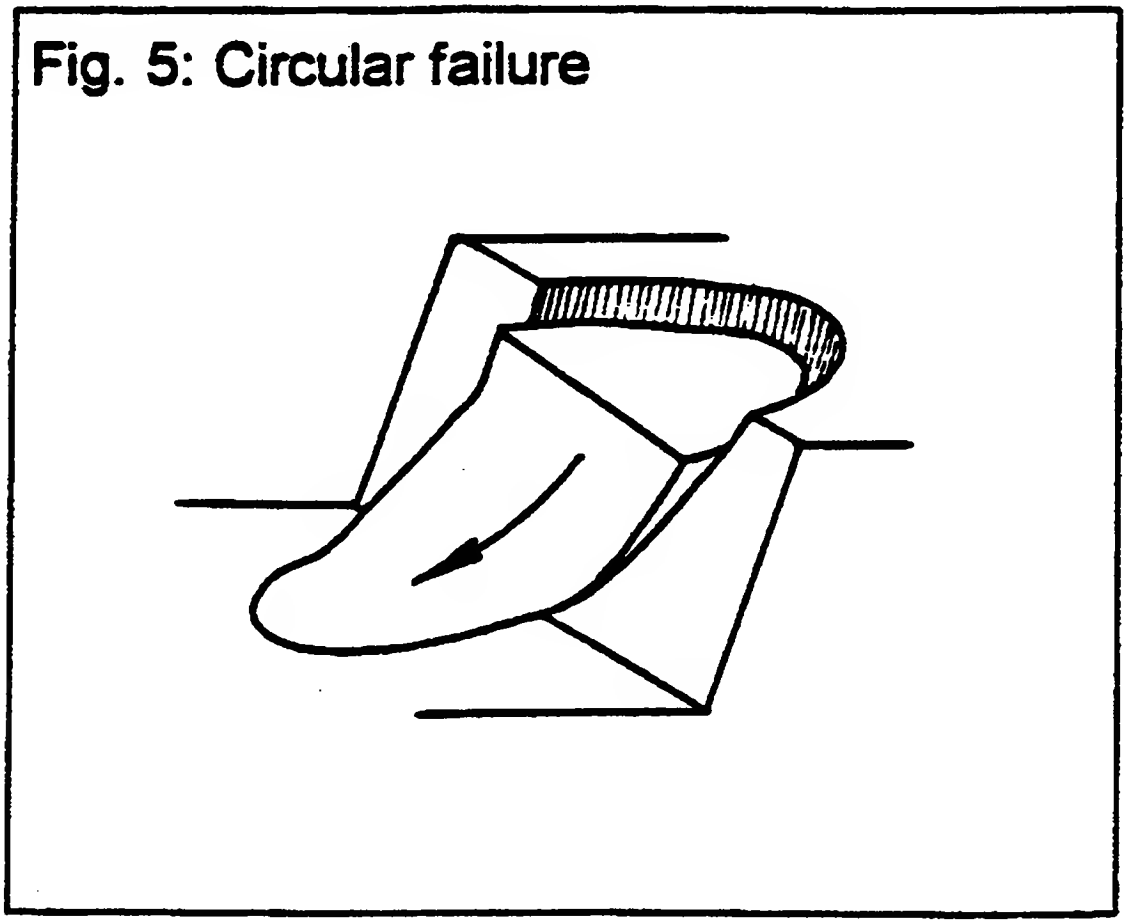
Fig. 5 Circular Failure



Toppling (Fig. 6) occurs when the sub-vertical layers become unsteady and fall over.

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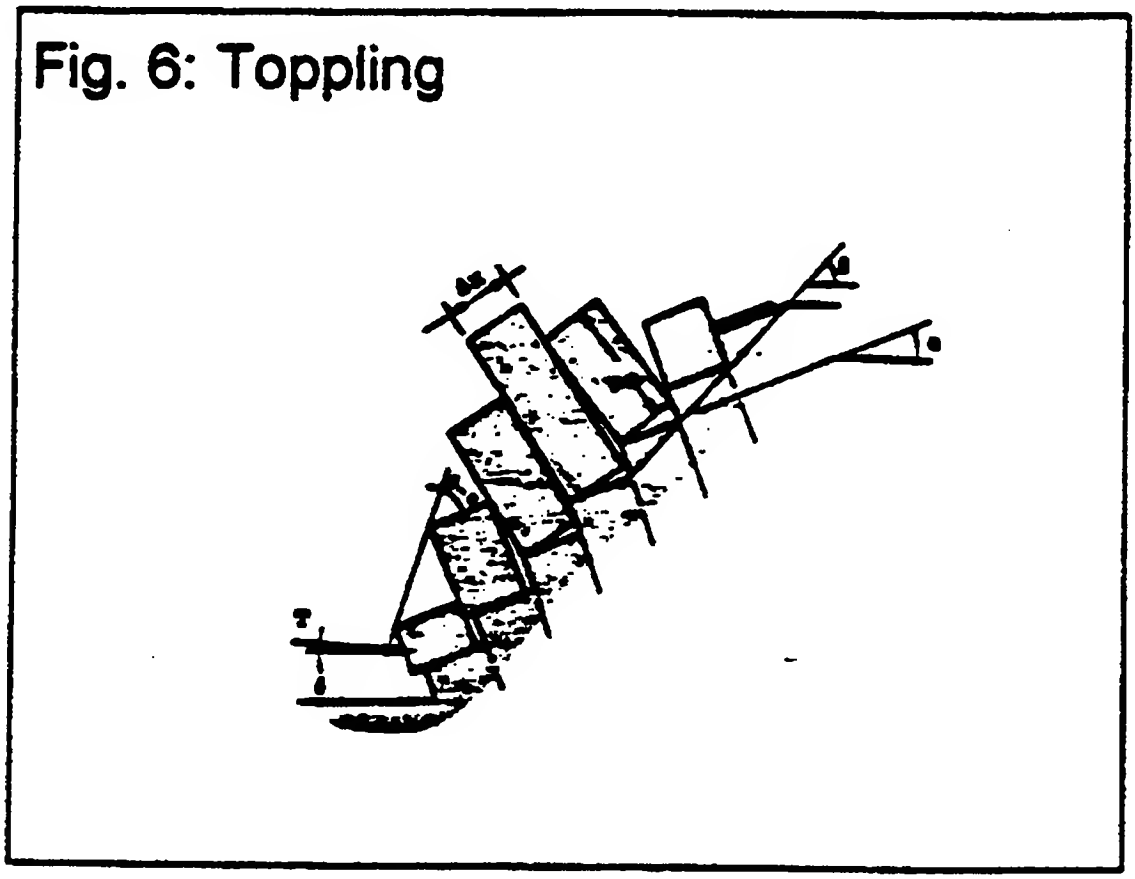
Fig. 6 Toppling



Weathering or deterioration of certain types of rock on exposure gives rise to a loosening of a rock mass and the gradual accumulation of loose materials on the surface and at the base of the slope.

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3.2 Overall slope stability

An estimate of safe slope angle is required for the calculation of ore to waste ratios and for the pit layout. The only structural information, which may be available at this stage, is that which has been obtained from diamond drill cores logging and from geological mapping and structural measurements. Scant as this information may be, it does provide a basis for the estimate of potential slope problems.

Slope height versus slope angle

In Figure 7 (after Hoek and Bray, 1981), the slope heights and corresponding slope angles in materials classified as hard rock has been plotted. This plot shows that the highest and steepest slopes, which have been successfully excavated, fall along a clear line shown in dashes. This line gives a useful, practical guide to the highest and steepest slopes, which can be contemplated for normal open pit mine planning. **A steeper slope could only be justified if a very comprehensive stability study shows that there is no risk of indicating a massive slope failure.**

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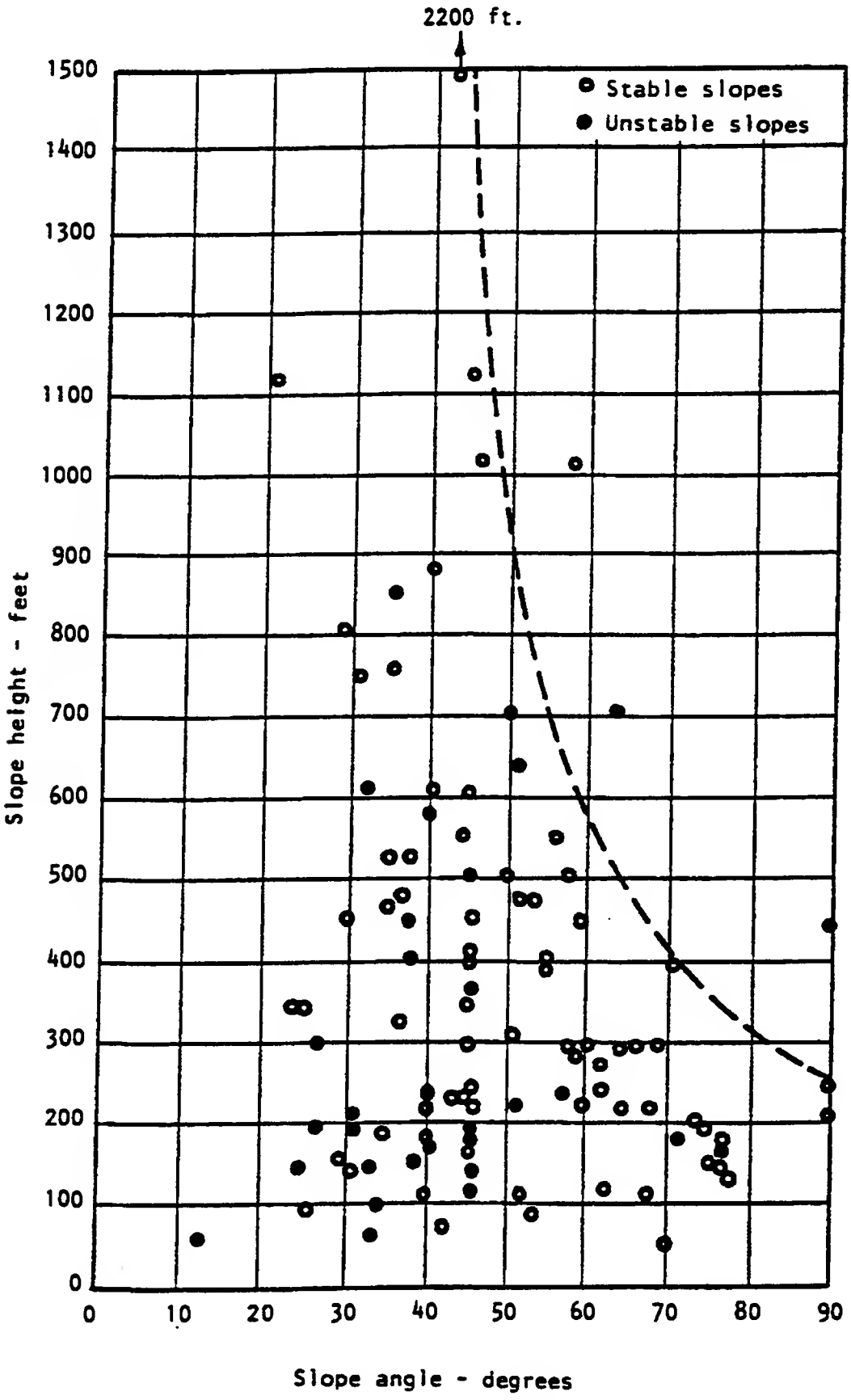
Fig. 7 Slope height versus slope angle (Hoek and Bray, 1981)



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Chapter 8

Environmental Aspects of Mining and Rehabilitation

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Environmental aspects of mining and rehabilitation

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1. INTRODUCTION

Cement manufacturing process has various impacts on the environment and the ecology. For this reason, most countries impose strict legislation for environmental protection. It is therefore essential to consult the environmental regulations when selecting potential raw materials resources. Rehabilitation planning can be carried out with the help of the special "Holderbank" software, Quarry Engineering Design (QED). A clear presentation of the different phases of minerals exploitation and quarry rehabilitation with 3D animation can be shown.

2. ECOLOGY

Ecology is defined as the relationship between nature (plants, animals), man and the environment. The environment is the sum of all external forces or influences. Every plant and animal exists within a unique and continually changing microenvironment. The ecosystem is made up of three of ecological systems:

- individual
- population
- ecosystem

The ecosystem, being a balance or equilibrium of the existence and the interactions between populations, has been disrupted by man causing a profound change of these balanced ecosystems.

In part, the quarrying activities can be disruptive, if not totally destructive to an ecosystem. Apart from the more apparent effects of an anaesthetic pollution caused by dust and smoke, there is also the local destruction of the flora and fauna. Usually, quarries are situated in rocky outcrop areas, where the overburden is very thin. These areas are often the refuge of various animals and specific plants.

What can be done do to minimise, divert or correct such disruptive effects of quarrying activities?

3. REHABILITATION OF QUARRIES

Rehabilitation schemes as part of an exploration concept include the reincorporating of the quarry area into the surrounding landscape, its re-cultivation with the purpose of creating an integral ecosystem with a new function. The possibilities of utilising a worked-out quarry are numerous. Apart from the most common utilisation in agriculture and forestry, it could also serve for industrial purposes such as a site for a power plant or a refuse tipping site.

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Possibilities of utilisation of a work-out quarry are as follows:

- Agriculture
- Forestry
- Recreation areas: woods, ponds, rock garden
- Scientific activities
- Sanctuary (wild life)
- Pisciculture
- Lakes, ponds
- Sporting grounds
- Tipping areas
- Commercial/ industrial : storage area, traffic area, training area...

Experience has shown that a decision on the future utilisation of an exhausted quarry should be made very early on in the project program. Terms of an agreement between the exploiting company and the executive of public planning authorities must be drawn up and implemented.

In each individual case, local conditions will determine the choice of the rehabilitation programme and accordingly the cost estimates including return of investment. In the example described in table 1 which is an average case, the cost for rehabilitation is approximatly $ 100'000 per ha or $10 per m^2.

Table 1: Cost for quarry rehabilitation

Activities	Cost in $/m^2
Soil removal	1.00
Preparation of filling materials	1.00
0.3 m soil cover	2.00
0.35 m humus cover	2.00
Plants	1.00 -2.00
Wood	2.00
Spraying of slopes	1.00 - 2.00
Forestation	1.00

A rehabilitation or utilisation scheme is established after a thorough evaluation the most important factors, namely geological conditions and the groundwater situation. In a second stage, data on weather conditions, landscape planning, economy, need to be collected and evaluated.

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4. EXAMPLE OF REHABILITATION

There are many examples of recultivation / rehabilitation of quarries in many countries. One such example is the problems caused as a result of uncontrolled landfill in Switzerland.

4.1 Landfill of the quarry "Bärengraben", Switzerland

This state owned limestone quarry was exploited between 1930 and 1960 as raw materials source for a cement plant. A thick overburden consisting of sand and conglomerate overlies the suitable limestone resource. Exploitation of the limestone occurred on the flank of the hill, leaving an exposed open-cut slope.

1960 to 1975. Once exploitation of limestone had ceased in 1960, the quarry was utilised as a repository for demolition material and organic wastes. In 1975, as the quantity of waste materials increased considerably, the local authorities decided it was necessary to establish a waste management system.

In 1975, the situation was as follows :

- deposition of materials of demolition materials (inert building rubble) was mixed with other waste material. Uncontrolled waste waste deposition.
- drainage system (cement pipes to collect the groundwater) was defective due to aggressive water (high content in sulphate)

From 1975 onwards, the new project of landfill consisted of (Fig 1):

- a new drainage system with gravel beds between waste layers and along the slopes of the quarry
- establishment of the final slope of the deposition materials in order to avoid slope failure (landslides)
- installation of a measurement system to determine compaction
- cladding the external slope with soil and humus
- a compactor

The price of the installation and the costs of the operation amounted to CHF110.- per cubic meter of waste.

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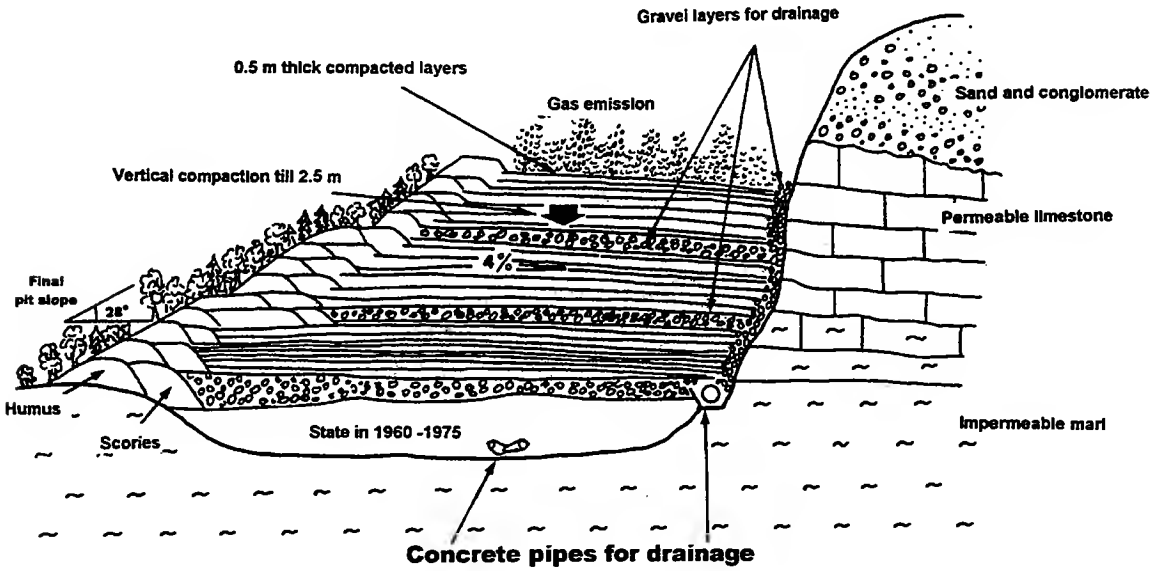
Fig. 1 Cross-section of the landfill / quarry at Bärengraben



Compacting was regular and there were no problems with the drainage installation.

For many years, this quarry was backfilled without any control of the quality of the materials being dumped. With the result that the waste materials consisted of building rubble and other disallowed products. For example :

- domestic waste and detritus
- scories of two chemical plants
- excavation materials contaminated by acetate and acids
- inert materials of demolition

However, in 1986 at the end of the landfill, groundwater investigations in the valley confirmed that as a result of the migration of contaminated water out of Bärengraben the groundwater in the main valley was contaminated. Hazardous wastes are lixiviated by the clean groundwater flowing thorough the overburden and limestone. This drainage water, or deposition juice, flows into a small basin of decantation on the site before permeating back into the groundwater system. This contaminated water contains heavy metals, which are very dangerous to the health.

The groundwater is the potable water source for many villages in the area. Fig. 2 shows the situation of the groundwater circulation. It was estimated that 45 l/minute of contaminated water, which is not successfully collected in the drainage pipes, flowed into the valley.

A project to collect all the contaminated water was implemented. This involved construction of a collection tunnel behind as well as an impermeable barrier in front of the repositry, in order to prevent contaminated water flowing into the groundwater. The cost estimation amounted to CHF40 - 60 Mio.

Early in 1990, signifcant gas emissions coming from the decomposition of organic materials incommoded the people of the village near the old quarry. In order to collect this gas a system of boreholes and pipes was installed. This consisted of a flow system for the gas equipped with a ventilator, an analyser (continuous analysis of the gas) and a gas burner. The quality of gas was not sufficiently good enough for commercial use (low methane).

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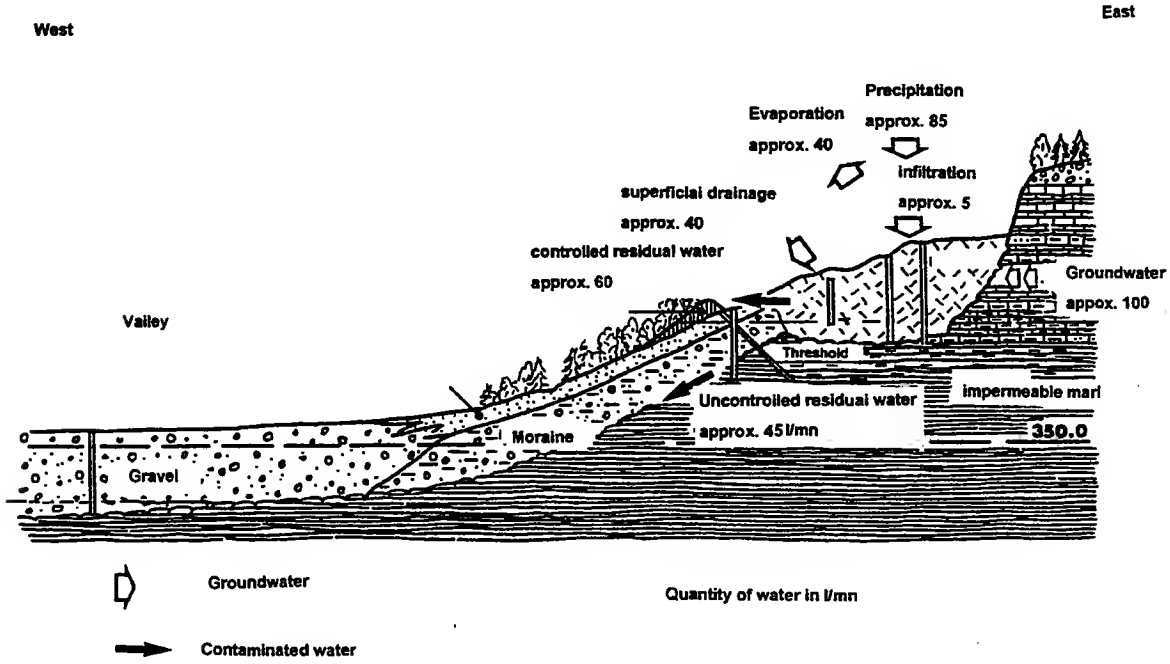
Fig. 2 Hydrogeological model of the landfill of Bärengraben



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Chapter 9

Project Management Raw Materials Surveys

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Significance of raw materials assessment in planning and execution of a project

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2. THE "HOLDERBANK" APPROACH 204
2.1 Desk Study 205
2.2 Preliminary Raw Material Prospecting 205
2.3 Overall Raw Material Exploration 206
2.4 Detailed Raw Material Exploration 206
3. COST OF RAW MATERIAL PROSPECTING AND EXPLORATION 207
4. SIGNIFICANCE OF RAW MATERIALS INVESTIGATIONS IN PROJECTS 210

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1. EXPLORATION PROCEDURE

The following paragraphs describe a procedure developed specifically for the exploration of cement raw material deposits. The proposed scheme is based on experience and modelled after the needs of the cement and aggregate industry. Of course, every exploration project requires modifications necessitated by local circumstances.

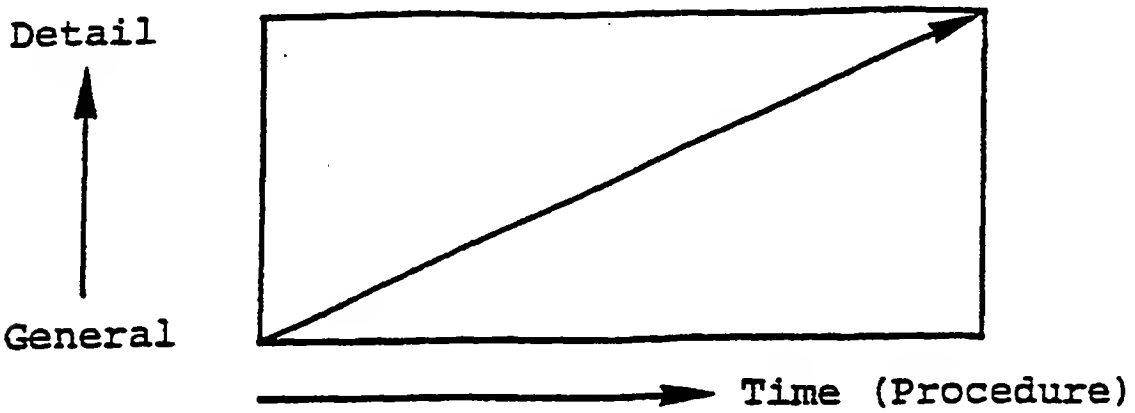
Basically, the scope and extent of every exploration procedure is governed by the following series of criteria:

- Scope of work
- Status of geological knowledge of the area to be examined
- Raw material-inherent characteristics: geological exposures, etc.
- Time and financial resources available
- Climatic conditions
- Physiography and topography of the area

2. THE "HOLDERBANK" APPROACH

The "Holderbank" approach to recognition and exploration of raw material deposits is to proceed from a general basis to more detailed investigations (Fig. 65) continuously condensing the mass of information down to the significant details only.

Fig. 65 The "Holderbank" Approach



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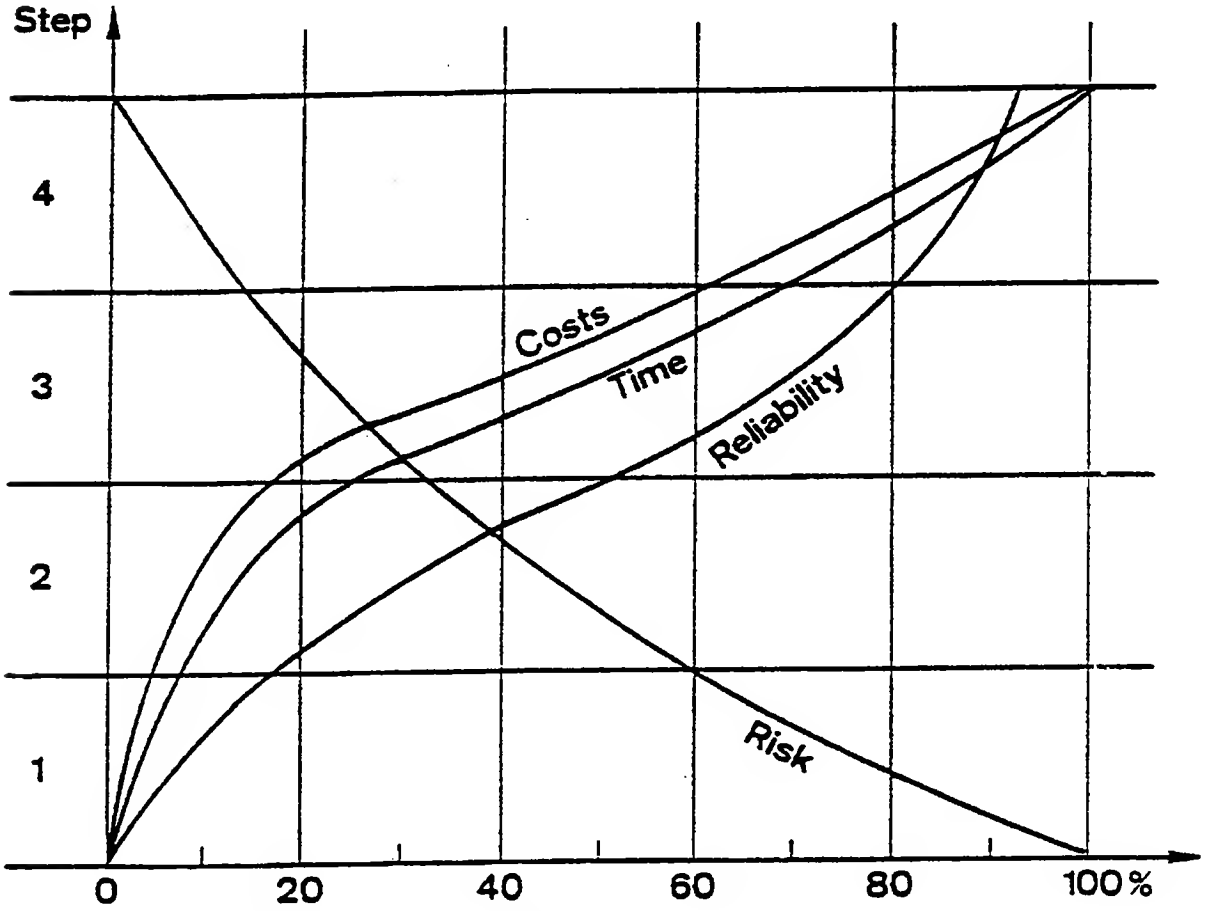
The procedural steps are defined with regard to their respective scope, extent, costs, time required, reliability of results (error limits) and the development of risk (Fig. 66).

Fig. 66 **Development of costs, time, reliability and risk during raw material investigations.**



In practice, the proposed procedure entails the following scheme:

1) desk study
2) preliminary raw materials prospecting
3) overall raw materials exploration
4) detailed raw materials exploration

2.1 Desk Study

After the preceding activities (acquisition of the project, quotation, etc.) have been completed, a fact finding study (or desk study) is conducted. The aim thereof is to acquire and evaluate all available geological, chemical and topographical facts and documents in order to generate a sound basis for subsequent investigation phases.

2.2 Preliminary Raw Material Prospecting

An initial evaluation of potential cement raw material deposits, both with regards to quality and quantity is based on existing geological maps and a suite of surface samples from the regions or areas under consideration. If the quantity or quality does not meet the

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preconditions set (ie. the results are negative), the whole project is either shelved or similar investigations are initiated in other nearby regions.

Positive results, however, call for further, more thorough investigations in the promising areas. In any case, after completion of this phase, the client is to decide whether or not further investigations should be carried out.

2.3 Overall Raw Material Exploration

If by the above decision it was agreed to proceed, the next step would be the quantitative and qualitative characterisation of potential raw material deposits in one to three areas per raw mix component. Fresh representative raw material samples are collected from trench and/or drilling sampling. Finally, the most suitable deposits of each component are selected on the grounds of quantity, quality, accessibility, availability etc. The findings of this step of the investigation ought to be detailed enough to assess the feasibility of the project with regard to the raw material aspects.

2.4 Detailed Raw Material Exploration

After the decision - based on the overall feasibility of the project - to continue with the investigations, the selected quarry area is thoroughly investigated by drilling and subsequent quantitative and qualitative characterisation of the respective raw material areas.

In addition, a quarry exploitation scheme for all components is established.

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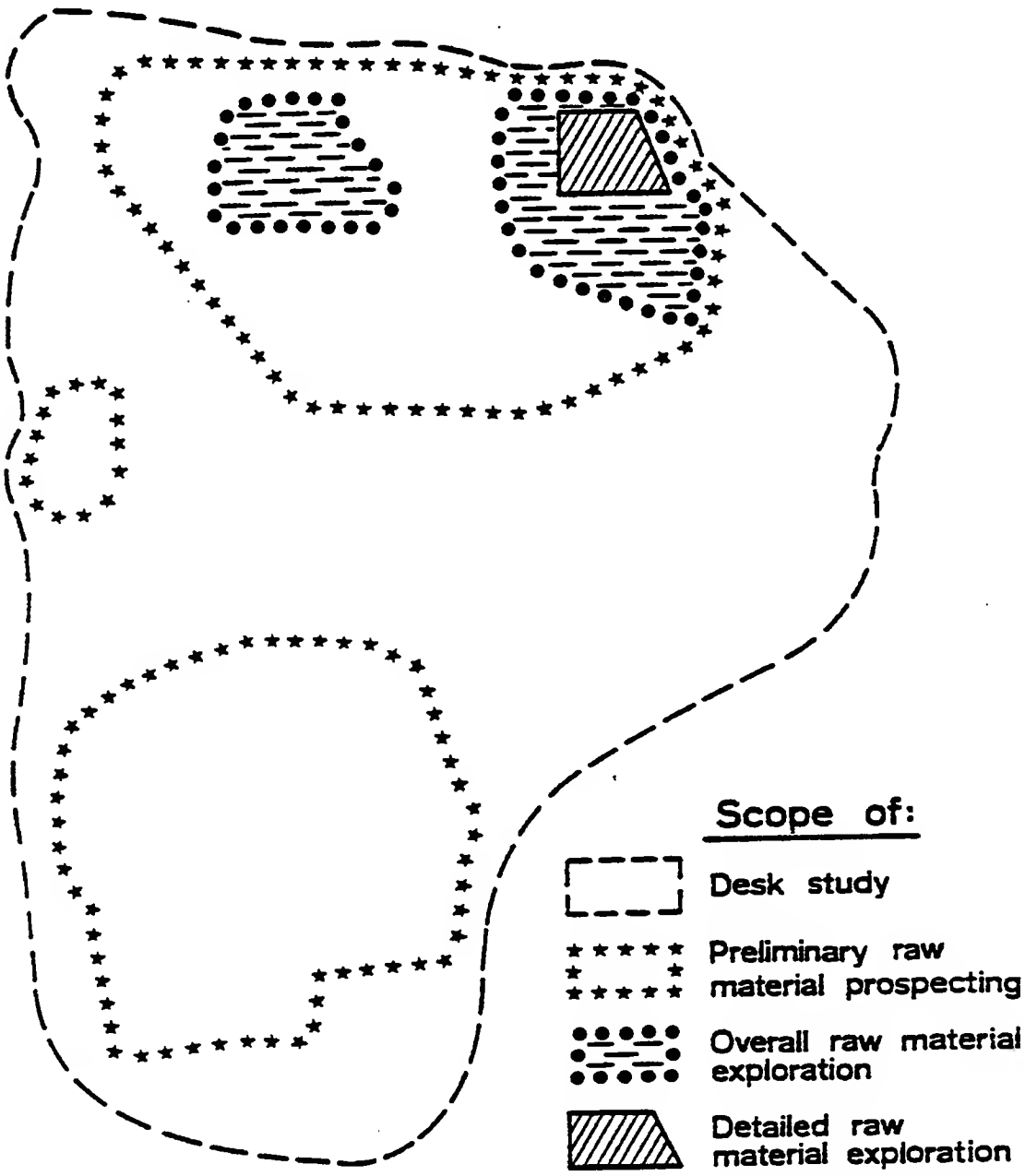
Fig. 67 Succession of Investigation Steps.



Note: The "Holderbank" approach is described in detail in the following report:

Report MA 78/2581/E "Procedure Manual for Raw Material Investigations"

Fig. 67 displays the above described procedure for a hypothetical case.

3. COST OF RAW MATERIAL PROSPECTING AND EXPLORATION

The cost of raw material prospecting is very often perceived to be high. However it must be stressed that inadequate or worse still inaccurate knowledge of the cement raw material

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deposits will certainly create innumerable difficulties during the subsequent processing stages (exploitation, preparation, processing, quality assurances).

It is obvious that the more complicated and less homogeneous a deposit is, the more thoroughly it must be investigated and the higher the costs of prospecting and exploration will be. This is especially true for low quality raw material deposits, which are situated far from market areas.

Experience has shown that the costs of a comprehensive raw material exploration campaign normally amounts to betweenUS $ 100'000 and US $ 300'000 excluding additional costs possibly required for extra drilling and geophysical surveys. A vague relationship exists between capacity, investment cost and costs for prospecting (Table 66):

Table 66: Cost of raw material prospecting versus investment costs

Plant capacity (tons of clinker per year)	Investment costs in mil (US$)	Cost quota of raw material prospecting (%) (US$)
500'000	80	0.4 - 0.6 or 400'000.--
1'000'000	150	0.3 - 0.5 or 600'000.--
2'000'000	250	0.2 - 0.4 or 750'000.--
aggregate plants tons of aggregates 500'000	15 - 20	1.5 - 2 or 300'000.--

The above figures include drilling costs, geological surveying costs etc., although it is difficult to make any general cost estimates, particularly of drilling costs, because the total length of drilling, the characteristics of the material to be drilled, the anticipated depth of the drill holes, water and power supply, transport and catering of drilling equipment and crew vary from one case to the other. The drilling costs also differ between the various countries as Table 66 a shows.

Table 66a: Cost of drilling for prospecting purposes

Country	Cost: US $ per meter
Honduras	100 - 150
Libya	300 - 400
Switzerland	100 - 250
Australia	100 - 150
Saudi Arabia	100 - 200
Viet Nam	80 - 120

Volume of drilling required for exploration activities depends on the level of detail required. On average the following drilling quantities should normally be considered:

Overall raw material exploration 1000 m
Detailed raw material exploration 1000 - 3000 m

A break-down of the drilling costs looks like this:

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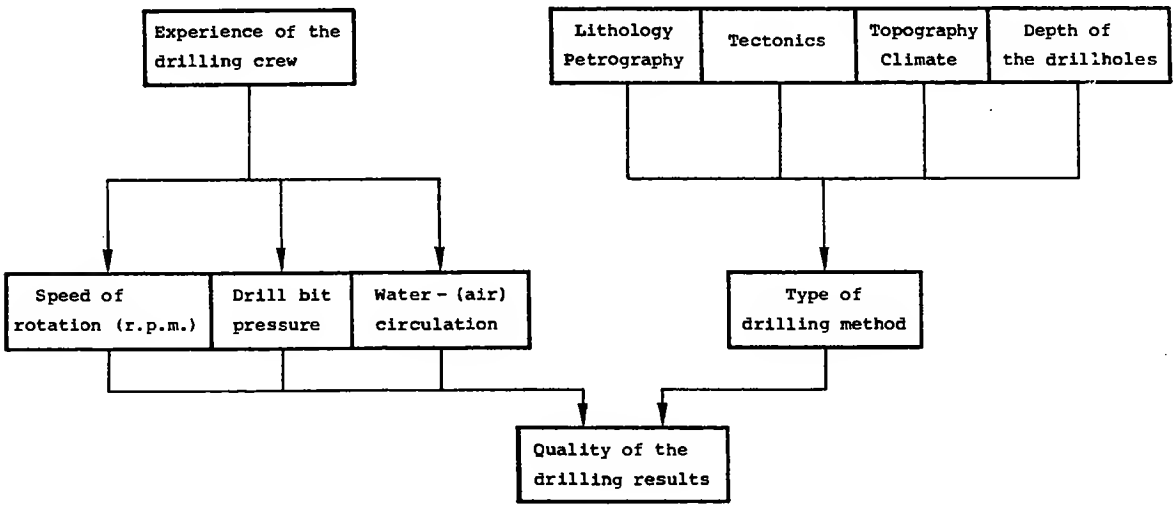
Components	%
Labour	35.0
Lodging and wages for foremen	8.0
Transport on site	2.0
Drill bits	16.0
Repairs, maintenance and spares	5.0
Fuel and lubricants	3.5
Cement and rock bolts	4.5
Water supply	2.0
Annuities and depreciation	3.5
Rock samples handling and preparation	*1.0
Concession royalties, licencing & surveying	9.0
TOTAL	100

Table 70 Factors Affecting the Quality of Drilling Results



4. SIGNIFICANCE OF RAW MATERIALS INVESTIGATIONS IN PROJECTS

The term "projects" as used in this context comprises the tasks summarised in the following table:

Project	Objectives of geologist
New cement plant/aggregate plant ("grass-root-project")	Location and evaluation of potential raw material reserves
Extension of existing plant (cement or aggregate)	Verification of reserves, search for additional raw materials
Acquisition of existing (cement or aggregate)	Assessment of raw material reserves; & quarrying concept
Conversion of process mainly cement plants	Technological assessment of raw materials situation
Operational difficulties in existing plant	Search for substituting materials: & adaptation of quarrying concept
Manufacture of new cement types	Location and evaluation of new raw materials for cement type

Securing raw materials which are satisfactory with respect to quantity and quality, is the main objective in all the above cases, but in particular for the conversion / extension projects and "grass root" projects. A project could still be realised even if the market survey study does not provide conclusive results, or the infrastructure and technical concept study is not evaluated thoroughly enough. However, if the raw material investigations are not conducted with sufficient care or if pertinent results are negative, the project as a whole would not be viable.

Naturally, the raw material investigations (Table 72) are the very beginnings of a project and forms part of the feasibility phase. The feasibility phase, which is the basis of the decision whether or not a project should be continued, is followed by more detailed and specific investigation, if the go-ahead is given (Table 72).

CONCLUSION

To assess cement raw material resources means to plan and conduct the necessary investigations and to evaluate the obtained findings according to present objectives.

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Table 72 Raw Material investigation as part of a complete project

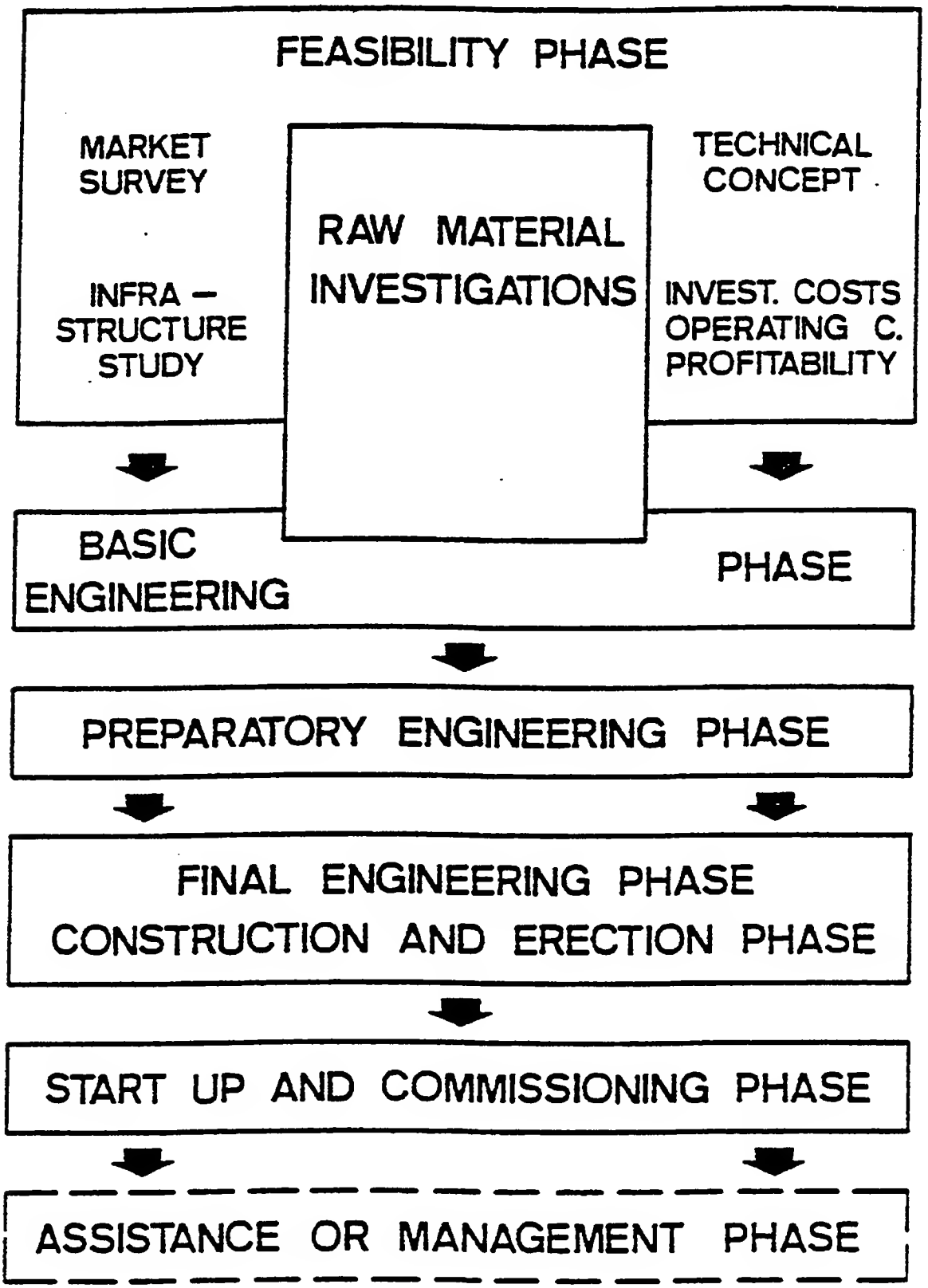

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Table 73 Synopsis of the phases of a project.

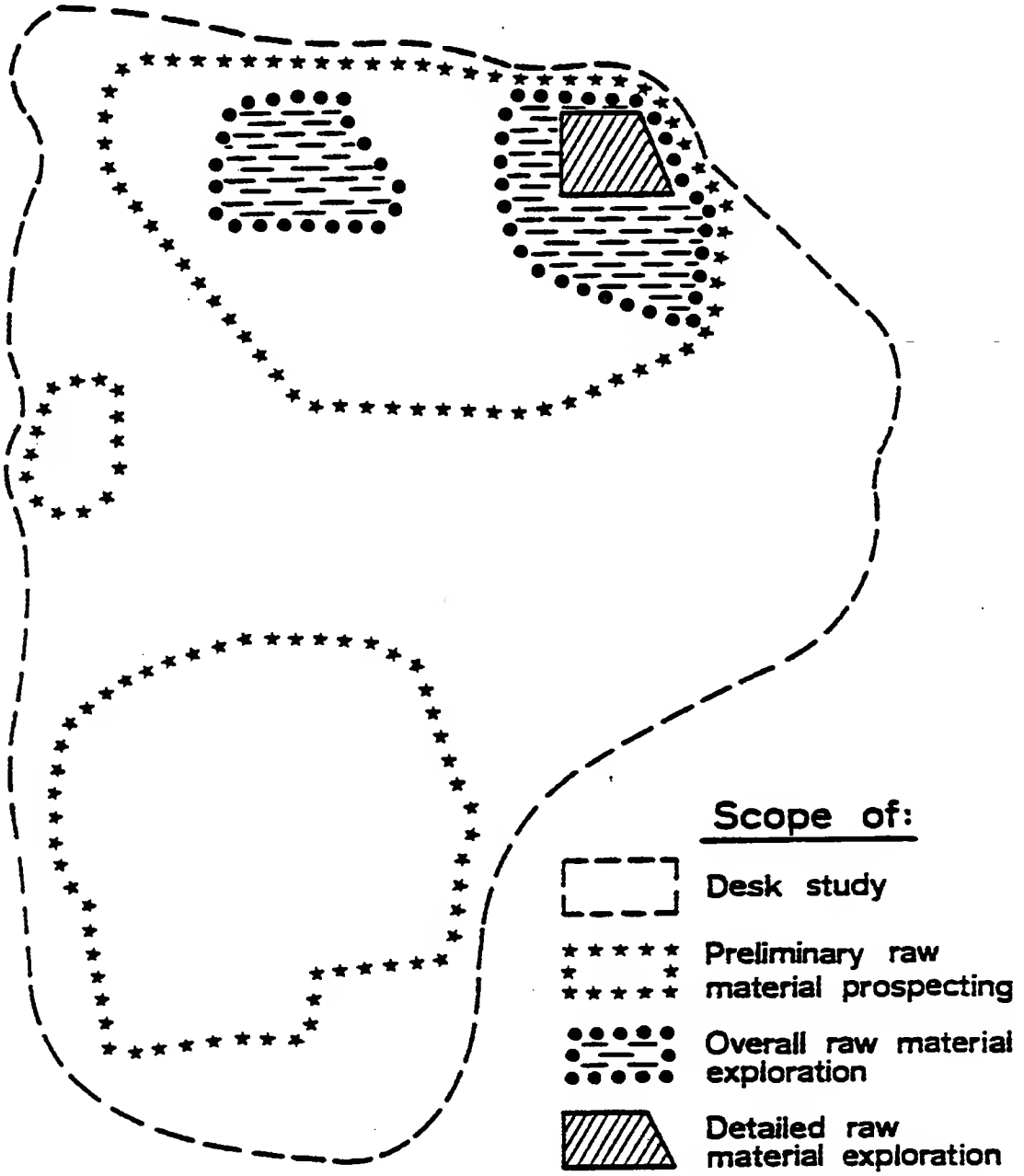

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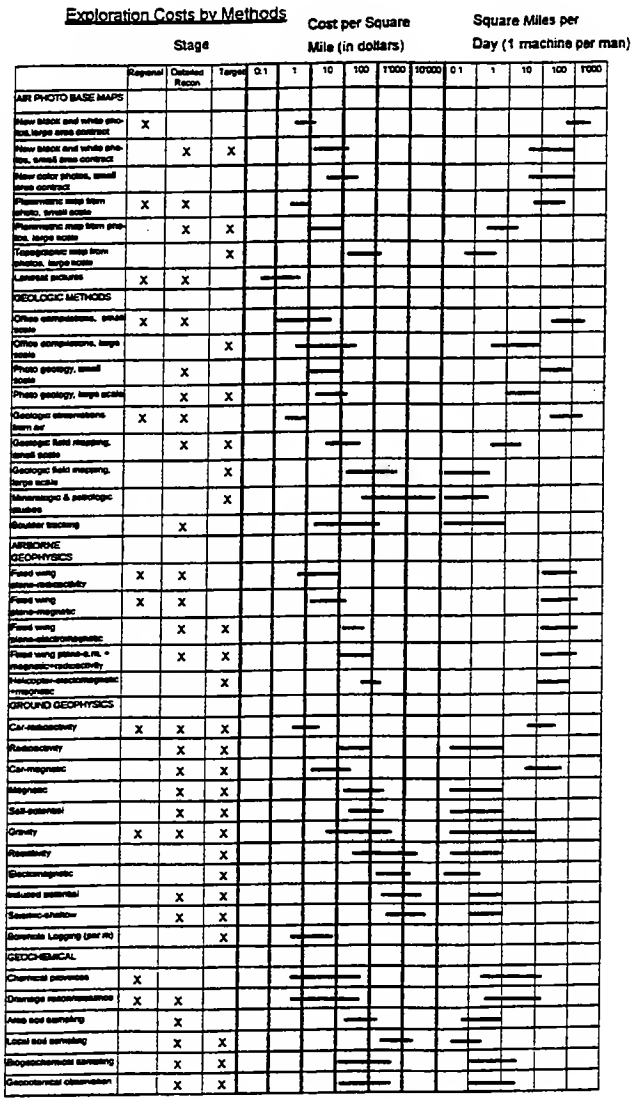
Table 74 Exploration costs by method.



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2. Procedure and Scope of Services

The detailed scope of work will be set up according to the requirements of the project. The following list may serve as a guideline for services carried out in a typical project. Other services are available on request.

2.1 Step 1: Desk study

- Review and desk study of existing documents in HMC
- Compilation and evaluation of all existing documents, geological maps, aerial photographs, chemical analysis and reports which might be provided by the client
- Geological reconnaissance
- Critical examination of all acquired documents in view of the selection of deposits for investigation in the second step
- Evaluation of the results and establishment of a report

2.2 Step 2: Preliminary field investigations

- Fieldwork in the selected deposits
  - Collection of representative samples by channel sampling, pitting or trenching depending on the type of exposure
  - Geological reconnaissance field work
  - Preliminary determination of the geological structures of the deposits
- Preliminary reserves calculation
- Complete chemical cross-check analyses and determination of the mineralogical components on selected samples
- Evaluation of the chemical data
- Preliminary design of the raw mix
- Evaluation of the results and establishment of the report

2.3 Step 3: Overall field investigations

- Elaboration of technical specifications to enable the client to tender for a drilling contractor and a chemical laboratory
- Elaboration of optimal drilling programme (maximum information for minimal amount of drilling) in the raw materials deposit
- Elaboration of guidelines for lithological description of the drill hole cores, sampling of cores and sample preparation
- Initiation of the drilling campaign and permanent supervision of drilling
- Geological mapping and elaboration of the geological structure of the most suitable raw materials deposit
- Complete chemical cross-check analyses on selected samples during the drilling campaign
- Technological testing: mineralogy, grindability, abrasiveness, burnability
- Determination of the design of optimal raw mix
- Geological reserves of the deposit

GEOLOGICAL SERVICES AND RAW MATERIALS MANAGEMENT

The objective of Geological Services and Raw Materials Management is to ensure the availability of raw materials for the manufacture of cement.

The potential deposits of limestone, marl, gypsum, pozzolan or other raw materials will be investigated in order to fulfill the following requirements:

- quality suitable for cement manufacture
- sufficient reserves
- optimised quarry planning for the most economic utilisation of the deposit
- consideration of environmental impacts of quarrying

1. Services available at HMC

In order to realise prospecting in a cost effective manner, and to acquire at the same time a sound knowledge of the raw materials deposit, it is proposed to follow a stepwise approach in the exploration and evaluation work.

- **Desk study** based on the available geological maps, geological reports and if necessary satellite pictures for delimiting potential deposit of limestone.
- **Preliminary field investigations** in order to determine the quality of the limestone by surface sampling and to appraise the potential reserves.
- **Overall field investigations** on the most favourable deposit with a few drill holes and possibly geophysical measurements for the overall structural and chemical assessment of the deposit. This step will prove the feasibility of the project from a raw materials point of view.
- **Detailed deposit investigation** by means of a larger number of drill holes to establish an accurate description of the deposit. The data created will serve as a basis for a deposit inventory as calculated in the next step.
- **Computer aided deposit evaluation (CADE)** to accomplish the evaluation and interpretation of the drill hole data and all other information. The computation of a block model results in an unbiased, dependable, detailed and reproducible deposit description.
- **Quarry scheduling optimisation (QSO)** in order to use the deposit in the most economical way. This cement-specific software allows simulation of the mining development in order to ensure a steady and homogenous supply of raw materials over the longest possible lifetime, at lowest possible cost.
- **Quarry engineering and design (QED)** to make operational and environmental planning fully transparent to all interested parties involved.
- **Selection of main mining equipment** to determine the suitable size and number of units required for an efficient operation of the quarry.
- **Technological testing** confirms the suitability of the raw materials. The service is part of the overall and detailed field investigation.

To summarise, "Holderbank" Management and Consulting Ltd. offer a comprehensive range of services including all aspects of raw materials investigation and testing (in our own laboratories). The work is carried out by a team of skillful specialists (geologists, geoststatisticians, chemists, mineralogists, mining engineers, mathematicians) having a large experience in the exploration of raw materials for more than 200 cement projects.

➢Optional: geophysical measurements in order to precisely define the ore body boundaries and to determine the overburden thickness. These measurements will be carried out if necessary.

➢Comprehensive report, comprising all main results and conclusions

2.4 Step 4: Detailed deposit investigation

➢Elaboration of optimal drilling programme (maximum information for minimal amount of drilling) in the raw materials deposit

➢Permanent supervision of drilling

➢Detailed structural evaluation of raw materials deposit

➢Complete chemical cross-check analyses on selected samples during the drilling campaign

➢Technological testing: mineralogy, grindability, abrasiveness, burnability

➢Determination of the design of optimal raw mix

➢Geological reserves of the deposit

➢Optional: geophysical measurements in order to precisely define the ore body boundaries and to determine the overburden thickness. These measurements will be carried out if it is necessary.

➢Comprehensive report, comprising all main results and conclusions

2.5 Computer aided deposit evaluation (CADE)

➢Statistical evaluation of drill hole data

➢Computation of block model

➢Calculation of reserves in terms of raw mix

➢Deposit display by means of coloured perspective views

➢Development of strategic medium and long-term mining plans

2.6 Quarry scheduling optimisation (QSO)

- Software licence
- Implementation and training on site
- Development of detailed mining plans
- Production scheduling and control

2.7 Quarry engineering and design (QED)

- Optimal open pit design
  - slope angle, bench geometry
  - number of benches
  - haul ramps
  - access road
  - crusher location
  - camouflage walls
- Detailed maps
- Visibility analysis
- Realistic perspective views of quarry development and rehabilitation
- Computer animation

2.8 Selection of main mining equipment

- Type and size of trucks
- Type and size of crusher
- Investment and operation costs

2.9 Technological testing

- Mineralogical composition
- Grindability of raw materials and raw mix
- Abrasiveness of components
- Burnability of raw mix
- Other tests as required

Chapter 10

Case Study

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Case Study

The case study is not included here

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Chapter 11

Raw Materials Management

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Raw Materials Management

Clemens Mann, HMC/CTS/MT

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SUMMARY

The securing of raw materials, and their optimum utilisation, are essential factors for our industry, and they represent strategic competitive advantages. It is therefore necessary to have an effective and efficient system for the management of raw materials.

The general objectives of the management of raw materials are to ensure a continuous supply of the required raw materials at lowest possible costs and over the longest possible lifetime.

More than 20 years ago, "Holderbank" commenced developing a computer-aided system for this task, which consists of four main elements:

1) inventory of the raw material resources using a block model of the deposit(s) (CADE),
2) strategic long-term planning for optimum utilisation of raw materials (QSO-Expert),
3) quarry design and visualisation (QED) and
4) daily production scheduling (QuarryMaster).

"Holderbank" now possess a complete cement specific planning system, ranging from exploration to excavation, which has since been accepted as a high level tool in the industry.

The assessments carried out by "Holderbank" resulted in total yearly savings of more than 100'000 to over 1 million USD . This corresponds to savings of about 0.1 to 0.3 USD/t raw meal. This result was achieved by reducing the waste removal and the costs of correctives, and by increasing kiln OEE with a more uniform raw mix composition.

1. THE NECESSITY OF RAW MATERIALS MANAGEMENT

Raw materials are essential for our core business activities: production of cement and aggregates production, and even though they may represent a relatively small part of the total cement production cost, they can have a significant influence on the production cost and product quality. Another important factor is that the raw materials are not renewable, and new, suitable deposits are of limited availability. Therefore, securing of raw materials supply and their optimum utilisation are of essential for our industry, and offer a strategic competitive advantage. Without prudent, forward-looking planning, not only the profitability, but also the medium and long-term position of a cement manufacturer are placed at risk.

Raw material planning itself is exposed to continuously increasing requirements. The cement industry has now attained such a level of complexity where cost-effective planning and production is no longer possible without the use of computerised tools.

The increasing complexity in the field of securing and providing raw materials for cement is caused largely by the following factors:

- higher raw material demand by larger kiln capacities, with
- ever stricter quality specifications for raw mix composition
- more complex raw mix compositions (alternative raw materials and fuels)
- greatly increasing environmental regulations,
- continuously rising costs, and
- growing number of cement types and rapidly changing product specifications.

All of which necessitate the accurate evaluation of the deposit and reliable planning of the exploitation. Because of the large amount of data from exploration and production, which must be processed continuously, and the large number of requirements and constraints which must be taken into account, computerised tools are essential.

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For these reasons, "Holderbank" started to develop and apply a computer-aided system for the management of raw materials more than 20 years ago. Deposit modelling systems were introduced in 1977, followed by QSO (quarry scheduling optimisation) in 1982. The latest development in this system is QuarryMaster launched in 1998, which is a tool for daily production scheduling.

2. "HOLDERBANKS" SYSTEM OF RAW MATERIALS MANAGEMENT

In order to fulfill the objectives

- of securing a constant and continuous supply of raw materials for the plant, and at the same time
- the cost-effective utilisation of the available raw materials, i.e. with the lowest possible costs over the longest possible life

in the best possible way, the "Holderbank" system consists of four main elements:

- an accurate inventory of the raw material resources using a model of the deposit (CADE),
- strategic medium and long-term plan for optimum raw material utilisation (QSO-Expert),
- quarry design and visualisation of quarry development (QED) and
- daily production planning (QuarryMaster).

The gradual development of combined application of the modules, has also lead to an array of instruments which provide efficient and optimum solutions to the problems listed above, ranging from strategic long-term planning to daily production planning. The most frequent questions about the characteristics and benefits of the most important modules will be dealt with in the following sections.

2.1 Block model (CADE)

The object of calculating deposit or block models is to obtain an objective, reliable, and reproducible description of the deposit. A block model is a complete three-dimensional description of a deposit (Fig. 1). To calculate the model, the deposit is subdivided into a large number of small blocks. Each block represents a quantity of quarried material corresponding to a week or a month's production. Based on drillings and other exploration data, interpolation methods are used to assign the contents and other parameters to each block. In other words, a variety of information such as drill results, geological investigations, geochemical and geophysical results, etc., are converted into the best possible consistent description of the deposit.

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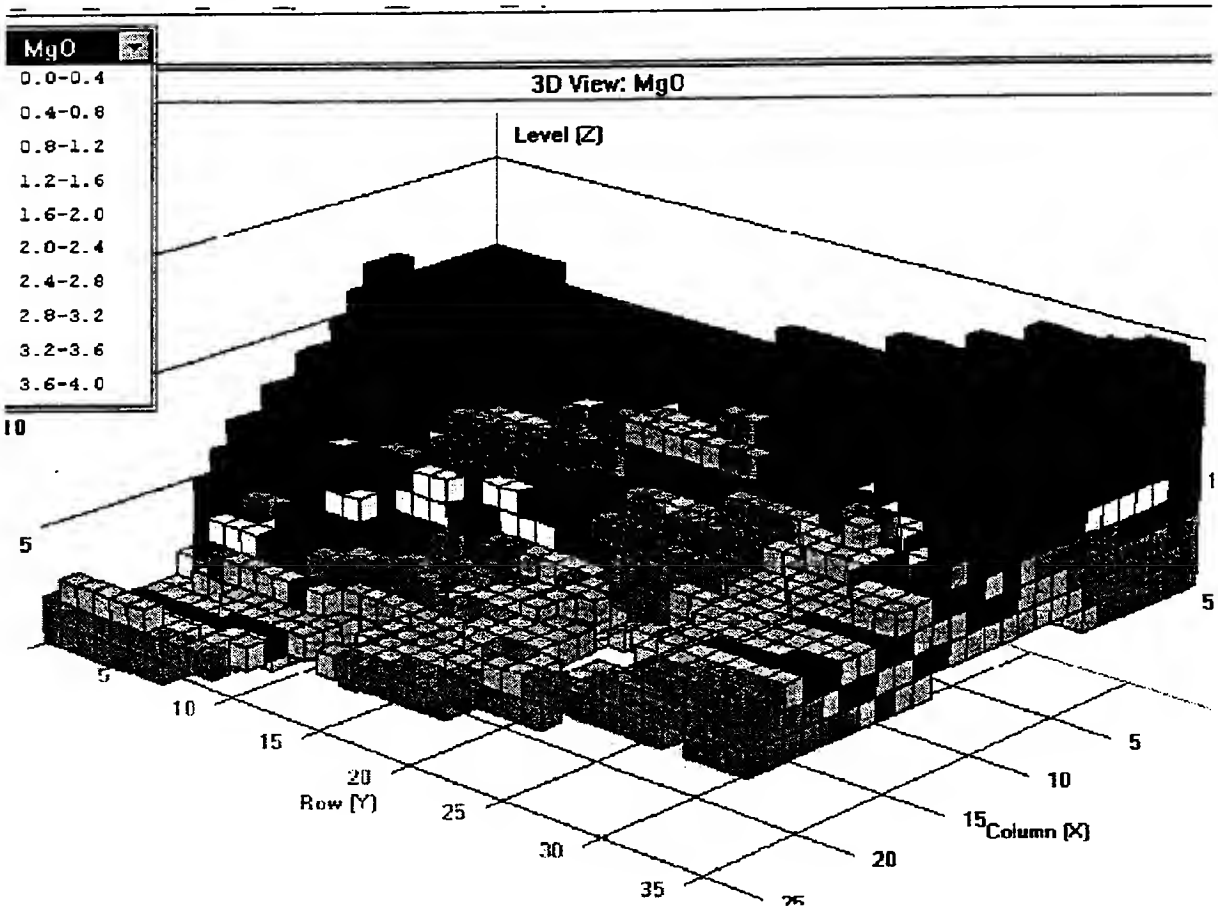
Fig. 1 Block model of a clay deposit.



The resulting block model is an accurate inventory of the deposit, which describes the expected quality and quantity for each point (block). It now represents one of the most effective instruments for raw material evaluation.

2.2 Quarry Scheduling Optimisation (QSO)

The block model only describes the state of the deposit and, in particular, the raw material reserves. The next question is: how this given deposit may be used in the best possible way? To answer this question, QSO is used to calculate geological and raw mix reserves whilst considering different raw mix requirements and different corrective materials. In this way, the influence of different raw mix compositions on the utilisation of the deposit can be investigated, and finally long and medium-term exploitation plans can be developed. QSO uses in-built operations research techniques to determine the optimum solution. For the optimisation procedure, QSO considers not only the chemical raw mix requirements, but also mining rules, which are specified by the user. This results in chemically correct and technically feasible mine plans.

The first practical application for a "Holderbank" group company took place as early as 1982. The ideas developed at that time, and the results achieved, were encouraging and showed the direction to be followed. The application of these methods was, however, relatively expensive as it required large computers and well-trained personnel. Due to the fact that QSO is a planning tool frequently used on-site, the breakthrough happened in 1985

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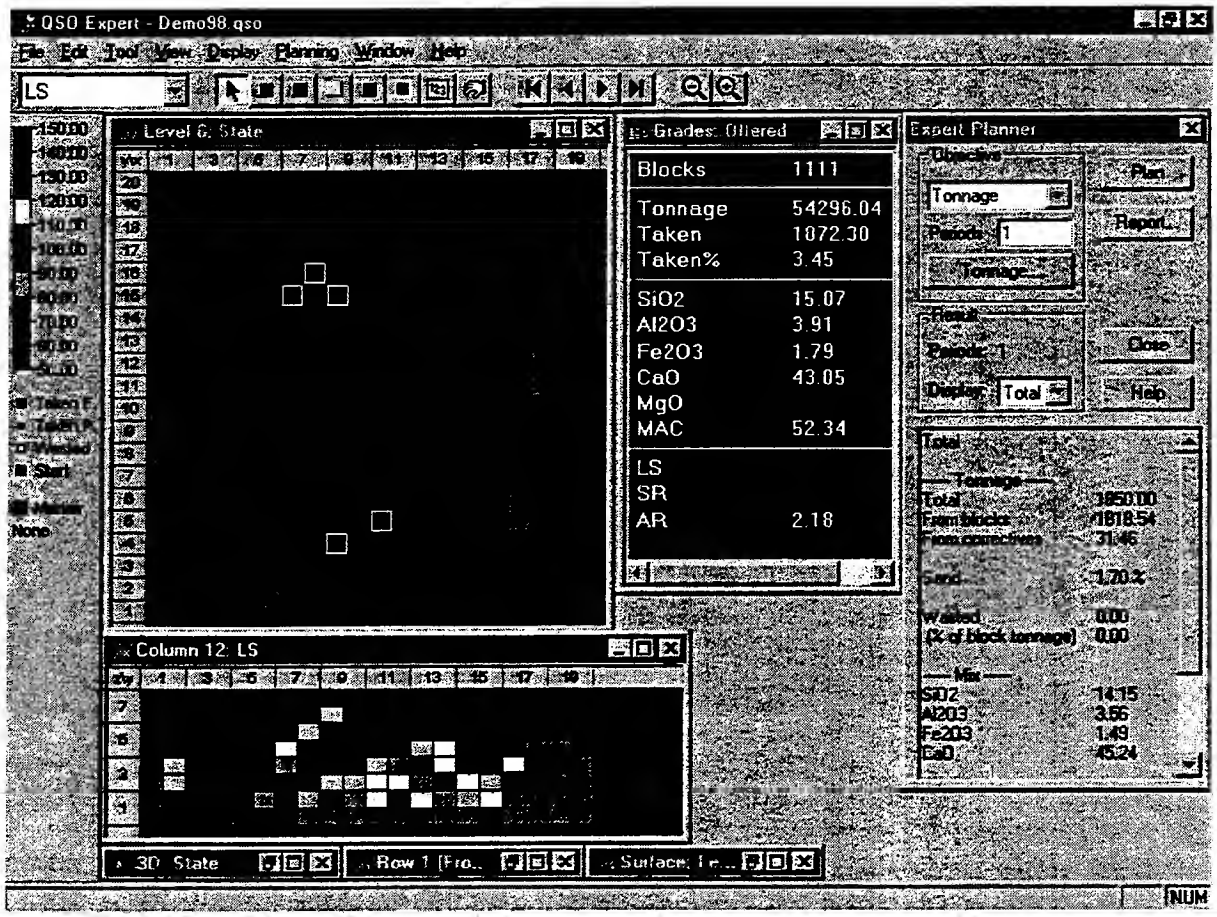
because of the wider spread use of PC's. QSO is now in its fourth generation, which is known as QSO-Expert for Windows.

Fig. 2 Example of a typical QSO-Expert screen.



2.3 Quarry Engineering and Design (QED)

In addition to the optimum raw mix composition, some technical design features for the quarry have to be determined. These are bench height, berm width, general mining strategy (e.g. top to bottom) and mining method (e.g. truck haulage, in-pit crushing or glory hole system).

QED also includes the integration of these technical features in the quarry plans, such as haul roads, ramps, dewatering ponds, waste dumps etc. In this way, the optimum long-term mining strategy is transferred into detailed technical quarry plans.

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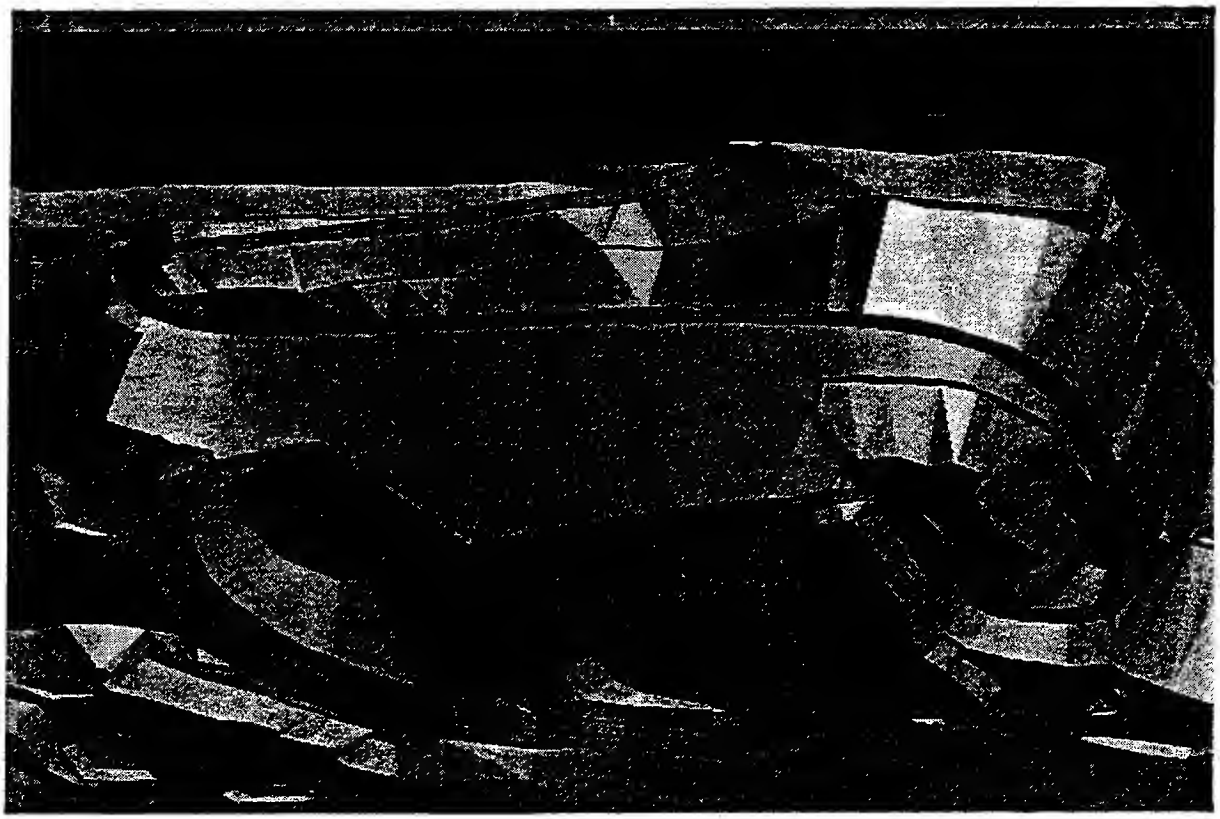
Fig. 3 Haul road design for a quarry.



In order to obtain a mining permit for a deposit, it is of utmost importance to show what the future appearance of the quarry will be. Therefore, mine plans representing the future quarry development are elaborated with QSO. These plans are then transformed into photo-realistic 3D-views using the technique of digital terrain modelling, whereby the appearance of the quarry at any given time can be realistically shown, refer the proposed quarrying plan (Fig. 4). For this task, special commercial CAD programmes and animation software are used.

With such detailed representations of the quarry, it is possible for the authorities, and more important for the public, to visualise the quarry development in an easily understandable way.

The same technique can also be used to analyse the visual impact of the quarry on the surroundings. The aim of such a visibility analysis is to achieve the increasing objective of blending the quarry as inconspicuously as possible into the landscape.

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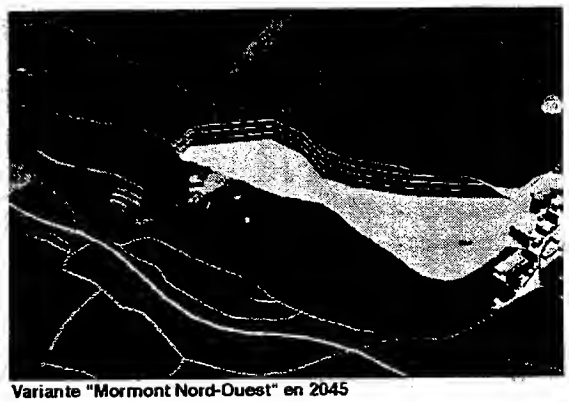
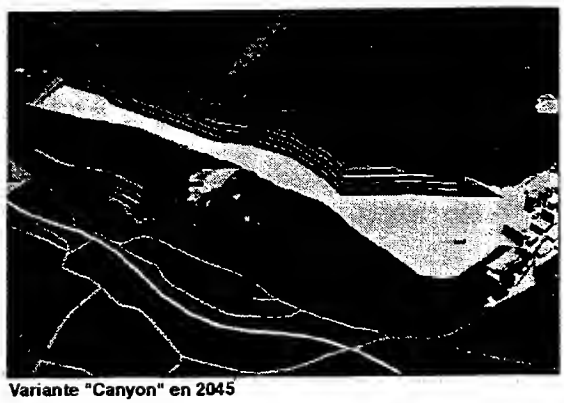
Fig. 4 Three-dimensional representation of the development of a quarry.

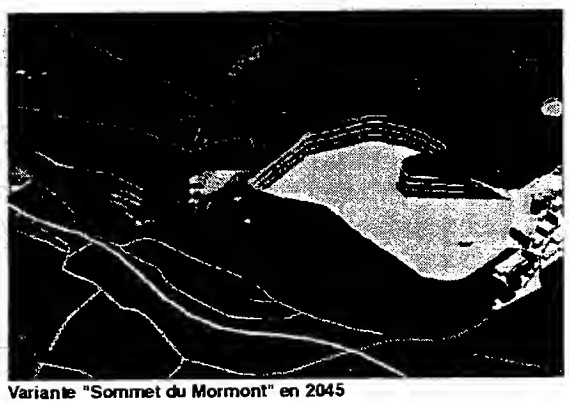
Variante "Mormont Nord-Est" en 2045



Variante "Mormont Nord-Ouest" en 2045



Variante "Canyon" en 2045



Variante "Sommet du Mormont" en 2045

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2.4 Production Scheduling (QuarryMaster)

The most uniform raw mix with the correct chemical composition is essential for an efficient plant operation and a high product quality. Using QSO, the optimum long-term strategy is determined and long and medium-term mine plans are developed. The shortest time is normally not less than one year due to the size of the blocks. However, for daily operations, it is necessary that production for a much shorter period of time fulfills the process requirements. That means every preblending pile, respectively the quarry production for every day, has to be within the limits. For this very purpose QuarryMaster was developed. It is a tool for daily or weekly production planning in the quarry.

QuarryMaster assists the quarry manager in deciding how many tons have to be mined from which blast in order to produce the correct raw mix, with the given equipment and manpower whilst staying in line with the long-term mine plan. QuarryMaster is based on a digital terrain model of the quarry, which means the quarry topography can be represented in all necessary details.

Short-term planning or production scheduling requires more accurate data with respect to the raw material situation. The average values from the block model, which are adequate for long and medium-term planning, are not sufficiently reliable. The daily fluctuations require special consideration, which now necessitate a more detailed database. Therefore, analyses data from blasthole dust, sampling stations, or cross-belt analysers are used for production scheduling.

QuarryMaster is an important tool for quality control. In fact, it represents the first step in a modern production process, as the raw mix quality must be planned at the very beginning, which means right in the quarry. There is a strong interaction between production scheduling and quality control, and the use of QuarryMaster ensures the link between planning and control.

Taken as a whole, it provides uninterrupted co-ordination from deposit evaluation right up to excavation planning, and production control and quality checking. Nowadays, this is a precondition to ensure the required quality and quantity of a product over a fairly long period, with lowest possible daily fluctuations. At the same time best possible use of the available raw material reserves is made.

2.5 Applicability

From a mathematical point of view, the calculation of optimum excavation plans is a demanding task. Calculation of the block models is also a computationally intensive undertaking, which is not at all simple. Twenty years ago, it was necessary to have large and expensive scientific computers and well-trained personnel.

Nowadays, with the powerful low-cost PC's, most of these tasks can be carried out in close proximity to the operation by the quarry personnel themselves. The drill hole databank, the block model, and the excavation planning are prepared by experts, and QSO-Expert and the QuarryMaster are implemented on-site. After a relatively short training phase (1 – 2 weeks) the quarry personnel can operate and use the tools to their full extent.

It is important to remember that in addition to the high level of appropriate software products, one of the success factors in implementing raw material planning systems is experience. Proof of "Holderbank's" expertise in this field is given with the successful completion of more than 150 projects, which have been carried out using computer aided raw material planning. The present software system can be described as having been developed "by a cement manufacturer for a cement manufacturer".

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3. COST EFFECTIVENESS

Under the continuously mounting pressure of costs, the question of cost savings using computer-aided raw material planning arises with increasing regularity. In addition to the indirect benefits - such as, better and more reliable knowledge about the deposit, rapid and optimum results in evaluation and planning, quantification of risks, evaluation of different scenarios - the main interest lies in the measurable savings in production costs. Estimates carried out by "Holderbank" in the past showed potential annual savings of up to a few hundred thousand USD in various situations. Relative to one tonne of raw mix this normally means cost savings of 10 - 30 cents. These were achieved by reductions in the quantity of overburden and in the costs of correctives consumption, and through increased OEE of the kiln because of a more uniform raw mix composition.

The costs of implementing a computer-aided raw materials management programme amount to between USD 100,000-200,000. Bearing in mind the above mentioned cost savings of 10-30c/tonne of raw material consumed, a payback period of 1-2 years more than justifies the initial investment. So far, over 150 international projects have been carried out for cement lime and aggregate producers all over the world.

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QSO-Expert for WINDOWS
Quarry Scheduling Optimisation
Using Artificial Intelligence Techniques

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1. OBJECTIVES

QSO-Expert represents the most recent innovation in deposit evaluation, mining simulation and production planning. The two main objectives of applying QSO-Expert are:

- secure a steady supply of raw materials from the quarry to the clinker production units
- economic utilisation of the raw material sources, in most cases this means at minimum cost and maximum lifetime.

By means of improved planning, the problem of waste minimisation can be better dealt with, and capacity of mining equipment and their constant load level can be better optimised, as can mining and transport logistics, etc. – to mention only the most important factors.

2. THE METHODS BEHIND QSO-EXPERT

It is important to realise that QSO-Expert is not only concerned with the chemical raw mix requirements. Mining technical and logistical questions can likewise be considered, or may even be the prime objective.

This is due to a novel approach, the combination of conventional numerical optimisation methods with symbolic and heuristic techniques of Artificial Intelligence. A powerful user interface under the WINDOWS environment facilitates easy computation and evaluation of various planning tasks. The results can be interpreted "at a glance", thanks to compact graphical visualisation.

Besides the comprehensive graphical capabilities, the essential innovation is the previously mentioned combination of numerical and heuristic methods.

Computation of solutions, which are chemically optimal and technically feasible with as little user interaction as possible, can be a tedious task, when only numerical methods are employed. Theoretically speaking, there are purely numerical solutions, however, computationally intractable on the PC of today. Hence, to solve the problem - computation of chemically optimal and technically feasible plans - in an elegant way, heuristics ("experiences and rules") must be conceived.

Utilising a heuristic approach potential mining areas are selected (i.e. taken into consideration for mining) and evaluated by means of numerics. The best solution is accepted. Based on it "the next solution is computed" etc. In practice, this represents individual stages of mining progress, the quarry schedule.

Without going into further technical detail: The solutions achieved represent the chemical optimum, whilst mining technical feasibility is warranted. The activity of a human expert is emulated, and in many respects even exceeded. For example in:

- simultaneous consideration of several chemical variables
- special evaluation of mining areas
- quick computation of technically feasible plans
- reproducibility of results and appropriate reasoning.

It should be stressed, that it is not an uncontrolled, fully automatic quarry scheduling that is desired, but an efficient support of the human expert. Tedious, repetitive routine is eliminated, complex chemical and special relations are understood correctly and managed easily.

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3. QSO-EXPERT IN PRACTICE

In particular, QSO-Expert offers the following possibilities.

1) Block Model content: all chemical variables and block states are displayed in form of coloured block maps in an impressive and clearly arranged manner.
2) Geological (in-place) reserves can be calculated quickly in all detail for the total deposit or parts of it.
3) Computation of reserves in terms of raw mix, under consideration of eventual correctives:
   - without mining technical restrictions (chemically optimal)
   - with mining technical restrictions (chemically and technically feasible) and the option to permit waste, if necessary.
4) Development of medium to long-term mining plans.
5) Using blast hole dust analysis data or other continuous production data, short-term production planning on a daily to weekly basis can be achieved. The build-up of batch type or continuous preblending beds can be planned and controlled.

3.1 Graphical User Interface

QSO-Expert is a software package according to the WINDOWS standard. Fig. 1 to 4 show the user interface and provide some additional detail to QSO-Expert in practice (see above).

What are the main differences between QSO-Expert (from 1989) and QSO-Expert for Windows (from 1995)?

Some new features and benefits are:

- simplified programme handling; fully mouse driven
- compatibility with WINDOWS environment (e.g. direct printing on a large variety of plotters)
- better waste handling (optimisation of waste treatment; maintaining constant stripping ratios)
- widely improved block model display (blockmaps to scale; cross-sections; perspective views; display of world co-ordinates and topographical features)
- quick on-line help (context sensitive help utility)
- block models of up to 16'000 blocks can be dealt with, better control of quarry scheduling due to additional advanced mining rules.

QSO-Expert represents the latest state-of-the-art by its combination of Artificial Intelligence with Linear Programming, utilising the advantages of both methods.

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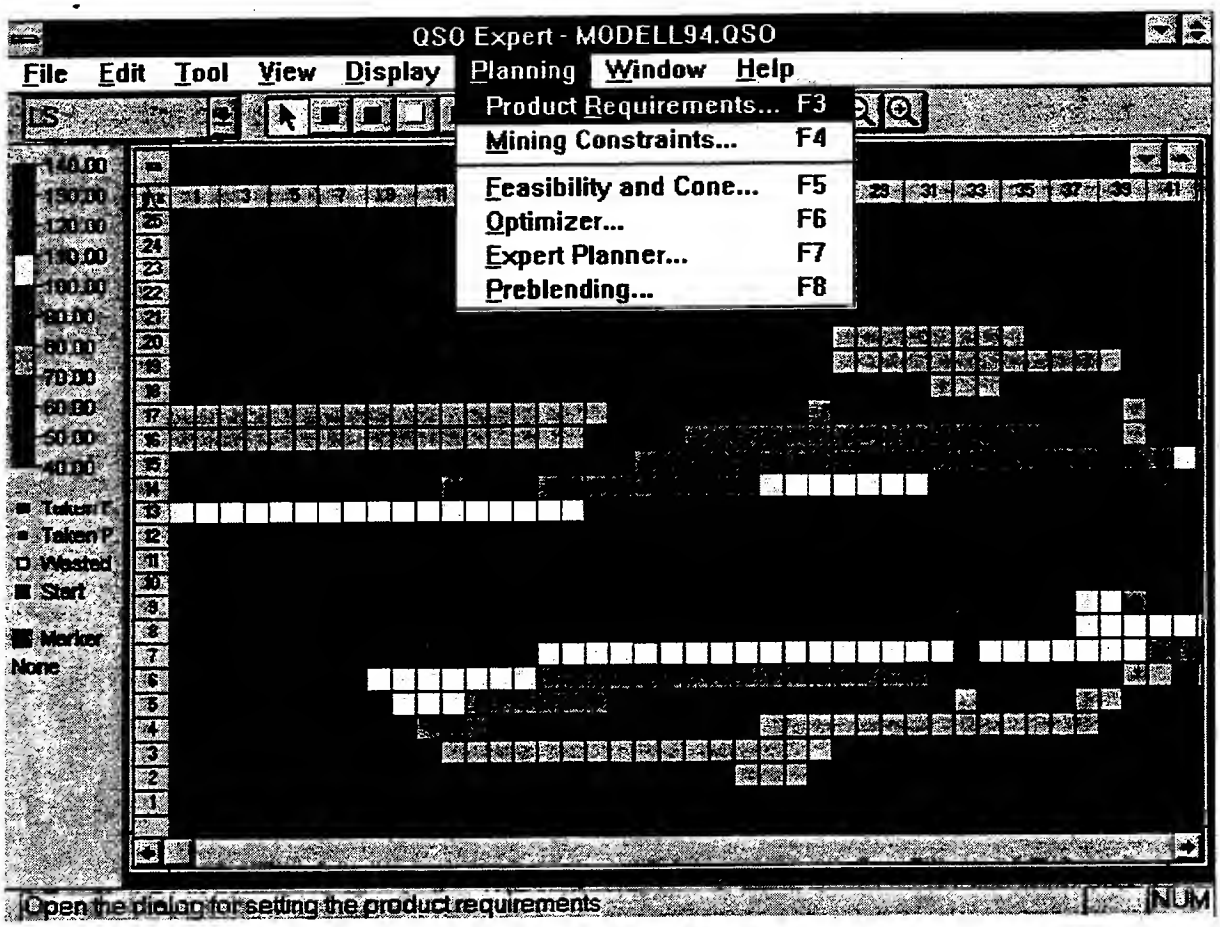
4. EXPLANATION OF FIGURES 1 TO 4

Figure 1:



A certain level of the deposit is represented as a coloured block map. Each square symbolises in this case a block of 60 x 60 x 10 m3. The colours display the lime saturation. Reddish and yellowish colours represent high grades, greenish and bluish ones low grades. Any variable which is contained in the block model can be shown. Thus, the deposit inventory is displayed in an impressive and easily understandable way.

The "Planning" menu is opened and allows the selection of various activities such as: definition of product (raw mix) requirements, specification of mining rules and constraints, checking mining feasibility, computing optimal deposit utilisation strategies, planning actual quarry schedules and doing short-term production scheduling by building-up a pre-blending bed.

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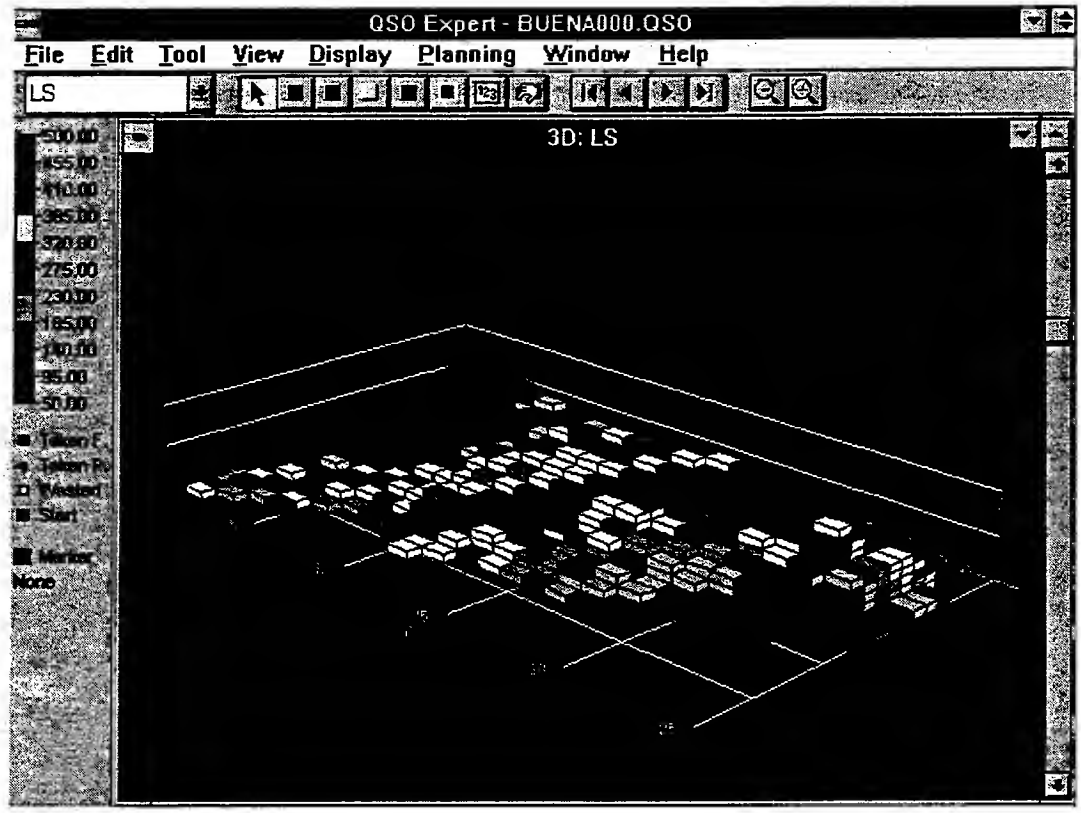
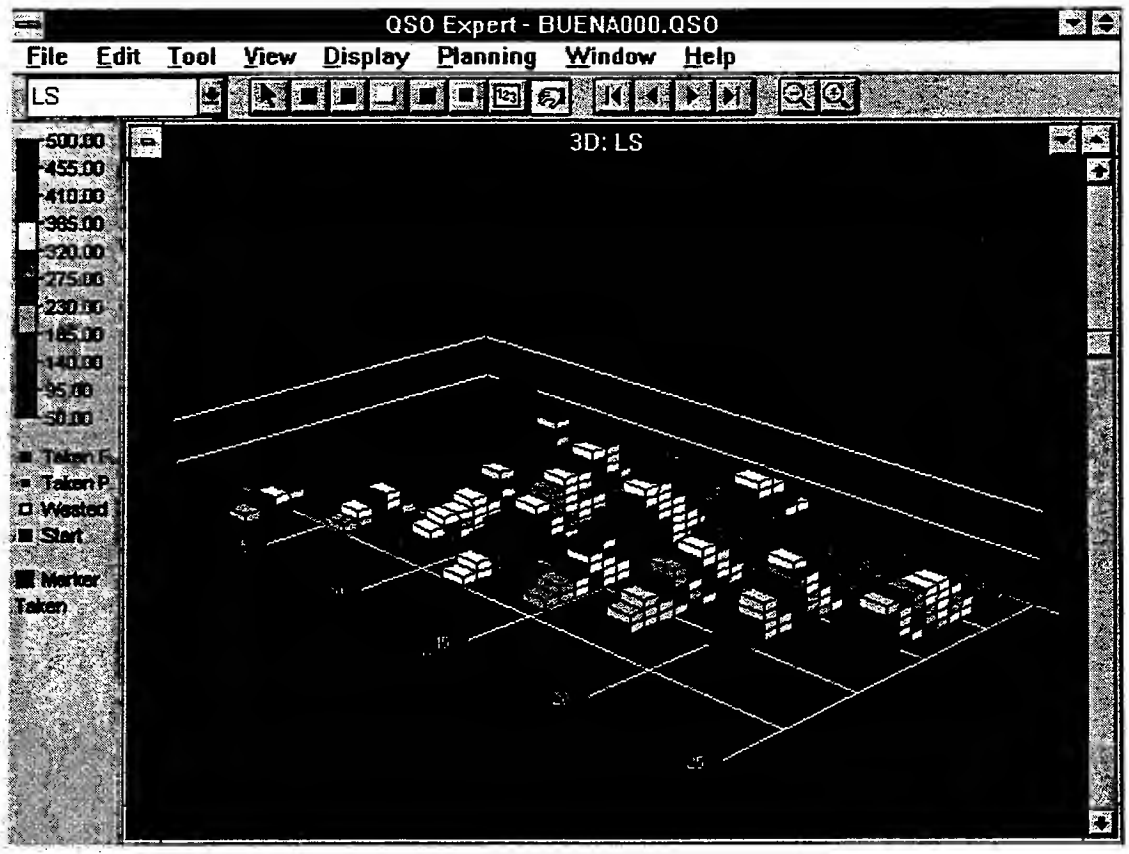
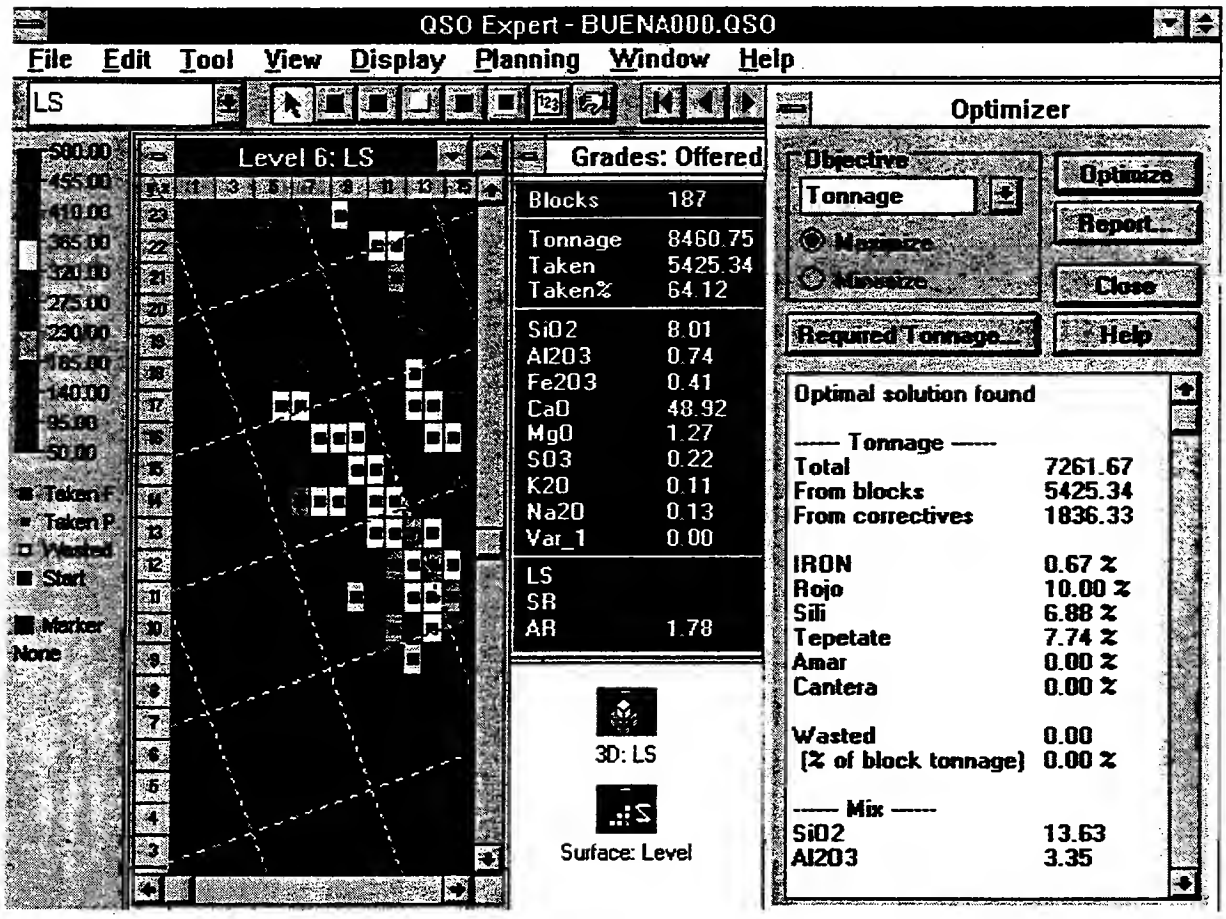
Figure 2: The Deposit in Perspective View





Spectators view the deposit from a specified viewpoint. They see the distribution of LS not only on the surface, but also along the cross sections. A very attractive way to provide insight into often complex deposit conditions.

Figure 3: Results of a Scheduling Run



This figure shows the results of a scheduling run. The window "mix offered" describes the geological reserves on level 6 (left window). The right window "optimiser" describes chemistry and maximum quantity of the raw mix achieved. Quantities of the required correctives (clay, iron-ore and sand, etc.) are reported as well. The dotted blocks (left window) are the blocks which have to be mined to achieve the reported raw mix.

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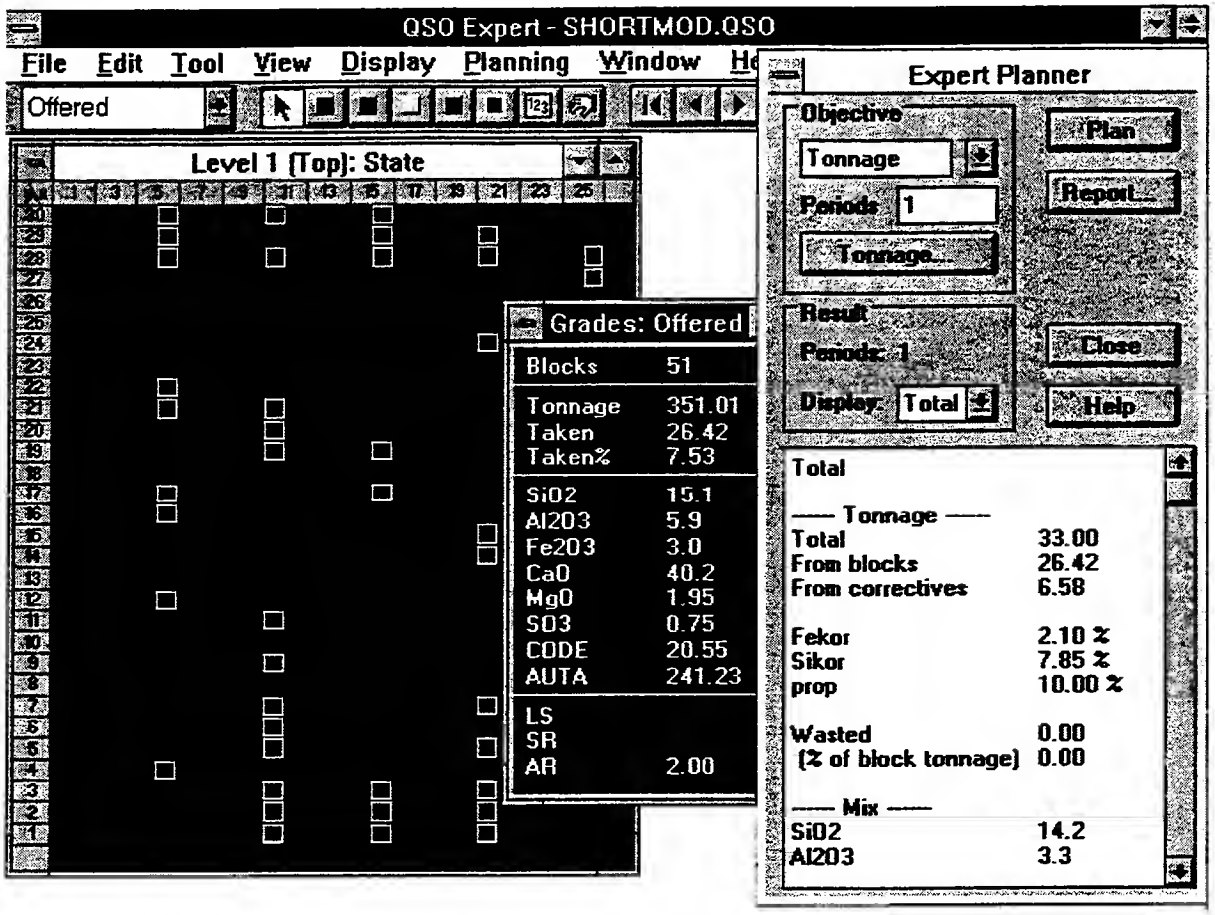
Figure 4: Short-term Planning



Each column in the left window represents a mining face, each red block represents blasted material, known in quality and quantity. There are 50 blasts with 355'000 tons of material (window "mix offered"). The four blocks marked with a black dot plus some correctives result in 33'000 t of raw mix in the desired quality, corresponding to the capacity of one preblending bed.

5. HARDWARE REQUIREMENTS

QSO-Expert runs on PCs with Intel 386 or 486 processors (with arithmetic coprocessor), 4 MB RAM, colour display (VGA) and an A4 colour plotter. Operating system is MS-WINDOWS 3.1 or higher.

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Quarry Engineering and Design

Clemens Mann MT/CTS/HMC.

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1. INTRODUCTION

With QSO-Expert a global strategy for the optimum utilisation of the deposit can be selected. Based on that strategy long term mine plans (periods of 10, 5 or single years) are developed to investigate how the exploitation should proceed so that the optimum strategy can be fulfilled period per period. These QSO plans are then transferred into topographical maps showing the quarry development.

The objectives of Quarry Engineering and Design (QED) can be summarised as follows:

- to set up a technical design for the quarry
- to generate topographical maps of the quarry development
- to include important technical features especially haul roads, dumps, dewatering systems and
- to visualise the quarry development in a most realistic way.

QED also includes engineering work of the quarry planning such

- selection of a quarry system
- haul road design
- dewatering concept

The quarry plans show the future development of the quarry in all necessary detail. They are used for

- strategic decisions (acquisition of property, location of crushers, construction of haul roads and waste dumps, investment for mobile equipment).
- control of exploitation progress
- environmental related planning (rehabilitation, visibility analysis, etc.) and
- the mining permission procedure.

2. QUARRY DESIGN

Before starting with the development of long-term mine plans, some basic design features for the quarry have to be set up. The design features determine the geometrical shape of the quarry. The main parameters are:

- quarry limits
- maximum final depth
- overall slope angle
- bench height

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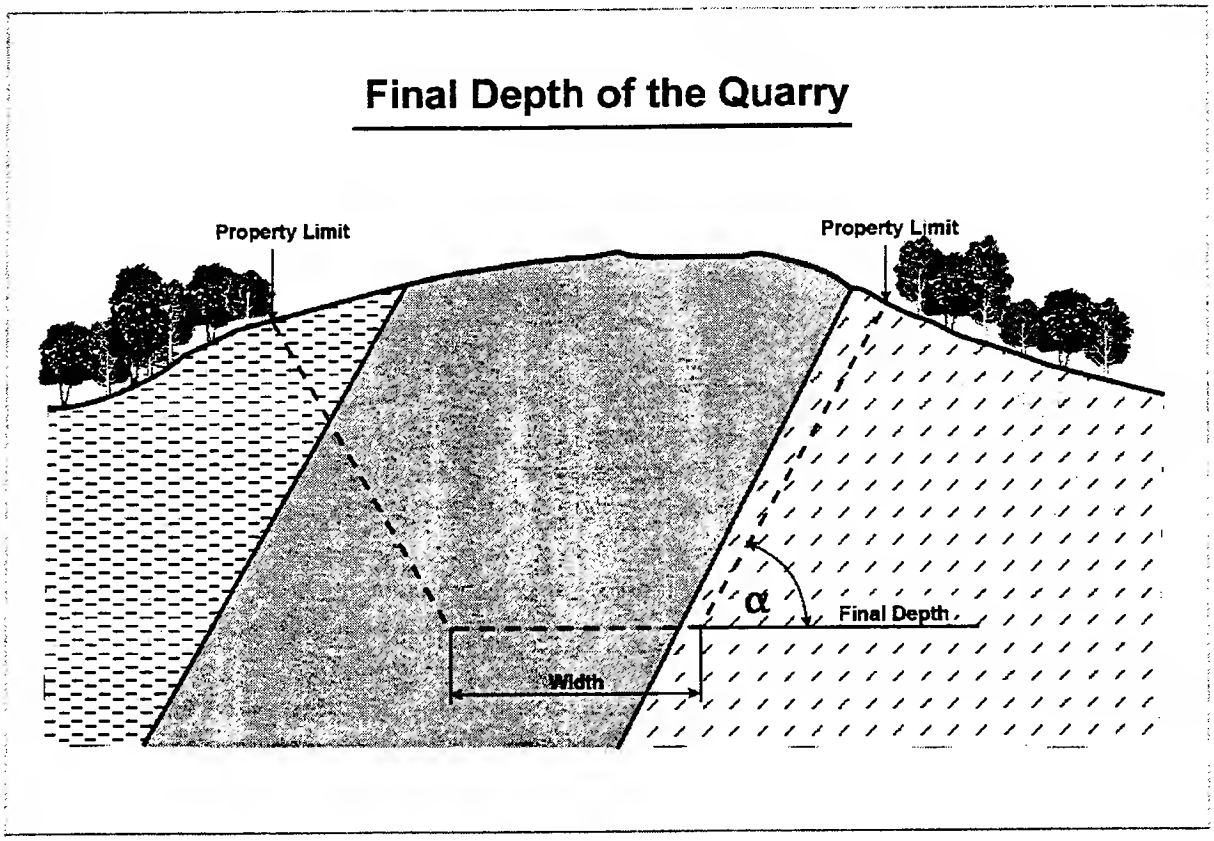
2.1 Maximum Final Depth

The maximum final depth is limited by the surface outline of the mining area, the maximum overall slope angle and the minimum width of the pit bottom (Fig.1). The surface outline of the mining area is generally restricted by external factors such as the concession or permission area, property limits and public installations like roads, railways, power lines etc.

Fig. 1: Quarry limits and maximum final depth



From the aspect of mining cost, the final depth is often limited down to:

- the level of the crusher location to avoid up-hill haulage with the loaded trucks or
- the level of the dewatering drain to avoid pumping.

The final pit depth can also be limited by the authorities (protection of ground water horizons).

2.2 Overall Slope Angle for Final Pit walls

The overall slope angle is the angle between a horizontal and a line that joins the toe of the bottom bench and the top edge of the quarry.

Why is the maximum slope angle for the final pit walls so important? It has a great impact on the economy of the entire project. The steeper the angle the less overburden to be removed, or the greater the amount of reserves to be mined within given surface boundaries. As the slope angle becomes steeper so the risk of slope failure increases.

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The stability of a slope is defined as a safety factor:

$$\text{Factor of safety} = \frac{\text{Sum of resisting forces}}{\text{Sum of moving forces}}$$

The resisting forces depend on the friction angle and the cohesion of the material. As a first estimation, we calculate, for the final pit walls in limestone quarries, an overall slope angle in the range of 50° to 60°. Depending on the exact geological conditions (joints and fissures) and the influence of water, the maximum angle can be lower or higher. Different types of rock have different stabilities. Therefore, a severe geotechnical study is recommended to determine the overall slope angle. This is also most relevant for the waste dump.

Slope angle for a bench system during quarry operation is normally much lower than the final angle because a certain bench width is required for the manoeuvrability of the mobile equipment.

2.3 Bench Height

The bench height is one of the most important design parameters of a quarry, because it strongly influences all sectors of the mining operation. Therefore, the selection of the correct bench height is an important task for the planning engineer, in order to optimise the quarry operation.

Of course the bench height has to be selected in accordance with mining laws or safety regulations which limit, in many countries, the bench height to a certain maximum (e.g. Switzerland 40 m).

Quarry and pit operators often have divergent views about bench height; while some favour high benches from 20 to 30 m, others feel very strongly that bench height should be designed from 10 to 15 m.

Lower bench height has the following advantages:

- greater selectivity of rock excavation
- greater drillhole accuracy
- higher penetration rates over the hole

Their disadvantages are:

- construction and maintenance of more benches
- more pre-stripping at lower working slope angle
- greater difference of haulage level for down-hill haulage
- limited to a certain drillhole size
- more drill down time for shifting the drill rig
- more subdrilling
- more boulders from the top of the benches
- higher consumption of detonators

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A general value of the optimum bench height which is valid for all quarries can not be calculated. The suitable bench height has to be selected individually for each case. However, experience in recent years has shown that in our industry there is a tendency to lower benches with a height of about 10 to 15 m.

3. QUARRY ENGINEERING

QED also includes engineering work for quarry planning such as

- selection of a quarry system
- haul road design
- dewatering concept

3.1 Quarry System

In our quarries, we have in most cases a system which consists of a drilling/blasting and a loader/truck operation. Sometimes the quarry planning requires also the selection of a different mining system. Because the mining system has a strong influence on the way in which the quarry can be developed and what quality can be produced, it can not be selected without considering the quality aspects.

3.1.1 Ripping

As an alternative to a drilling/blasting operation ripping can be used in soft rock (Fig.2). Ripping has the advantages of

- no permit, transport, storage and handling of explosives are necessary
- no ground vibrations, no air blast occur
- no specially trained personnel required and
- a continuous operation

The main question is: Can the rock be ripped? As a rule of thumb it can be stated that material up to a seismic velocity of 2.000 m/s is rippable, but other factors such as fractures and weaknesses, weathering, stratification and brittleness are also very important.

A rock mechanic analysis, a geological site investigation and a seismic analysis may give good indications, but only a site test can prove the rippability of the rock. Finally, a cost comparison has to show whether ripping provides cost benefits.

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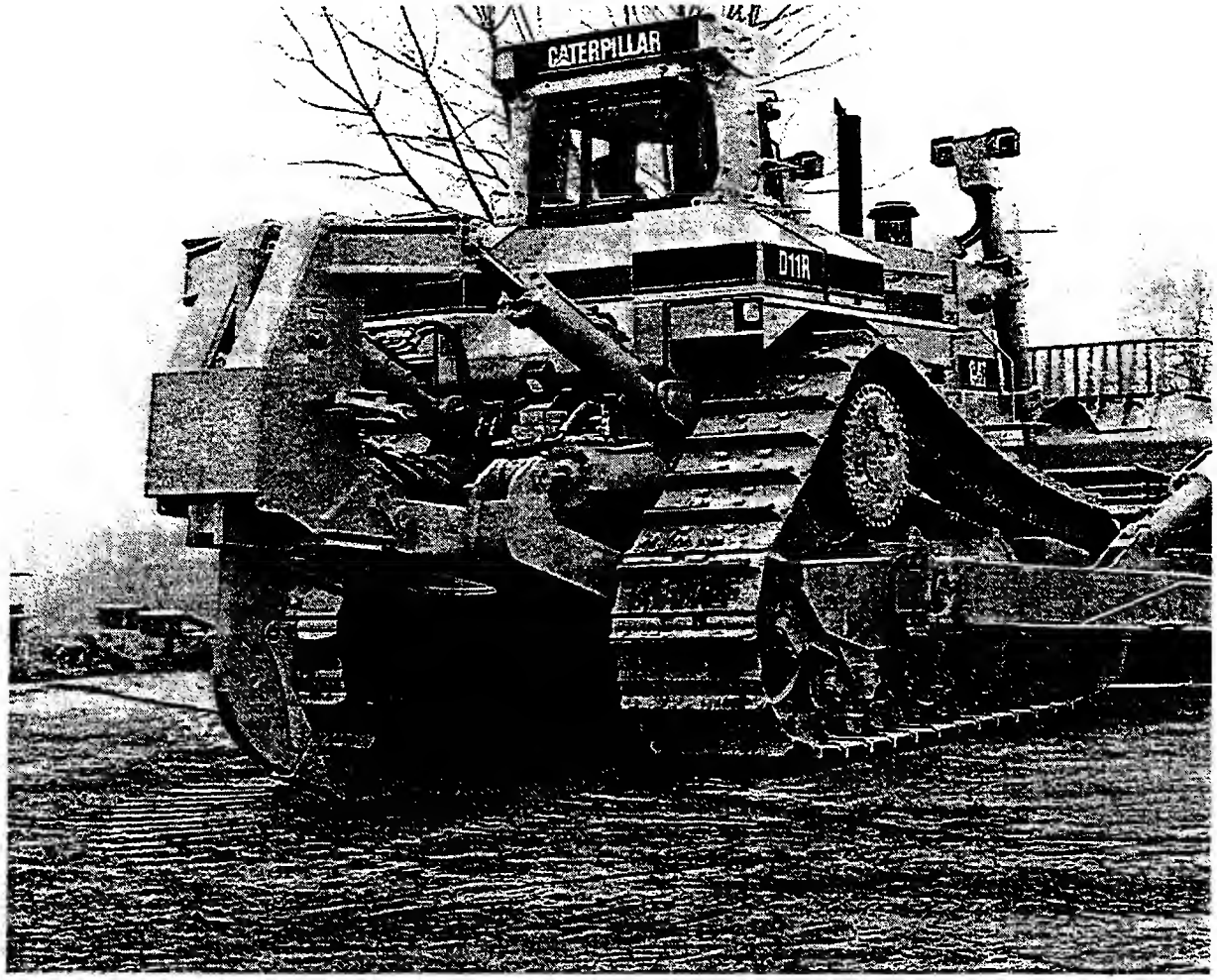
Fig. 2: Dozer with ripper



3.1.2 Dozing

Normally the haul road leads to all benches, so that the rock can be loaded and transported from each bench. As an alternative, the muckpile is pushed by a dozer over the crest of the bench and then loaded at the pit bottom into trucks.

The advantages of this method are that the haulage distances and the required number of trucks are reduced, and the construction of a wide haul road to the upper benches is not necessary. This results in low mining costs. The disadvantages are: reduced safety, poor quality control by segregation, selective mining is difficult, less flexibility in face advance, more dust and more fines.

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3.1.3 In-Pit Crushing

Haulage in the quarry is normally done by large trucks. The alternative is a combination of an in-pit crusher and conveyor belt.

While a truck system offer:

- high flexibility
- high availability
- low initial investment

it has relatively high operating costs due to:

- high energy consumption
- each truck needs an operator
- high maintenance effort (engine, tyres, brakes etc.)

Furthermore trucks necessitate the construction of a wide and smooth haul road, and are less safe, particularly at night and with poor weather conditions.

For this reason, many large scale open pit mines have installed in-pit crushing systems in recent years, to reduce their truck fleets and haulage costs.

There are several kinds of in-pit crushing systems:

- fully mobile system which is fed directly at the face (Fig.3)
- semi-mobile system which is fed by wheel loaders (Fig.4)
- semi-mobile system which is fed by trucks

If a stationary crusher is used, the location of the installation is of primary importance, as it determines the quarry operation over a long period of time.

Fig. 3: Fully mobile crusher loaded by face shovel

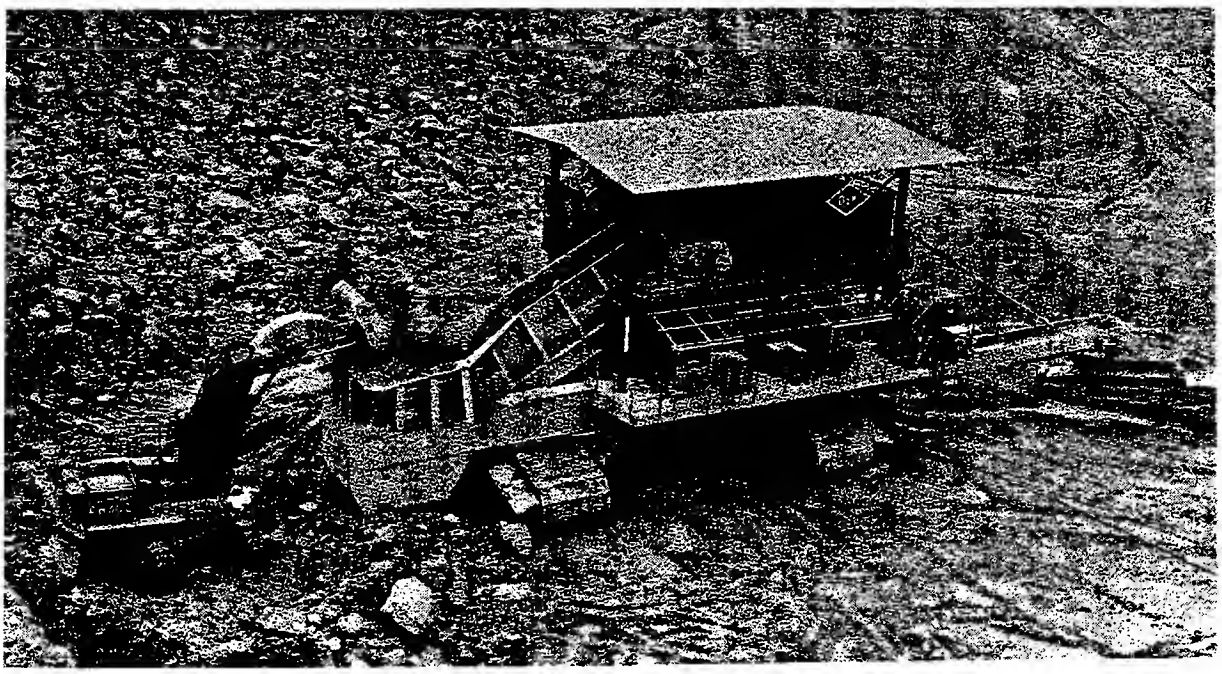

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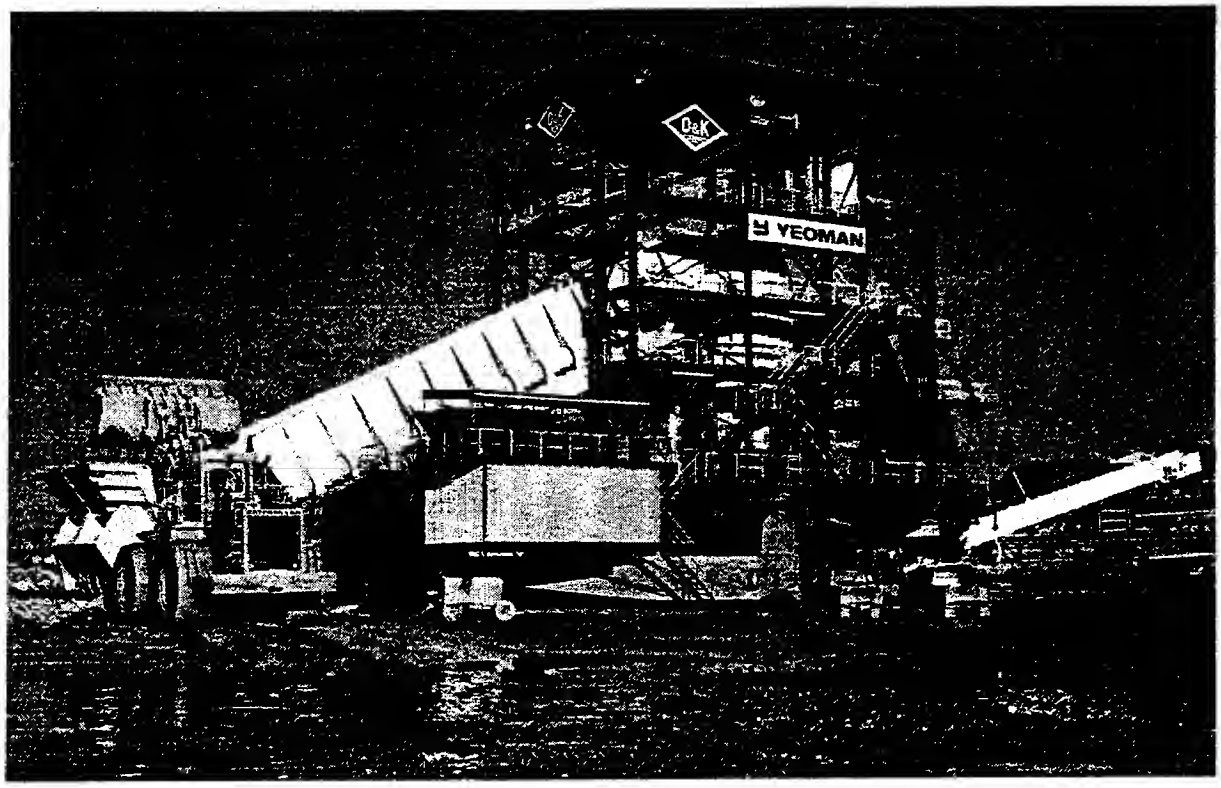
Fig. 4: Semi-mobile crusher loaded by wheel loaders



3.1.4 Glory hole

Should the deposit be located in steep mountains or rough terrain, it is sometimes very difficult or even impossible to construct a haul road or a conveyor belt to transport the rock to the plant. In such cases, the glory hole concept is used, which consists of a vertical shaft and a tunnel.

The crusher can be installed as a semi-mobile crusher on top of the glory hole or as a fixed installation at the base of the hole. In the second case, a larger hole diameter is necessary. The glory hole concept requires a large investment costs for the construction of the shaft and the tunnel.

3.2 Haul road design

The design and condition of the haul road is very important for the efficiency of the truck haulage operation. In a quarry with a truck system, about 50% of the mining costs are haulage costs. Therefore, optimised truck haulage plays an important role. Assuming mining costs of 2 USD / t and a 5 % reduction in haulage costs with an improved road design, cost savings of 50.000 USD per year for a typical quarry with a yearly production of 1 Mio. T can be realised.

There are three main parameters in design of haul roads:

- grade
- width
- location

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3.2.1 Grade

Generally, the best road grade lies in the range of 8 - 12 % for regular rear-dump trucks. Where climatic conditions are severe (excessive snow or rain) the tendency is to reduce the grade. Articulated haul trucks can work in severe terrain conditions, and can cope with steeper grades (25%).

3.2.2 Width

The width of the haul road should be 3 - 3.5 times that of the truck width to allow safe two lane truck traffic. The space required for the installation of protection berms and drains must also be added to the total width of the haul road.

3.2.3 Location

Location of the haul road is perhaps the most difficult part. The final haul road should be established as soon as possible to avoid construction of temporary roads, and the line of the haul road should result in short haulage distances. A good practice is to align the haul road along a final slope, because the haul road design influences the long-term planning, both of which must be made in accordance with each other.

When deciding on the location of the haul road, it is also important to position the road relative to the usable / waste contact. The haul road width normally exceeds final berm width, and consequently if the haul road is positioned in such a way that no reserves are to be lost, then additional waste has to be removed. The alternative is to select a location on usable reserves so that no additional waste removal is required. The positioning of the road will be a compromise that balances additional costs for waste removal against losses on reserves.

Sharp and steep curves on the road should be avoided as they reduce the travelling speed and increase wear and tear on tyres and chassis. Curves should be designed with a super-elevation to counter balance the centrifugal force.

3.2.4 Safety

Safety is an important factor on haul road design. When planning curves and hills, emphasis has to be laid on good visibility along the haulage route. The trucks should be able to stop within the range of visibility (Fig. 11). On long, steep haul roads break failures have to be reckoned with, leading the down-hill traffic lane on the hill side of the road. Middle berms or emergency exits are preventative measures against fatal accidents.

3.2.5 Calculation of truck capacity

Once the profile of the haul road has been established, the average cycle time can be calculated and the required number of trucks for a given production rate can be determined.

3.3 Dewatering

In areas with significant rainfall, dewatering is important for a safe and efficient quarry operation. Therefore, a good quarry design has to include a dewatering concept. That means the benches need a slight inclination in such a way that the water runs off in a certain direction from where it can be drained off to the quarry sump.

The drains and the sump must be designed with sufficient capacity to handle maximum expected rainfall.

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4. VISUALISATION OF QUARRY DEVELOPMENT

A quarry is not a static, but a dynamic structure that greatly affects the appearance of the surrounding area. Excellent planning is required today to cope with this continual change.

In recent years, an increased sensitivity by the public to the field of environmental protection has resulted in approval procedures becoming increasingly difficult. A submission for granting an extension of a mining licence can be helped by accurate computer based 3-D quarry models and easy-to-understand illustrations. These provide detailed insight for all steps of the quarry development, from the opening to the final recultivation. Visual displays of the quarry make operational and environmental related planning fully transparent. In particular the effectiveness of any measures for environmental protection can be clearly demonstrated. This accords well with the environmental policies of many cement manufacturers, and also saves time in the face of proliferating regulations and ever-lengthier licencing procedures.

Fig. 5 **Aerial view of a quarry. Clearly visible are the position of the crusher (small quarry, top left), the access road and the quarry wall with a height of up to 250m.**

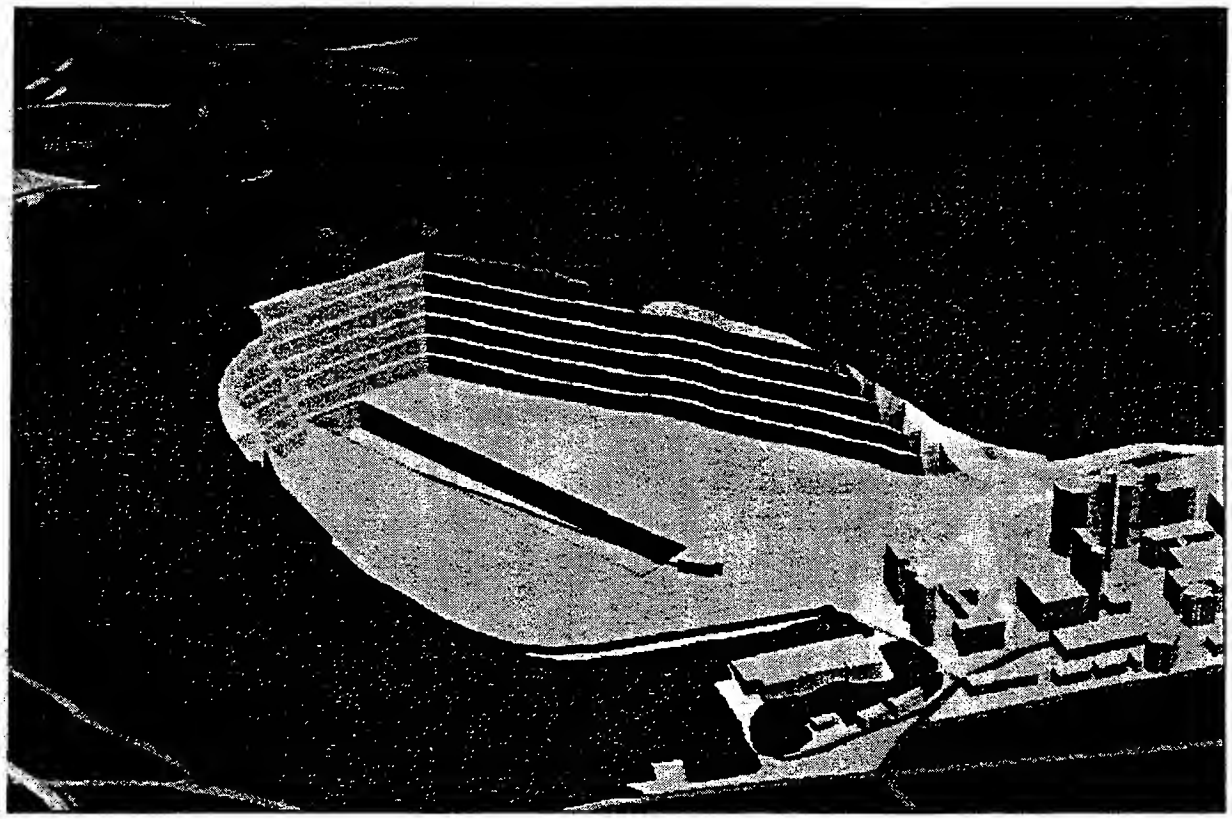

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Fig. 6 **Quarry at ground-level view. The main issues and the camouflage wall are visible.**

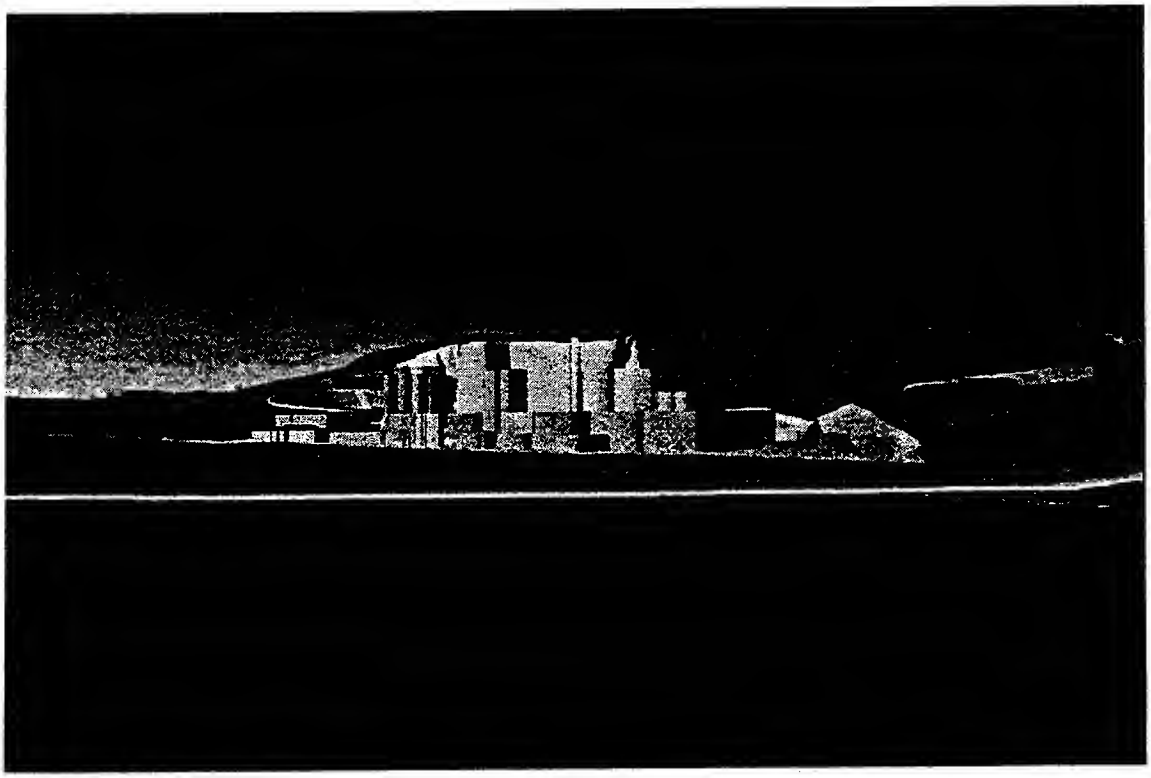

Fig. 7 **This view is achieved using a combination of computer simulation and the current appearance of a landscape using photomontage.**

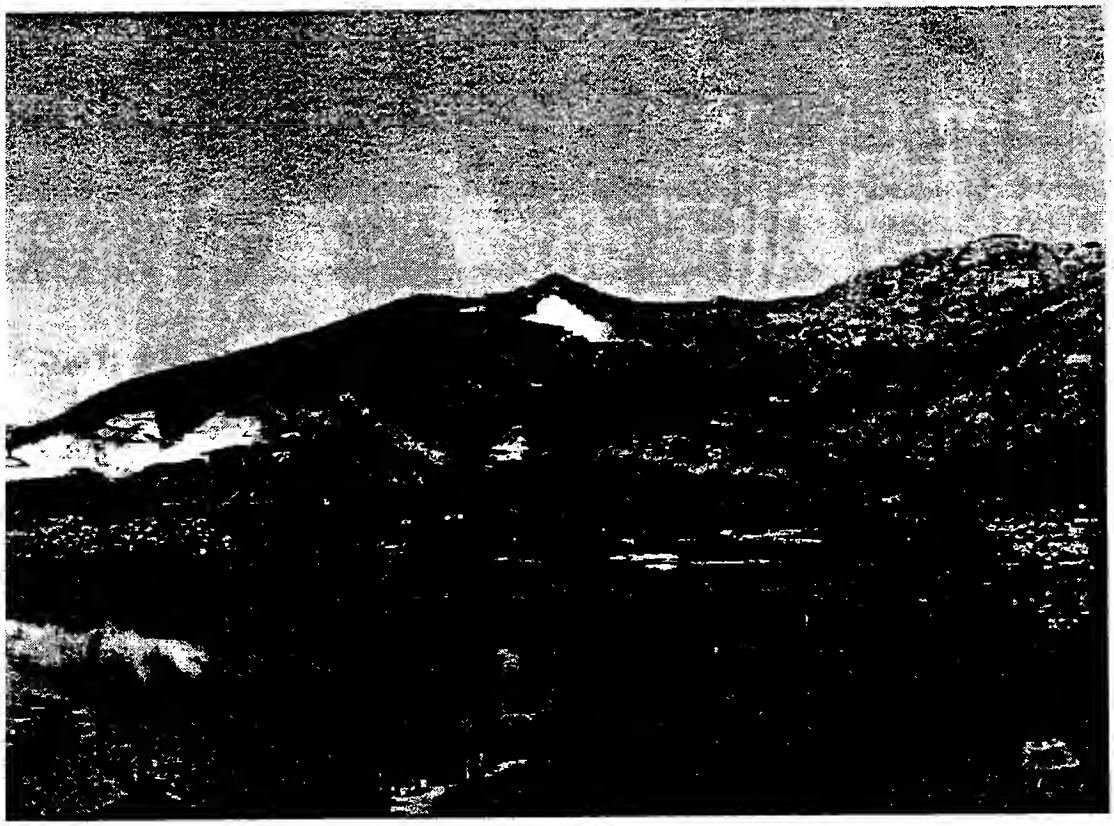

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Figure 5 Shows an aerial view of a quarry. This perspective is particularly suited for the clarification of excavation concepts, from an operational viewpoint. Clearly visible in the figure are the position of the crusher ('small quarry' top left), the access road and, in particular, the impressive quarry wall with a height of up to 250m.

Figure 6: Shows another quarry. Unlike the first example, this is a normal ground-level view. Here the main issues are visibility and the camouflage wall. These topics chiefly interest authorities and the general public.

Figure 7: Is a view achieved by a combination of computer simulation and the current appearance of the landscape using photomontage.

This is the same quarry as in Figure 5. In this project, the authorities and the general public are taking a close and critical look at the environmental impact, so the prospects for an excavation licence are being studied using 3-D modelling and photo-realistic visualisations.

It must be emphasised that all visual presentations are based on exact plans, which are accurate in every detail. These plans are based on CADE (Computer Aided Deposit Evaluation) and QSO (Quarry Scheduling Optimisation)'. They optimally satisfy long and short-term objectives. The objectives are steady supply of good-quality raw materials, with maximum deposit utilisation and lowest possible production cost.

QED as a further technique allows 3-D modelling and visualisation of the quarry development. With the method of QED, the planning becomes clear and easily understandable - clear and understandable for both the expert and the general public.

The final decision on further investments (for example for a detailed drilling campaign) will not be reached until later, after the prospects for an excavation licence are positive.

5. CONCLUSION

To bring raw materials management into operation the following steps have to carried out:

- The general layout of the quarry and a suitable mining procedure has to be determined.
- A long-term strategy for the quarry development has to be defined, which is guided by the optimum use of the deposit.
- More detailed periodical quarry plans have to be developed and transferred into detailed technical quarry maps.
- Photorealistic perspective views of the quarry development are produced to support the permitting process

Today, integrated computerised tools are used for digital terrain modelling, volume calculations and drawing of topographical maps. With the tendency to large scale operations and increasing environmental awareness, the necessity for reliable and detailed quarry planning becomes increasingly evident.

6. REFERENCES

BAUMGARTNER W., Computerised raw materials management - developments and progress. World Cement, Vol. 21, no. 5, pp. 207-213 (1990).

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Chapter 12

Basic Cement Chemistry

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BASIC CEMENT CHEMISTRY CHEMICAL COMPOSITION

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Elemental composition in weight percentage. By convention, the elements are expressed in form of their oxides (exception: Cl, F).

Table 1 Usual sequence of elements in cement analysis:

	Examples	
	Limestone	**Clinker**
L.o.I.[1]	42.2	0.24
SiO_2	1.9	22.7
Al_2O_3	0.81	5.7
Fe_2O_3	0.52	1.9
CaO	52.2	66.0
MgO	1.4	2.0
SO_3	0.56	0.33
K_2O	0.22	0.74
Na_2O	0.08	0.09
TiO_2	0.05	0.18
$Cr2O_3$		
$Mn2O_3$	0.02	0.03
P_2O_5	0.01	0.05
Cl	0.01	0.01
F		

[1)] loss on ignition, e.g. at 1050°C
mainly due to H_2O, CO_2

1. CHEMICAL FORMULAE

The chemical formula indicates the elements occurring in a chemical compound:

- for a molecular compound, type and absolute number of elements in a molecule are given

H_2O	O_2	C_6H_6
water	oxygen	benzene

- for a mineralogical compound, type and relative number of elements are given

SiO_2	CaO	$CaCO_3$	Ca_3SiO_5
quartz	lime	calcite	alite

Note: In mineralogical compounds, the elements need not necessarily occur in simple numerical ratios (impurities, solid solution)

- in the cement chemistry, shorthand's are often used:

CaO	SiO_2	Al_2O_3	Fe_2O_3	SO_3	H_2O
C	S	A	F	$\overline{S}$	H

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Examples:
C_3S for Ca_3SiO_5 (alite)
C_2S for Ca_2SiO_4 (belite)
C_3A for $Ca_3Al_2O_6$ (aluminate)
C_4AF for $Ca_4Al_2Fe_2O_{10}$ (ferrite)

2. MINERALOGICAL COMPOSITION

Table **Composition of a material, expressed in weight-percentage of the occurring minerals**
Example:

	Limestone	
Calcite	$CaCO_3$	90%
Dolomite	$CaMg(C0_3)_2$	5%
Quartz	SiO_2	5%
	Clinker	
Alite	C_3S	58%
Belite	C_2S	23%
Aluminate	C_3A	9%
Ferrite	C_4AF	7%
Periclase	MgO	1%
Arcanite	K_2SO_4	1%
Free lime	CaO	1%

Table **Difference between chemical and mineralogical composition:**

		Limestone		
Mineralogical	comp.		Chemical	comp.
Calcite	$CaCO_3$	90%	L.O.I.($C0_2$)	40,0%
Dolomite	$CaMg(C0_3)_2$	5%	SiO_2	5,0%
Quartz	SiO_2	5%	CaO	53,9%
			MgO	1,1%

(simplified, minor elements not included)

3. CHEMICAL PARAMETERS FOR CEMENT-SPECIFIC MATERIALS

3.1 Titration

Content of carbonates as determined by acid-base titration, expressed as $CaCO_3$
% Titration = 1.786 CaO + 2.48 MgO

Applied for:

- Limestone
- Marl
- Raw Meal

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3.2 Lime Saturation

$$LS = \frac{CaO \times 100}{2.80\,SiO_2 + 1.18\,Al_2O_3 + 0.65\,Fe_2O_3}$$

or

$$LSF = \frac{CaO}{2.8SiO_2 + 1.2Al_2O_3 + 0.65Fe_2O_3}$$

The LS is a measure to which extent the CaO-richest compounds C_3S, C_3A and C_4AF can be formed without the necessary presence of free lime. At LS > 100, free lime will unavoidably be present after burning.

Applied for:

- Raw meal
- Clinker
- Cement: neat OPC only
  $CaO = CaO_{total} - 0\,7\,SO_3$

Usual range in clinker: 85 - 100

Note: The influence of MgO can be accounted for

$$LS = \frac{(CaO + 0.75MgO) \times 100}{2.80\,SiO_2 + 1.18Al_2O_3 + 0.65Fe_2O_3}$$

max. 2 % MgO may be introduced in formula (not applied in cement specifications)

3.3 Silica Ratio

$$SR = \frac{SiO_2}{Al_2O_3 + Fe_2O_3}$$

Applied for

- Siliceous-argillaceous raw components
- Raw meal
- Clinker
- Cement

Usual range in clinker: 1.8 - 3.6

3.4 Alumina Ratio

$$AR = \frac{Al_2O_3}{Fe_2O_3}$$

Applied for

- Siliceous-argillaceous raw components
- Raw meal
- Clinker
- Cement

Usual range in clinker: 1 - 3

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3.5 Na_2O-equivalent

Total alkali content, expressed as Na_2O
Na_2O-equivalent = $Na_2O + 0.658\ K_2O$

Note: Limit for low alkali cement

Na_2O-equiv. $\leq$ 0.6 %

Applied for Clinker

Cement

4. CONTENT OF CLINKER MINERALS ACCORDING TO BOGUE

Percentage content of clinker minerals, assuming that chemical equilibrium is attaint, and that no impurities are present

$C_3S = 4.07CaO - 7.6SiO_2 - 6.73Al_2O_3 - 1.43Fe_2O_3$

$C_2S = 8.6SiO_2 + 5.07Al_2O_3 + 1.08Fe_2O_3 - 3.07CaO$
or $2.87SiO_2 - 0.754C_3S$

$C_3A = 2.65Al_2O_3 - 1.69Fe_2O_3$

$C_4AF = 3.04Fe_2O_3$

In reality, the mineralogical composition of industrial clinkers differs to some extent from that calculated according to Bogue.

4.1 Applied for

- Cement:
  - OPC only (excl. blended cements)
  - correction for CaO in $CaSO_4$:
    $CaO = Ca_{tot} - 0\ 70\ SO_3$
  - For ASTM: TiO_2 and P_2O_5 to be added to Al_2O_3
- Clinker:
  - CaO can be corrected for "$CaSO_4$" or for free lime, depending on objective of calculation

5. SIGNIFICANCE OF CLINKER MINERALS FOR CEMENT PROPERTIES

C_3S Contributes to early and late strength (1 d - ...)
Increases heat of hydration

C_2S Contributes to late strength (28 d - ...)

C_3A Contributes to early strength (1 - 3 d)
Increases heat of hydration
Impairs resistance to sulphate attack

C_4AF Little effect (Bogue)

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6. SIGNIFICANCE OF CLINKER MINERALS FOR ASTM CEMENT TYPES

Type I	Portland no restrictions regarding clinker minerals
Type II	Portland with moderate sulphate resistance C_3A max. 8 %
Type III	Portland with high early strength C_3A max. 15 %
Type IV	Portland with low heat of hydration C_3S max. 35 % C_2S min 40% C_3A max. 7 %
Type V	Portland with high sulphate resistance C_3A max. 5.0 % $C_4AF + 2\ C_3A$ max. 25 % or $C_4AF + C_2F$ max. 25 %

7. RELATIONSHIPS BETWEEN CHEMICAL MODULI AND CLINKER MINERALS

The following relationships are calculated for simplified clinker compositions, i.e. only containing the main elements SiO_2, Al_2O_3, Fe_2O_3, CaO

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Fig: **Clinker Minerals as Function of LS**
SR=2.5 AR=1.5

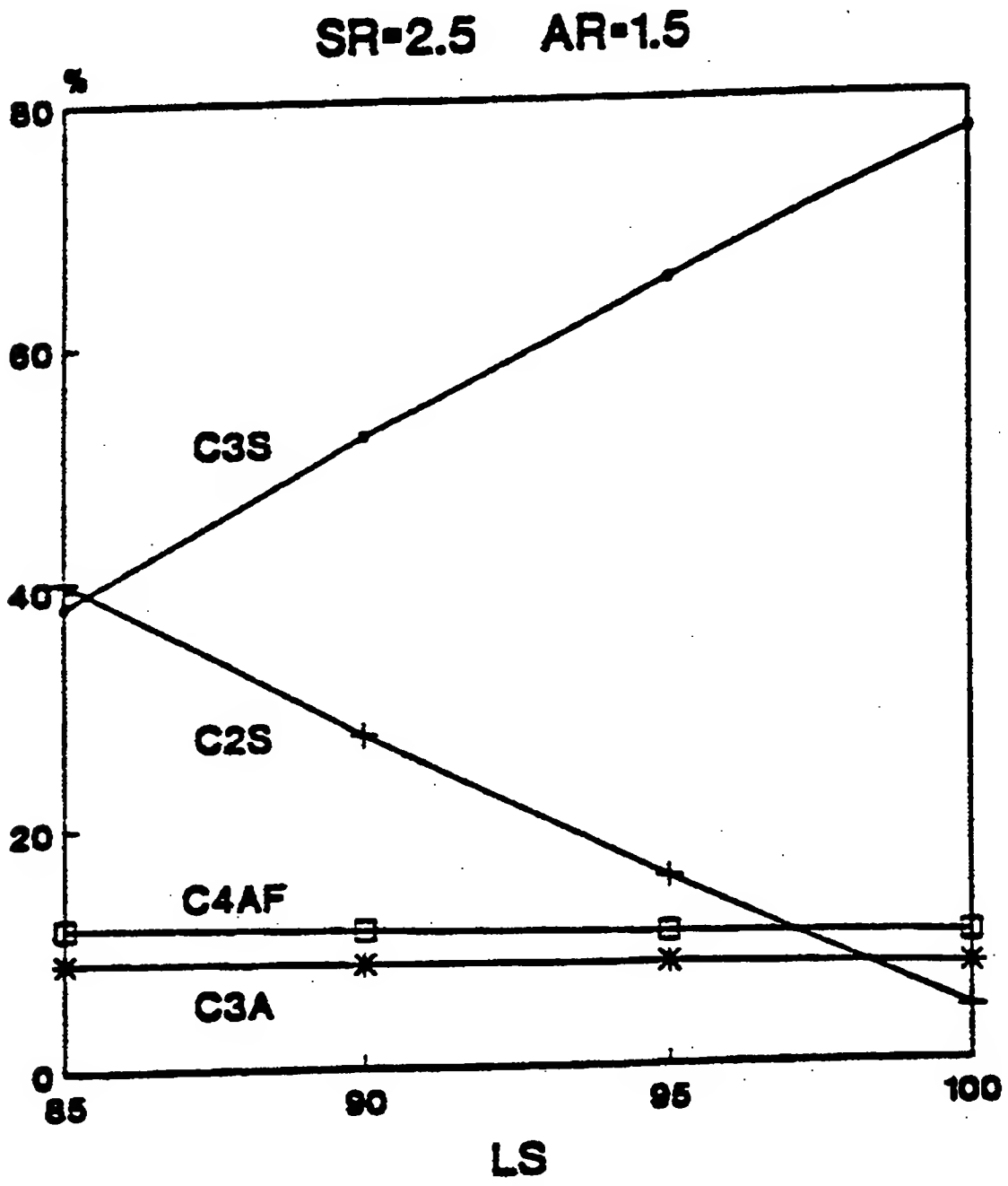

Fig: **Clinker Minerals as Function of SR**
LS=95 AR=1.5

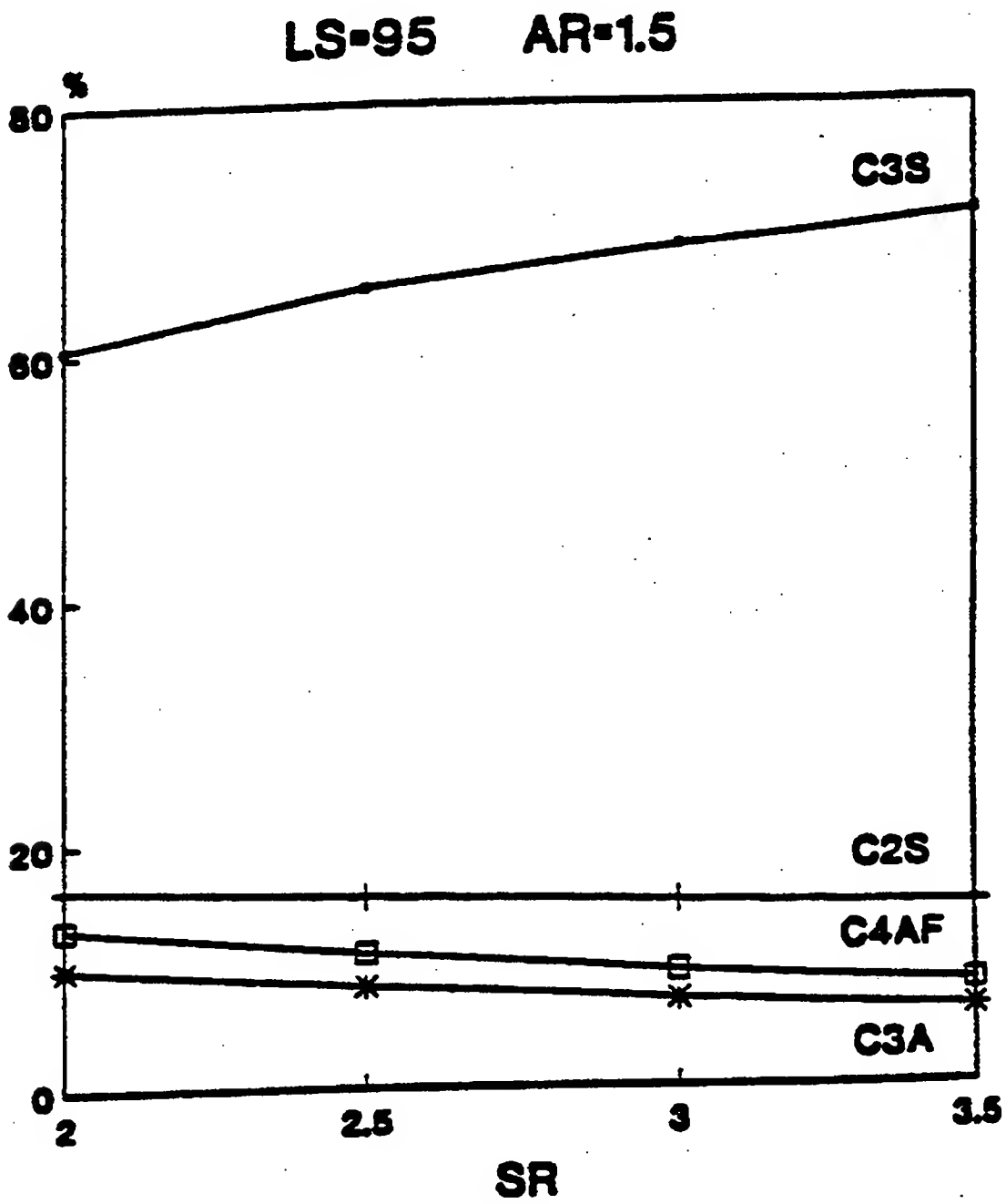

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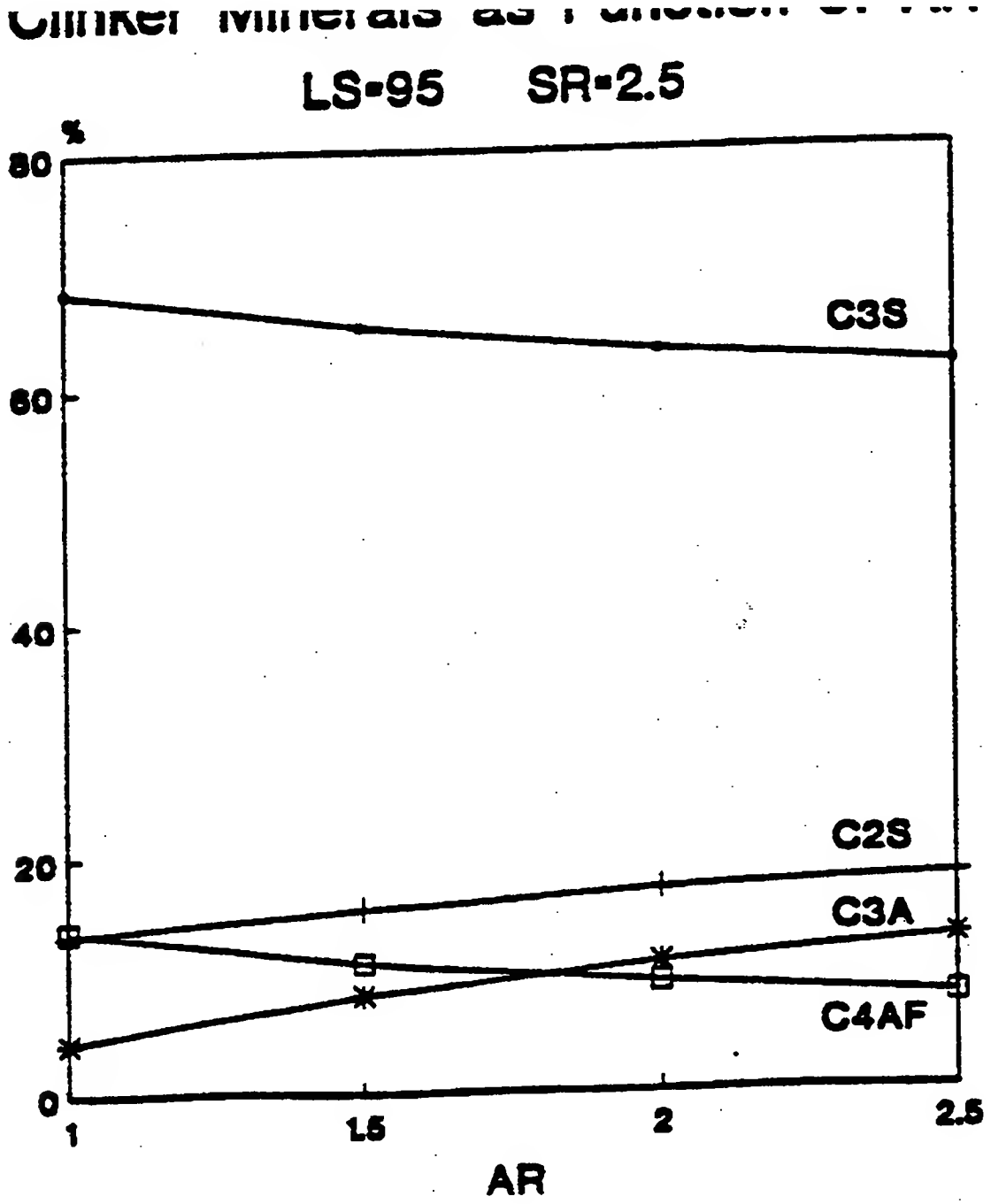
Fig: **Clinker Minerals as Function of AR**
LS=95 SR=2.5



8. EXERCISE FOR CALCULATION

	Raw Mix	Potential clinker composition
L.o.l.	35.1	.....
SiO_2	14.3	...22.....
Al_2O_3	3.6	...5.55...
Fe_2O_3	2.0	...3.08...
CaO	42.0	...64.7...
MgO	1.8	...2.77...
SO_3	0.25	...0.38...
K_2O	0.63	...0.97...
Na_2O	0.22	...0.34...
TiO_2	0.17	...0,26...
Mn_2O_3	0.10	...0.15...
P_2O_5	0.06	...0,09...
Cl	0.01	...0,01...
Titration	.....	
LS	...92,12...	...99,22...
SR	.....	.....
AR	.....	.....
Na_2O-equiv.		.....
C_3S		...54.4...
C_2S		.....
C_3A		.....
C_4AF		.....

CK factor = $\frac{100}{100 - \text{LoI}}$

CK = 1,54

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Loss on Ignition	
SiO_2	22.03
Al_2O_3	5.55
Fe_2O_3	3.08
CaO	64.71
MgO	2.77
SO_3	0.39
K_2O	0.96
Na_2O	0.34
TiO_2	0.26
Mn_2O_3	0.15
P_2O_5	0.09
Cl	0.02
F	
TOTAL	100.35
Freelime	
Insoluble Residue	

LS	92.13
SR	2.55
AR	1.80
C_3S	54.3
C_2S	22.2
C_3A	9.5
C_4AF	9.4
C_3S	54.3
C_2S	22.2
C_3A	9.5
C_4AF	9.4
Titration	-
Clinkerfactor	-
Na_2O-eq.	0.97
Mol. Alk./SO_3	3.16

cem-calc.xls

"Holderbank" Manage..ient and Consulting Ltd. Analytical Laboratory

Loss on ignition	35.1
SiO_2	14.3
Al_2O_3	3.6
Fe_2O_3	2.0
CaO	42.0
MgO	1.8
SO_3	0.25
K_2O	0.62
Na_2O	0.22
TiO_2	0.17
Mn_2O_3	0.10
P_2O_5	0.06
Cl	0.01
F	
TOTAL	100.23
Freelime	
Insoluble Residue	

LS	92.13
SR	2.55
AR	1.80
C_3S	54.3
C_2S	22.2
C_3A	9.5
C_4AF	9.4
C_3S	
C_2S	
C_3A	
C_4AF	
Titration	79.48
Clinkerfactor	1.54
Na_2O-eq.	0.97
Mol. Alk./SO_3	3.20

cem-calc.xls

Chapter 13

Mix Design

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MIX DESIGN

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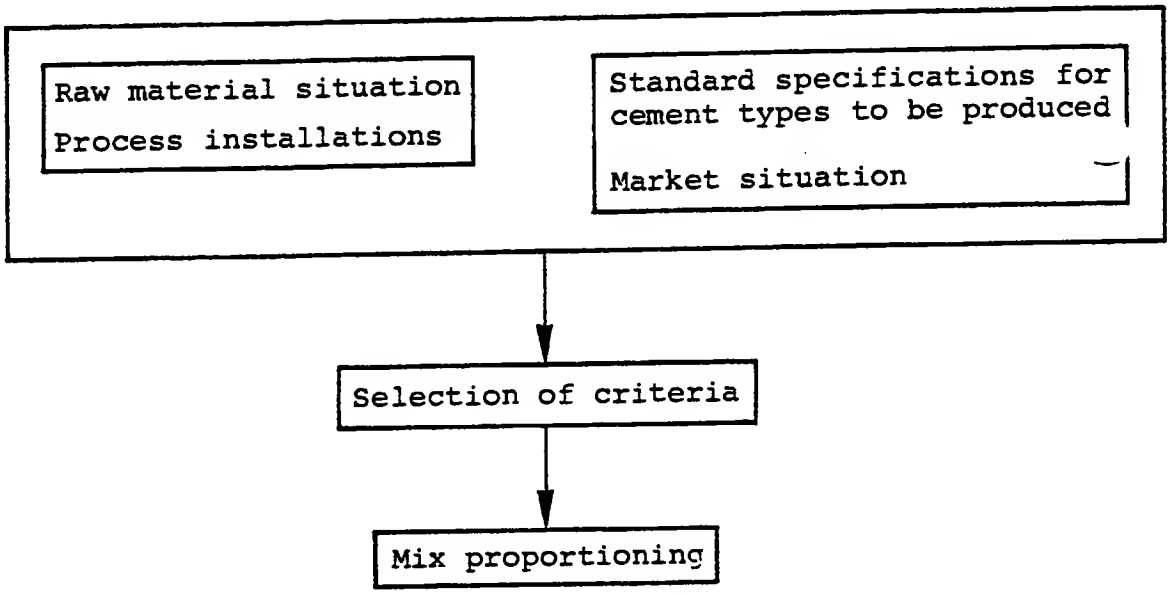
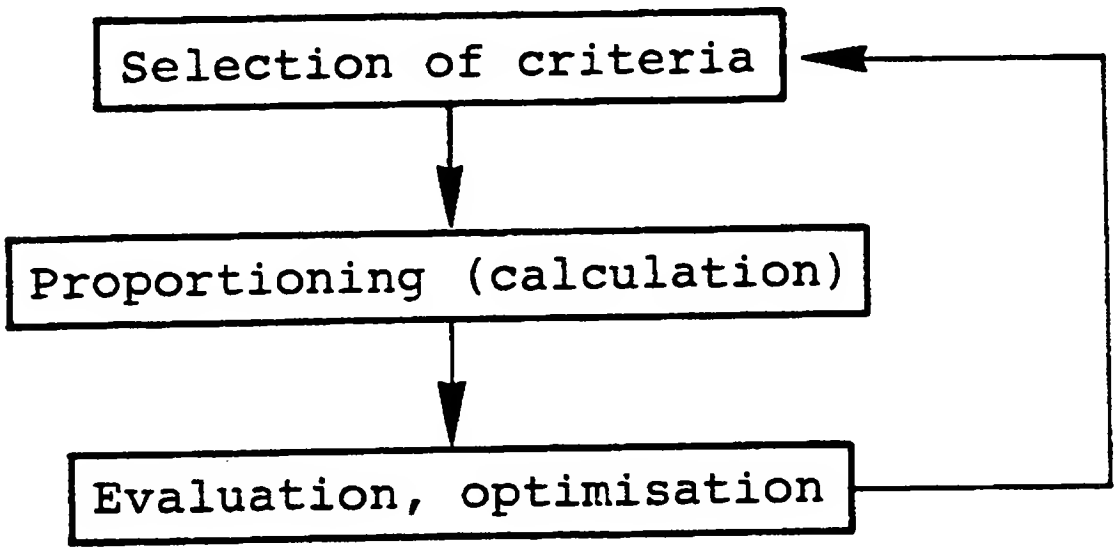
1. GENERAL

A raw mix design comprises not only raw mix proportioning but also considerations of such factors as standard specifications of the cement types to be produced, the market situation and the available process installations.



The selection of criteria is dictated by the standard specifications.

Designing raw mixes does not only involve the proportioning (calculation) but includes an evaluation of the obtained results. The latter involves optimisation with respect to costs and materials.



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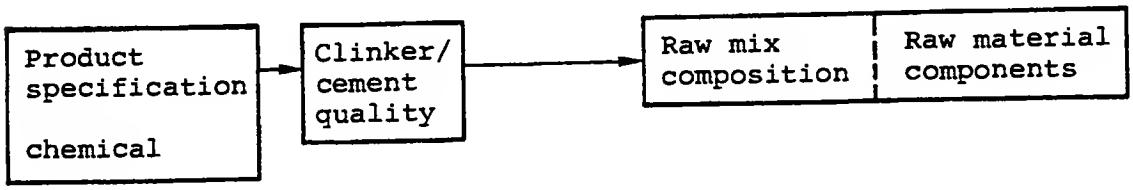
2. DEFINITION OF CRITERIA FOR MIX CALCULATION

Any type of cement has to conform to the individual cement standards of a particular country. Standards (standard specifications) normally include chemical specifications for clinker and cement. Together with the physical and strength requirements, they guarantee a suitable quality potential for the respective cement type.

With regard to the raw material aspects only the chemical requirements are significant:



In other words: the product specifications dictate the clinker/cement quality which in turn dictates the chemical composition of the raw mix and finally the selection of the raw material components.

The above sequence can also be reversed: an existing raw material configuration with little freedom as to the proportioning of the raw mix, may permit the manufacture of only one particular type of clinker.

Table 41 Influence of chemical requirements on raw materials

Chemical requirements	Influence on raw material
min. SO_3	Rejection of SO_3-bearing components (e.g.) gypsum-containing shale)
min. MgO	Rejection of MgO-bearing components (e.g. dolomitic limestone)
min. Alkali	Selection of raw material with low alkali-content
min. C_3A	Selection of components with very low alumina content and / or high iron content

Table 41 shows the influence of chemical requirements on the choice of raw materials.

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The following chemical criteria are normally used as a basis for raw mix proportioning (Table 42; on clinker basis):

Table 42 Chemical criteria for raw mix proportioning

criteria	"normal" range limit (for clinker)	formulas, remarks
MgO	max. 5% (6%)	for all cements
SO_3*	3 - 4,5%	depending on cement type
LIME STANDARD or LIME SATURATION FACTOR	0,9 - 1 or 90 - 100%	$\frac{CaO}{2,8\ SiO_2 + 1,2\ Al_2O_3 + 0,65\ Fe_2O_3}$
"Improved" Lime standard **	90 - 100%	$\frac{100\ (CaO + 0,75\ MgO)\ **}{2,80\ SiO_2 + 1,18\ Al_2O_3 + 0,65\ Fe_2O_3}$
Index of activity	2,5 - 3,5	$\frac{SiO_2}{Al_2O_3}$
Hydraulic ratio	2,0 - 2,4	$\frac{CaO}{SiO_2 + Al_2O_3 + Fe_2O_3}$
SILICA RATIO	1,8 - 3,4	$\frac{SiO_2}{Al_2O_3 + Fe_2O_3}$
ALUMINA RATIO	1,5 - 2,5 (0,7 - 3,5)	$\frac{Al_2O_3}{Fe_2O_3}$
Total alkali	< 0,6%	$Na_2O + 0,66\ K_2O$ for low alkali clinker
C_3S	50 - 60%	except for ASTM type IV
C_3A	max. 3% BS max. 5% ASTM	for sulfate-resisting cement

* for cement

** 100 (CaO + 1,5) for MgO < 2%

The proportioning of raw mixes for ordinary Portland cement is mostly based on the following specific criteria:

- MgO
- Lime standard or lime or saturation factor (or C_3S)
- Silica ratio
- Alumina ratio

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As Table 42 indicates, ratios are the preferred chemical criteria for proportioning since they offer the advantage of expressing the main and most important chemical parameters such as SiO_2, Al_2O_3, Fe_2O_3 and CaO in one single figure.

Other important criteria such as type and composition of fuels should not be overlooked. Coal ash as a combustion product of coal, for instance, has to be analysed quantitatively and qualitatively and should be treated as an individual raw material component. Fuel oil has to be considered as a potential carrier of sulphur, etc.

Additional criteria which could have bearing on the mix proportioning refer to performance characteristics, e.g.:

- minimum dust emission
- burnability and coating properties
- extreme components which affect machine performance

or to economic factors, e.g.:

- maximum overall economy
- easy and simple operations
- minimum number of components

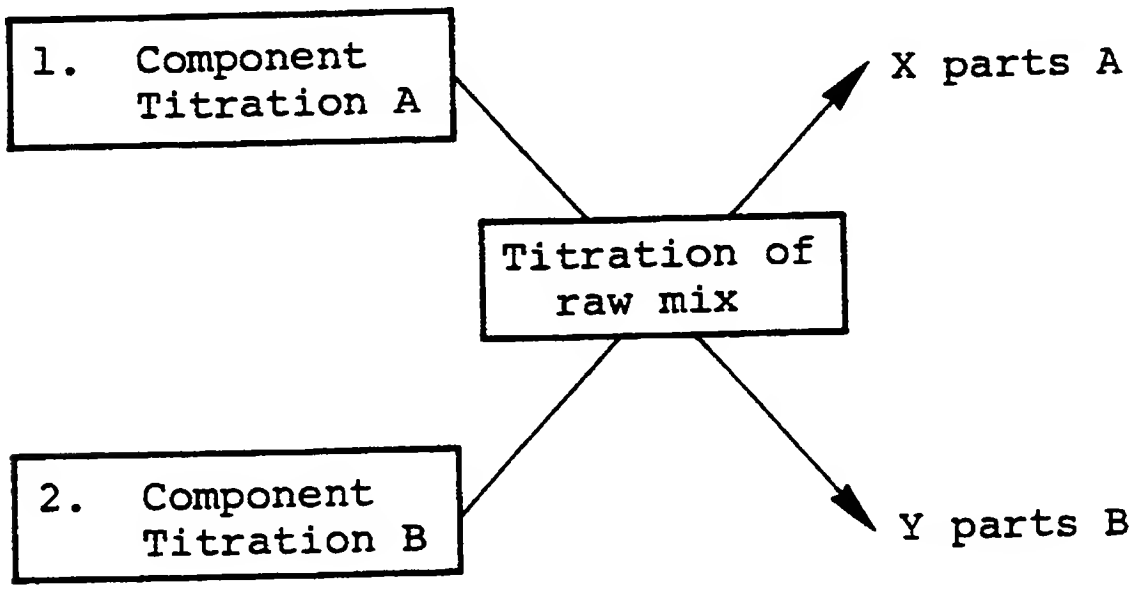
However, performance characteristics in particular can normally be controlled regarding the "normal" chemical requirements for cement raw mixes. The economic factors, on the other hand, are of the same significance as the chemical requirements.

3. PRINCIPLES AND METHODS OF MIX PROPORTIONING

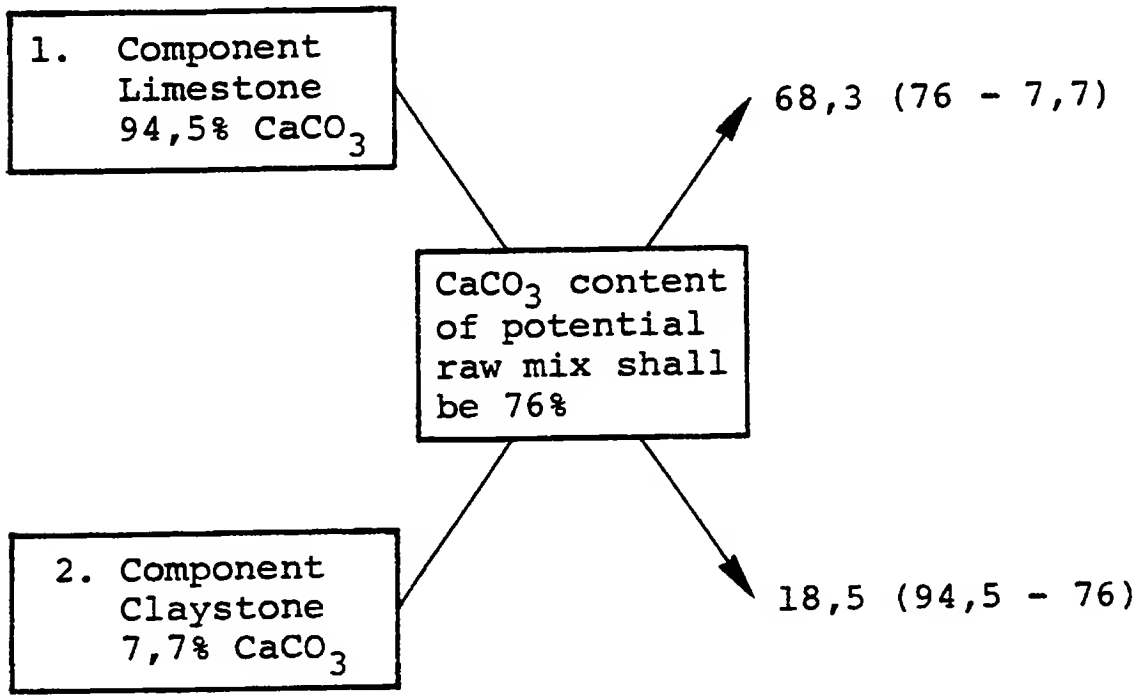
Proportioning (calculation) of potential cement raw mixes can be accomplished by various methods:

3.1 X-Pattern

The x-pattern represents a linear estimation of two raw material components by selecting the anticipated titration value (total carbonate content) of the potential raw mix as basis.



or as a numerical example.



The potential raw mix with a titration value of 76% would thus consist of:

$$\frac{\text{limestone}}{\text{claystone}} = \frac{68.3}{18.5} = \frac{3.69}{1}$$

or

limestone 78,6 % claystone 21,3 %

The resulting analysis of the raw mix has to be checked with regard to the requirements of the standard specifications.

3.2 Manual Calculation

There are a number of mathematical methods for two and three-component systems. Formulas are not complicated but comprise a large number of steps. The method of manual calculation as such is outdated.

3.3 Graphical Methods

These methods require preparatory work (manual calculations) for the determination of the relevant figures which are the basis for the construction of the diagrams and graphs. Graphical methods represent a rather archaic stage of mix proportioning.

3.4 Programmable Calculator

Programmable calculators normally produce one solution (out of possibly several). Obviously, this method is the best way to obtain a quick solution.

3.5 Computer Optimisation

It provides the optimum of a series of possible solutions considering the price factors as variables. If the available raw materials cannot meet the specified requirements for the raw

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mix, an approximate and an exact solution considering the missing constituents are produced (Tables 43 - 47).

Note: Mix calculations are normally based on dry raw materials. In practice, the natural moisture contents of the raw material components have to be considered too. This may entail alterations of the original mix proportions.

Table 43

19542222 CLAYSTONE

ANALYSEN

KOMPONENTEN	GVER	SIO2	AL2O3	FE2O3	CAO	MGO	K2O	NA2O	REST	CAOFREI	SO3	PREIS
CALLIDE COAL	0	46.17	37.81	11.34	1.80	1.55	.43	.35	.58	0	0	1.00E 06
LIME 7734	43.28	1.46	.45	.30	54.09	.49	.02	.02	.55	0	0	1.00E 06
GYPSUM 7733	13.92	48.38	17.58	12.20	4.10	.59	.14	.58	2.73	0	0	1.00E 08
SAND	.91	94.80	2.88	.2A	.35	.13	1.29	.20	0	0	0	1.00E 03
CINDERS	1.43	25.50	3.30	66.70	1.20	1.44	.24	.17	.08	0	0	1.00E 08

ANFORDERUNGEN

	OBERE GRENZE	UNT. GRENZE	RESULTAT
MGO	100.000		.812
NA2O+0.658K2O	100.000		.283
AL2O3	100.000		5.804
FE2O3	100.000		4.000
SIO2		0	21.794
KALKSTANDARD	1000.000	-0.000	92.353
TONERDE-MODUL	100.325	-0.025	1.170
SILIKAT-MODUL	2.675	1.075	2.025
C3S	95.500	54.900	55.500
C2S		-100.000	20.439
C3A	7.200		7.000
C4AF+2C3A	200.000		29.874
KOMPONENTE 1	.075	.024	

RESULTATE

MISCHUNGSVERHAELTNIS

CALLIDE COAL	.0245
LIME 7734	.7743
GYPSUM 7733	.1292
SAND	.0053
CINDERS	.0167
TOTAL	1.0000

KLINKER-ANALYSE

C3S	55.5000
C2S	20.6391
C3A	7.0000
C4AF	15.0744
MGO	.8121
ALKALI	.3449
CAOFREI	0
TONERDEMODUL	1.1705
SILIKATMODUL	2.0248
KALKSTANDARD	92.3526
SIO2	21.7947
AL2O3	5.8036
FE2O3	4.9596
CAO	65.6571
MGO	.8121
K2O	.1883
NA2O	.1596
MYDR. MODUL	2.8147
GIPS	0
PREIS	2.23881E 02

Table 44

2064 05050-044 MIXES ILIGAN NR 4 28. 6. 77 OPTIMIERUNG

A N A L Y S E N

NR	GVER	SIO2	AL203	FE203	CAO	MGO	K20	NA20	TIO	CRO	MNO	SO3	P20	CL	F	REST	PREIS
1	4290	190	69	39	5290	42	4	4	3	0	8	8	4	1	0	45	.10+00
2	1925	4150	1207	599	1576	239	99	109	52	0	8	9	11	9	0	0	.10+00
3	839	6380	1329	400	340	92	219	250	44	0	6	9	8	11	0	68	.22+00
4	293	1477	287	7297	247	58	40	31	55	0	2	141	0	0	0	65	.59+00

A N F O R D E R U N G E N R E S U L T A T E

	MAX.	MIN.	RRESULTAT	MISCHUNGSVERHAELTNIS		KLINKERANALYSE	
MGO	100.000	.000	1.145	LIMESTONE	.7535	C3S	59.5000
NA20+.658K20	100.000	.000	1.143	SHALE LOW	.1262	C2S	17.8651
AL203	100.000	.000	5.575	SHALE HIGH	.1145	C3A	9.6675
FE203	100.000	.000	3.022	PYRITE CIND	.0059	C4AF	9.1963
SIO2	100.000	.000	21.880		.0000	MGO	1.1453
KALKSTANDARD	1000.050	-.050	94.179		.0000	ALKALI	1.3610
TONERDEMODUL	1.845	1.795	1.845		.0000	HYDR. MODUL	2.1371
SILIKATMODUL	2.595	2.545	2.545		.0000	TONERDEMODUL	1.8450
C3S	60.500	59.500	59.500		.0000	SILIKATMODUL	2.5450
C2S	100.000	-100.000	17.865		.0000	KALKSTANDARD	94.1791
C3A	100.000	.000	9.669		.0000	SIO2	21.8795
C4AF+2C3A	200.000	-100.000	28.521	**********		AL203	5.5753
KOMPON. 1	1.000	.000	.753	T O T A L	1.0000	FE203	3.0218
						CAO	65.7423
						MGO	1.1453
OPTIMUM				PREIS	.11874+00	K20	.6385
						NA20	.7225
						TIO2	.2242
						CR203	.0000
						MN203	.1224
						SO3	.1441
						P205	.0864
						CL	.0509
						F	.0000
						REST	.6582

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Table 45

OPTIMIERUNG

7852 AE858,844 MIXES ILIGAN NO. 1

ANALYSEN

KOMPONENTEN	GVER	SIO2	AL2O3	FE2O3	CAO	MGO	K2O	NA2O	REST	CaOFREI	SO3	PREIS
LIMESTONE	42,90	1,90	,49	,40	52,90	,43	,04	,05	,69	0	0	1,40E 01
SHALE LOW	10,30	41,80	12,10	9,00	15,00	2,40	1,00	1,10	,70	0	0	1,40E 01
SHALE HIGH	8,40	63,40	12,20	4,00	3,40	,93	2,20	2,50	1,47	0	0	2,10E 01
PYRITE CINDI.	2,90	14,90	2,90	72,00	2,50	,59	,41	,32	1,02	0	0	5,41E 01
SILICA SAND	2,50	89,20	2,50	2,10	3,00	,28	,02	,00	,42	0	0	5,05E 01

ANFORDERUNGEN

	OBERE GRENZE	UNT. GRENZE	RESULTAT
MGO	100,000		1,253
NA2O+0,658K2O	100,000		,422
AL2O3	105,000		4,464
FE2O3	100,000		3,000
SIOE		0	21,021
KALKSTANDARD	1000,000	-5,000	88,437
TONERDE-MODUL	1,225	1,175	1,225
SILIKAT-MODUL	2,625	2,375	2,575
C3S	80,000	57,500	88,500
C2S		-100,000	10,000
C3A	100,000		5,427
C4AF+2C3A	200,000		23,423
KOMPONENTE 1	1,000	0	

RESULTATE

MISCHUNGSVERHAELTNIS

LIMESTONE	,7472
SHALE LOW	,1912
SHALE HIGH	0
PYRITE CINO	,0121
SILICA SAND	,0495
TOTAL	1,0000

KLINKER-ANALYSE

C3S	88,5000
C2S	10,0042
C3A	5,4272
C4AF	11,5742
MGO	1,2500
ALKALI	,7575
CAOFREI	0
TONERDEMODUL	1,2250
SILIKATMODUL	2,5750
KALKSTANDARD	94,4373
SIO2	21,0212
AL2O3	4,8050
FE2O3	3,0007
CAO	64,4103
MGO	1,2500
K2O	,3505
NA2O	,3070
MYDR. MODUL	2,1000
GIPS	0
PREIS	1,250E 01

Table 46

OPTIMALISIERUNG

7057 85050,844 MIXES ILIGAM NO. 7

ANALYSEN

KOMPONENTEN	ÜVER	SIO2	AL2O3	FE2O3	CAO	MGO	K2O	NA2O	REST	CAOFREI	SO3	PREIS
LIMESTONE	42,90	1,90	,69	,40	52,90	,43	,04	,05	,69	0	0	1,00E 01
SHALE LOW	19,33	41,40	12,10	8,00	15,00	2,40	1,00	1,10	,70	0	0	1,00E 01
SHALE HIGH	8,40	61,80	13,30	4,00	3,40	,93	2,20	2,50	1,47	0	0	1,10E 01
PYRITE CIND	2,96	14,90	2,90	73,40	2,50	,59	,41	,32	1,82	0	0	5,61E 01
SILICA SAND	3,50	89,30	2,50	2,10	1,00	,20	,02	,00	,42	0	0	5,05E 01

ANFORDERUNGEN

	OBERE GRENZE	UNT. GRENZE	RESULTAT
MGO	100.000		1.480
NA2O+0,658K2O	100.000		,821
AL2O3	100.000		5.874
FE2O3	100.000		3.218
SIO2		0	21.593
KALKSTANDARD	1000.050	-0.050	94.654
TONERDE-MODUL	1.825	1.775	1.025
SILIKAT-MODUL	2.425	2.375	2.375
C3S	60.500	59.500	59.500
C2S		-300.000	17.845
C3A	100.000		10.120
C4AF+2C3A	200.000		30.834
KOMPONENTE 1	1.000	0	

RESULTATE

MISCHUNGSVERHAELTNIS	
LIMESTONE	,7163
SHALE LOW	,2605
SHALE HIGH	0
PYRITE CIND	,0020
SILICA SAND	,0152
TOTAL	1,0000

KLINKER-ANALYSE	
C3S	59,5000
C2S	17,8454
C3A	10,1259
C4AF	9,7839
MGO	1,4877
ALK4LI	,9749
CAOFREI	0
TONERDEMODUL	1.8258
SILIKATMODUL	2,3758
KALKSTANDARD	94,4548
SIO2	21,5934
AL2O3	5,8736
FE2O3	3,2184
CAO	65,7490
MGO	1,4877
K2O	,4425
NA2O	,5144
HYDR. MODUL	2,1434
GIPS	0

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Table 47

2066 05050-044 MIXES ILIGAN NR 11 28. 6. 77 OPTIMIERUNG

ANALYSEN

NR	GVER	SIO2	AL203	FE203	CAO	MGO	K20	NA20	TIO	CRO	MNO	SO3	P20	CL	F	REST	PREIS
1	4290	190	68	39	5290	42	4	4	3	0	8	8	4	1	0	45	.10+00
2	1925	4150	1207	598	1576	239	99	109	52	0	8	9	11	9	0	0	.10+00
3	839	6380	1329	400	340	92	219	250	44	0	6	9	8	11	0	68	.22+00
4	293	1477	287	7297	247	53	40	31	55	0	2	141	0	0	0	65	.58+00

ANFORDERUNGEN	MAX.	MIN.	RRESULTAT	MISCHUNGSVERHAELTNIS		KLINKERANALYSE	
MGO	100.000	.000	1.362	LIMESTONE	.7290	C3S	59.5000
NA20+.659K20	100.000	.000	1.010	SHALE LOW	.2116	C2S	16.6421
AL203	100.000	.000	5.933	SHALE HIGH	.0555	C3A	10.2874
FE203	100.000	.000	3.216	PYRITE CIND	.0039	C4AF	9.7754
SIO2	100.000	.000	21.453		.0000	MGO	1.3622
KALKSTANDARD	1000.050	-.050	94.860		.0000	ALKALI	1.2040
TONERDEMODUL	1.845	1.795	1.845		.0000	HYDR. MODUL	2.1438
SILIKATMODUL	2.395	2.345	2.345		.0000	TONERDEMODUL	1.8450
C3S	60.500	59.500	59.500		.0000	SILIKATMODUL	2.3450
C2S	100.000	-100.000	16.642		.0000	KALKSTANDARD	94.8600
C3A	100.000	.000	10.287		.0000	SIO2	21.4528
C4AF+2C3A	200.000	-100.000	30.350	**********		AL203	5.9227
KOMPON. 1	1.000	.000	.729	TOTAL	1.0000	FE203	3.2156
						CAO	65.6038
						MGO	1.3622
OPTIMUM				PREIS	.11111+00	K20	.5672
						NA20	.6368
						TIO2	.2507
						CR203	.0000
						MN203	.1259
						SO3	.1410
						P205	.0927
						CL	.0538
						F	.0000
						REST	.57699

4. PRINCIPLES OF RAW MIX ASSESSMENT

Basically, evaluation and assessment of raw material components (4.4) and raw mixes refer to the same principles. The only difference exists in the immediate comparison of the chemical composition of a raw mix with the standard specifications of the products for which it is intended.

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4.1 Mix Type

The possible combinations of different rocks used in raw mixes can be classified as mix types. Important varieties are:

- Argillaceous limestone (marl) having the composition of a natural cement. An optimum homogenisation is realised in the rack texture itself. The reactions can easily take place even with a coarsely grained raw mix.
- The same rock in a metamorphic condition contains well crystallised silicates instead of clay minerals. Under otherwise similar conditions, the reactivity is lower than in the first case and there is a high probability that dust formation will occur in the preparation and burning process.
- Contrary to the above cases is the combination of pure limestone with pure clay. To get a close contact between lime and silicate, both components have to be ground finely and homogenised intensively. Depending on the type of clay minerals, the mixes can be more or less reactive,
- A further mix type is the combination of relatively pure limestone, argillaceous limestone and sandstone. Quartz introduced by the sandstone will decrease the grindability and the burnability to some extent. Problems may occur when less reactive minerals are present in the other two components.

Rock combinations actually used can easily be related to this series of mix types. The situation becomes more complicated when additions like pyrite ash, iron ore or bauxite are used.

4.2 Comparison of Raw Mix with Standard Specifications

Any raw mix composition has to be compared with the locally applied standard specifications in order to evaluate potential conformity. As an example, Table 48 compares two analyses of typical Portland cements with the ASTM-specifications for the five main types of Portland cement, whereby these types are designated as follows:

Type I	Ordinary Portland cement
Type II	Moderate sulphate resistance or moderate heat of hydration
Type III	High early strength
Type IV	Low heat of hydration
Type V	High sulphate resistance

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Table 48 Raw mix composition and specification.

	clinker composition		chemical requirements according to ASTM specifications for type:				
	I	II	I	II	III	IV	V
Loss on ignition	0.43	0.69	<3.0	<3.0	<3.0	<2.5	<3.0
SiO_2	20.8	22.8		>21.0			
Al_2O_3	6.0	3.8		<6.0			
Fe_2O_3	2.5	4.4		<6.0			
CaO	66.7	65.2					
MgO	1.4	2.2	<6.0	<6.0	<6.0	<6.0	<6.0
So_3 *	0.52	0.16	<3.0 <3.5	<3.0	<3.5 <4.0	<2.3	<2.3
K_2O	0.80	0.39					
NA_2O	0.20	0.30					
Mn_2O_3	0.50	0.05					
P_2O_5	0.16	0.07					
TiO_2	0.27	0.26					
Cl	0.01	0.01					
Total	99.84	100.33					
Silica ratio	2.4	2.9					
Alumina ratio	2.4	0.9					
Lime saturation	99.6	93.4					
C_3S	59.9	65.1			<35		
C_2S	14.4	16.2			>40		
C_3A	11.7	2.8		<8	<15	<7	<5
C_4AF	7.6	12.7					<20 **

* depending on C_3A content

** $C_4AF + 2\ C_3A$

It is obvious in Table 48 that mix I conforms to the specifications for type I (ordinary Portland cement) and type III (high early strength), but not for the other types.

Mix II conforms to all cement types except type IV (low heat of hydration).

If a composition of a potential raw mix does not meet the specifications for a particular type of cement, the following measures have to be weighed:

- Modification of proportioning criteria (lime saturation factor, silica ratio, C_3A- or Al_2O_3 content, etc.)
- Selection of necessary corrective materials (silica sand, etc.)
- Replacement of components (replacement of an alumina rich claystone by a silica-rich material for production of ASTM type IV and V cements, etc.)
- Replacement of the selected fuel type or fuel quality (coal with little ash instead of coal with a high ash content, if the coal ash composition becomes a critically influencing parameter, etc.)

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Influence of minor ("deleterious") elements

The main influencing effect of the so-called deleterious constituents or elements on preparation and production is discussed in chapter 4.4.2. The following deals only with limits and effects of these constituents in the cement raw mix. Under normal circumstances, the following ranges and limits are to be expected:

Table 49 Deleterious constituents in cement raw mixes

deleterious constituents	"normal" range % (clinker basis)	limits % (clinker basis)	remarks
Alkalis:			
K_2O	0,5 - 0,8	0,6	for low-alkali
Na_2O	0,2 - 0,4	as Na_2O	clinker
MgO	1 - 3	5 - 6	according to local specifications
SO_3	0,2 - 1,0	1 - 1,5	higher SO_3 in clinker reduces quantity of gypsum to be added
P_2O_5	0,0 .. - 0,3	0,5 - 0,8	
Cl	0,01 - 0,03 (0,01 - 0,1)		depending on and determining the process
F	0,01 - 0,1		air pollution
Cr_2O_3	0,01 - 0,04		dermatitis
Fe_2O_3	3 - 5	0,3	for white cement production

These limits should not be regarded as isolated figures but rather as part of a multi-component system (including contributions from the fuel). Particular attention should be given to the systems of:

K_2O ----- Na_2O ----- SO_3

K_2O ----- Na_2O ----- Cl

whereby an effort should be made to achieve equalised alkali sulphur balance in order to prevent problems in the kiln system.

Only a few deleterious constituents are limited by specifications, e.g. the MgO and the total alkali-content (for low-alkali clinker). The others are not specified (limited) but practical experience with processing and quality requirements of the product (clinker/cement) dictate their quantitative limits.

4.3 Assessment of the Mineralogical Composition of Cement Raw Mixes.

A routinely performed assessment of a raw mix includes as a very important part the examination of the mineralogical composition (Table 50).

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Table 50 Mineralogical assessment of raw mix

Minerals	Effects on technology
Aragonite ($CaCO_3$)	dry grinding → coating in the mill and high power consumption
Quartz (SiO_2)	grinding → abrasion, wear and high power consumption burning → impairs burnability
Feldspar	burning → impairs burnability, low reactivity
Clay minerals: Montmorillonite Illite Kaolinite Chlorite Mica Palygorskite	preparation → water absorption, stickiness burning → improved burnability dust production → reduced dust prod. coating properties → facilitates coating
Minerals of good crystallinity	reactivity low, require more energy for transformation
Minerals of low crystallinity	reactivity high, less energy necessary for transformation

4.4 Assessment of Raw Mixes with regard to Cement Production and Choice of Process

As discussed previously, the properties of the raw materials, i.e. raw mixes, largely influence the choice of process in general, and the various stages of production. Tables 51 and 52 indicate the most significant relations and functions.

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Table 51 Significance of raw mix properties in cement production. (Compare also Table 51 p. 5/3 referring to raw material properties).

Aspects of production	Raw mix properties
Quarrying Crushing Transport Storage Grinding	see Table 40
Slurry preparation	clay mineral content, fineness
Drying	clay mineral content, porosity
Homogenising	chemical and mineralogical variability
Nodulising	clay mineral content
Dewatering	clay mineral, slurry characteristics(filtration)
Burnability	mineralogical composition, fineness, degree of weathering, intergrowth and size of rock fragments
Dust formation	mineralogical composition crystallinity
Coating formation	chemical composition

It becomes obvious that the clay mineral content is of paramount importance form many aspects of production.

Table 52 Summarises the most important raw mix properties influencing the choice of process.

Raw mix properties	Related features	WET PROCESS	DRY PROCESS
moisture content	clay mineral content, porosity	high	low
plasticity, stickiness	clay mineral content	high	low
homogeneity	chemical, physical and mineralogical variability	poor	high
chemical characteristics	chemical composition regarding alkalis,. sulphur, chloride, etc. (contents)	high	low

Table 52 only summarises raw mix aspects. However, other factors, e.g.

- seasonal fluctuations of moisture content
- transport, haulage etc.

are, of course, also determining factors in the choice of process.

4.5 Evaluation of Laboratory Test Results

The steps which are regarded as the final part of a mix design, are preparation, examination and evaluation of test results produced in a laboratory.

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4.5.1 Preparation

The proper preparation of laboratory raw mixes for testing is the prerequisite for reliable test results and subsequent evaluation.

It is as important as sampling and it should, therefore, be emphasised that both these processes have to be carried out under observation of strictly defined rules and controls.

4.5.2 Significance of Laboratory Investigations

The characteristics and behaviour of a cement raw material or mix during the various stages of production can never be predicted on the basis of the test results and findings of laboratory investigations alone. Laboratory testing has the disadvantage that many influencing and technologically important parameters such as kiln atmosphere, industrial preparation, etc., can be neither simulated nor reproduced on a laboratory scale. Laboratory produced test results, however, permit the recognition and interpretation of tendencies, whereby a broad variety of individual findings assures a more reliable final evaluation. It is thus recommendable to conduct a series of tests, the results of which can be used to support and control each individual finding. For instance, when the filtration properties of a cement slurry have to be assessed, mineralogical/chemical investigations grain size distribution tests, rheological tests on slurry and specific filtration tests should be conducted rather than a specific filtration test only. The same idea is applicable for all the other assessments of technological properties such as burnability characteristics, grindability properties, etc.

In order to guarantee that the laboratory results correspond as closely as possible to the findings of industrial practice, the design of the laboratory testing methods and other aspects such as limits, reproducibility, etc. should periodically be checked and compared.

4.5.3 Summary of Laboratory Tests

The following tests are available and normally applied in the cement industry (Table 53).

Table 53 Laboratory tests

Material aspects	Test designation	Limits, reproducibility, practice relevance
Stickiness	soil tests	
Burnability	burnability test	tendencies only, but good practice relevance
Grindability	grindability test	quantitative estimate of kWh/t requirements
Volatility of circulating elements	volatility test	quantitative estimate of primary volatility in various atmospheres
Coating behaviour brick selection	coating test	tendencies only, acceptable practice relevance
Filterability	filtration test, testing of slurry rheology	quantitative estimate of key-factors quantitative assessment of rheology
Nodulisability	granulation test, thermo-shock test strength test	tendencies only, practice relevance acceptable

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Normally these technological tests are supported by:

- chemical analysis (highly accurate)
- mineralogical analysis (semi-quantitative)
- grain-size analysis (accurate)

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Program for Optimal Raw Mix Design

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1.1 OPTIMAL RAW MIX DESIGN ON PERSONAL COMPUTER 284
2. PROGRAM FOR OPTIMAL RAW MIX DESIGN 284
2.1 FACILITIES 284

1. MIXOPT

1.1 OPTIMAL RAW MIX DESIGN ON PERSONAL COMPUTER

MIXOPT is a comprehensive program for the optimal design of raw mixes and determination of product characteristics. It is tailored to the specific needs of the cement industry and offers a wide range of applications in the field of raw meal preparation and product design:

- Up to 9 components may be included to determine a raw mix of desired characteristics.
- Fixed values or ranges may be set for chemical (e.g. Na_2O-equivalent) and mineralogical (e.g. C_3A) constraints of product characteristics.
- The component quantities can also be restricted. This might be necessary in case of limited availability or for fuel ash.
- Lowest cost product is determined within given product constraints if costs of components are entered
- Alternatively to cost optimisation, optimisation or minimisation of a particular chemical component is possible.
- Blending of materials or determination of raw meal, clinker and cement characteristics can be accomplished apart from raw mix optimisations.
- Program includes data base and print facilities.

Typical MIXOPT applications are illustrated on the following pages.

The knowledge required to operate the programs can quickly be acquired. Dialogue, window menu selection, and user's help facilities make it very user-friendly. Previous computer knowledge is not required.

The program or further information is provided by:

"Holderbank" Management and Consulting Ltd.,
Technical Centre Materials Division,
5113 Holderbank,
Switzerland

or

"Holderbank" Consulting Ltd.
2310 Lakeshore Road W.
Mississauga,
Ontario
Canada LSJ 1K2

2. PROGRAM FOR OPTIMAL RAW MIX DESIGN

2.1 FACILITIES

- BLENDING OF COMPONENTS
- DETERMINATION OF PRODUCT CHARACTERISTICS
  - Raw Max
  - Clinker
  - Cement

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- RAW MIX DESIGN WITH UP TO 9 COMPONENTS
  - Fixed Value or Range for Product Constraints of:
  - Limitation of Component Concentrations
  - Input Specification such as:
  - Reference to Moist or Dry Components
- COST OPTIMISATION OF PRODUCT
- DATA BASE FACILITIES FOR:
  - Components
  - Product Constraints (e.g. for different clinker types)
  - Calculated Product
- PRINT FACILITIES
  - Single Mix Design
  - Compilation of Mixes
- GENERAL USER FACILITIES
  - Insertion and Copying Mode
  - Title Specification
  - File Clearing
  - Help

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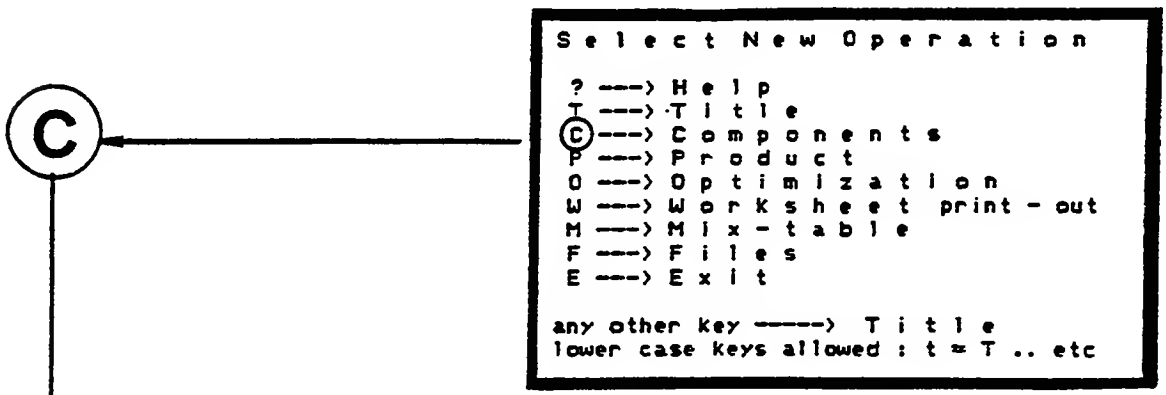
Fig. 1 Blending of Components.



COMPONENTS

Material	limest	limest	limest	limest	limest	limest	limest		TOTAL
Mat.- No	gt1	gt2	gt3	gt4	gt5	gt6	gt7		
i/a/c No	i 01	i 02	i 03	i 04	i 05	i 06	i 07		i 10
Humidity	0.00	0.00	0.00	0.00	0.00	0.00	0.00		0.00
L.O.I.	36.61	35.10	36.61	36.80	36.45	36.61	36.61		36.40
SiO2	11.51	13.50	11.51	11.51	11.00	12.00	12.60		11.95
Al2O3	2.51	2.51	2.61	2.51	2.65	2.66	2.51		2.57
Fe2O3	1.44	1.44	1.38	1.44	1.53	1.49	1.44		1.45
CaO	39.40	38.40	39.00	40.00	39.40	38.76	39.40		39.19
MgO	4.53	4.85	4.53	4.53	4.53	4.53	4.20		4.53
SO3	0.65	0.65	0.65	0.65	0.65	0.65	0.55	⟹	0.64
K2O	1.44	1.44	1.44	1.44	1.44	1.44	1.43		1.44
Na2O	0.18	0.18	0.18	0.18	0.18	0.03	0.18		0.16
TiO2	0.17	0.17	0.17	0.17	0.17	0.17	0.17		0.17
Mn2O3	0.14	0.14	0.14	0.14	0.14	0.14	0.14		0.14
P2O5	0.43	0.43	0.43	0.43	0.43	0.43	0.43		0.43
Cl-	0.02	0.02	0.02	0.02	0.02	0.02	0.02		0.02
.....	0.01	0.01	0.01	0.01	0.01	0.01	0.01		0.01
Rest	0.97	1.17	1.33	0.19	1.41	1.07	0.32		0.92
Total	100.01	100.01	100.01	100.01	100.01	100.01	100.01		100.01
Cost,wet	1.00	1.00	1.00	1.00	1.00	1.00	1.00		1.00
Fr.Lime									
Insolub.									
Proport.	1.00	1.00	1.00	1.00	1.00	1.00	1.00		7.00

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Fig. 2 Product Characteristics.

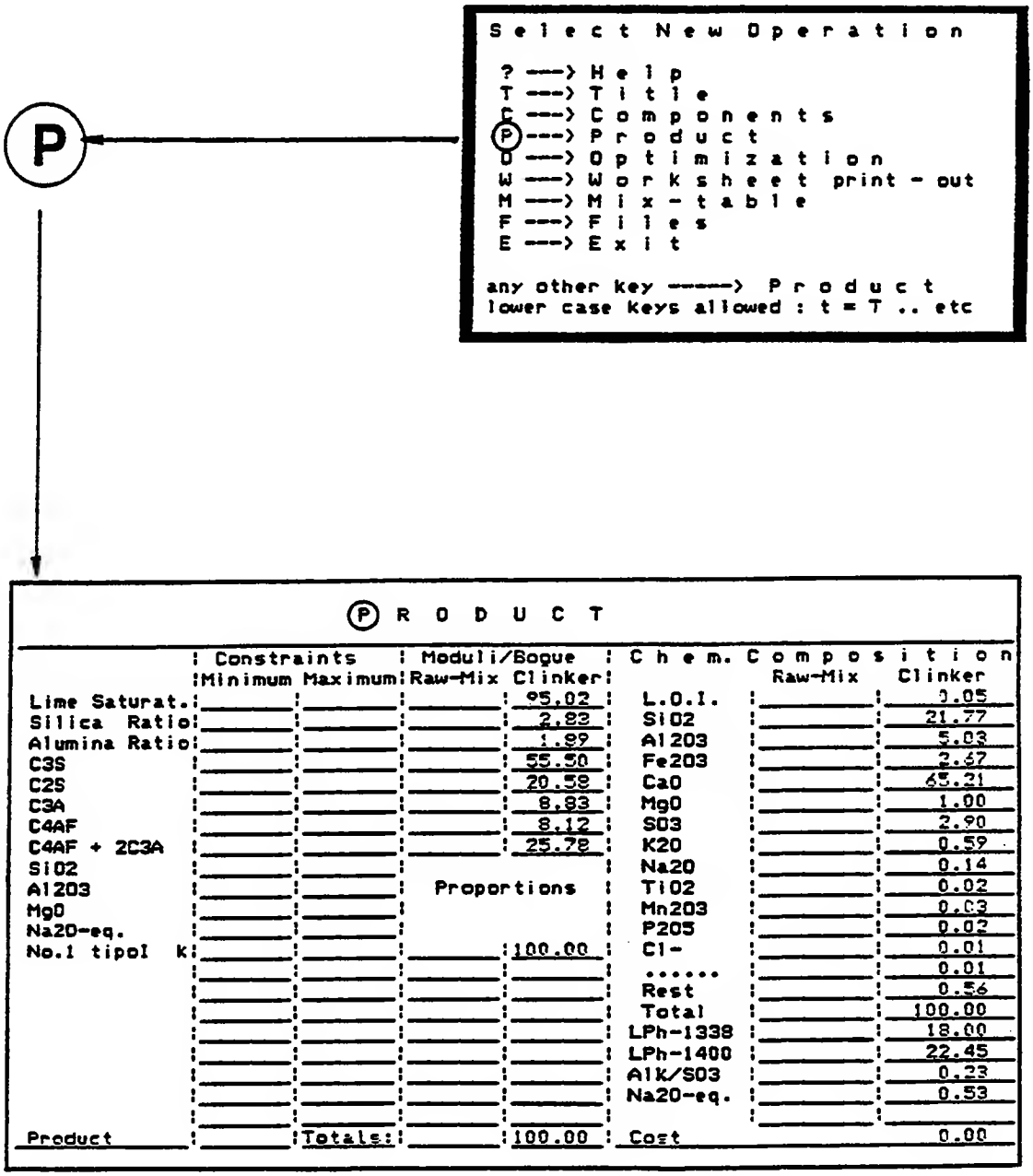

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Fig. 3 Mix Optimisation.

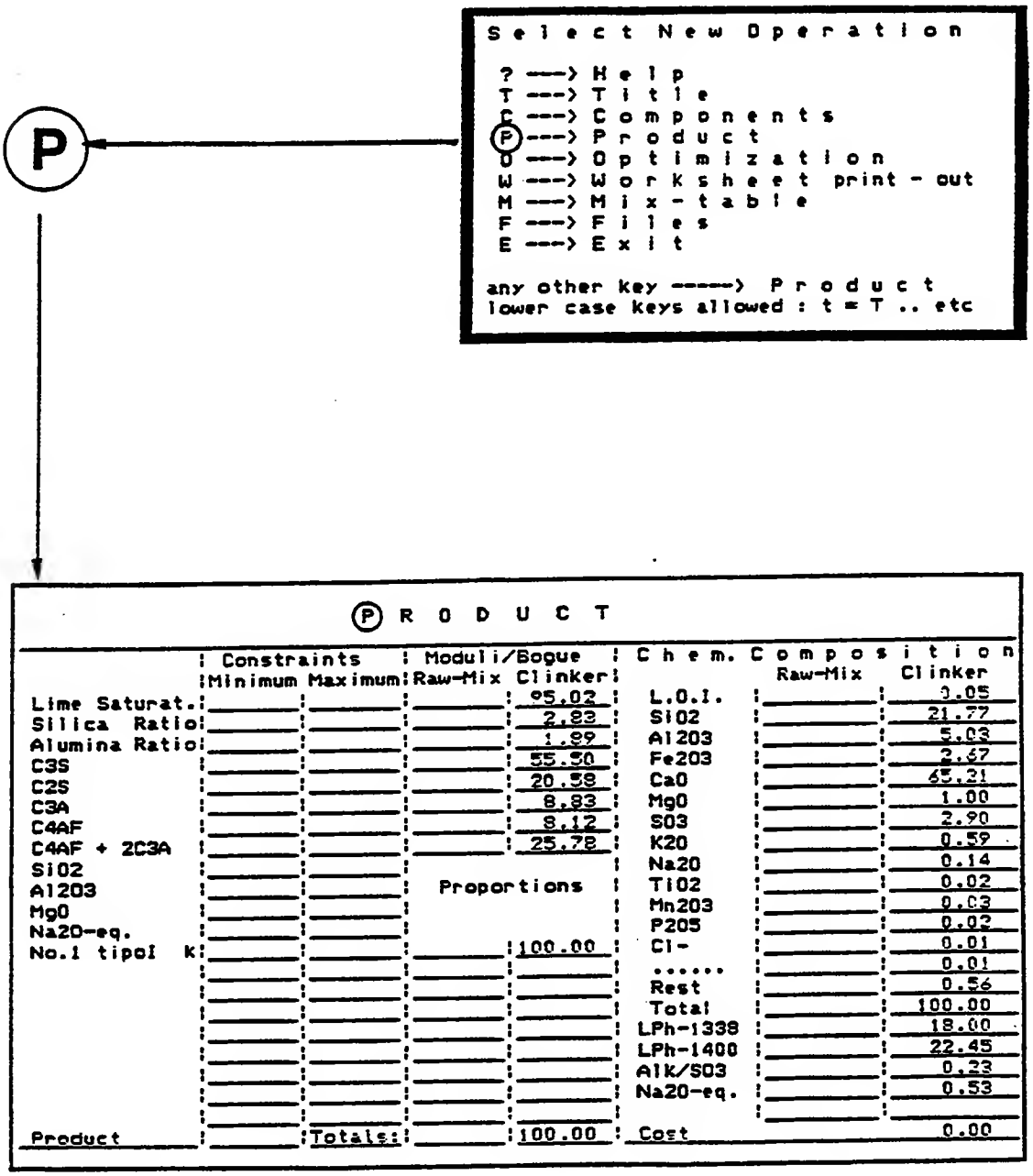

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Fig. 4 Compilation of Results.

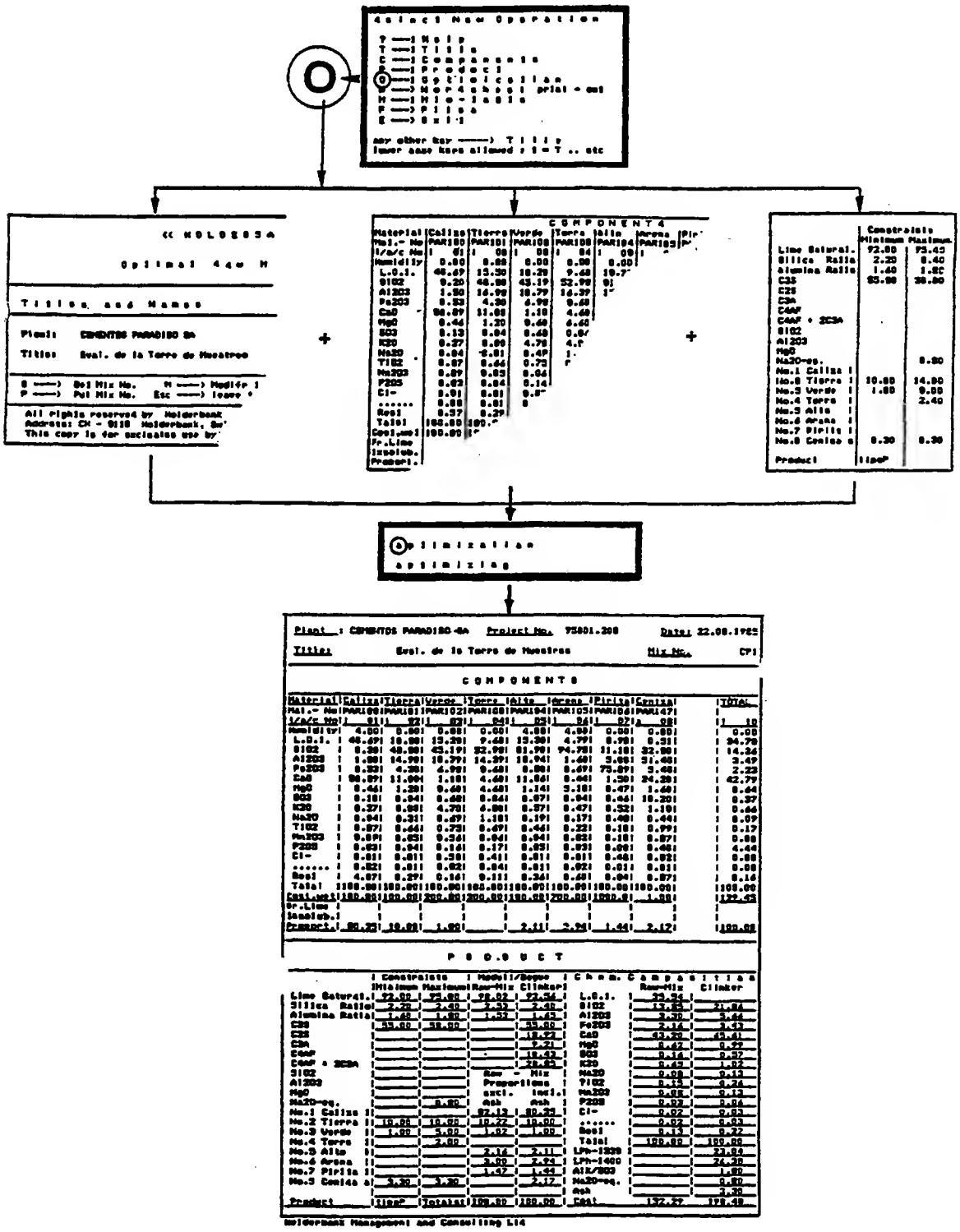
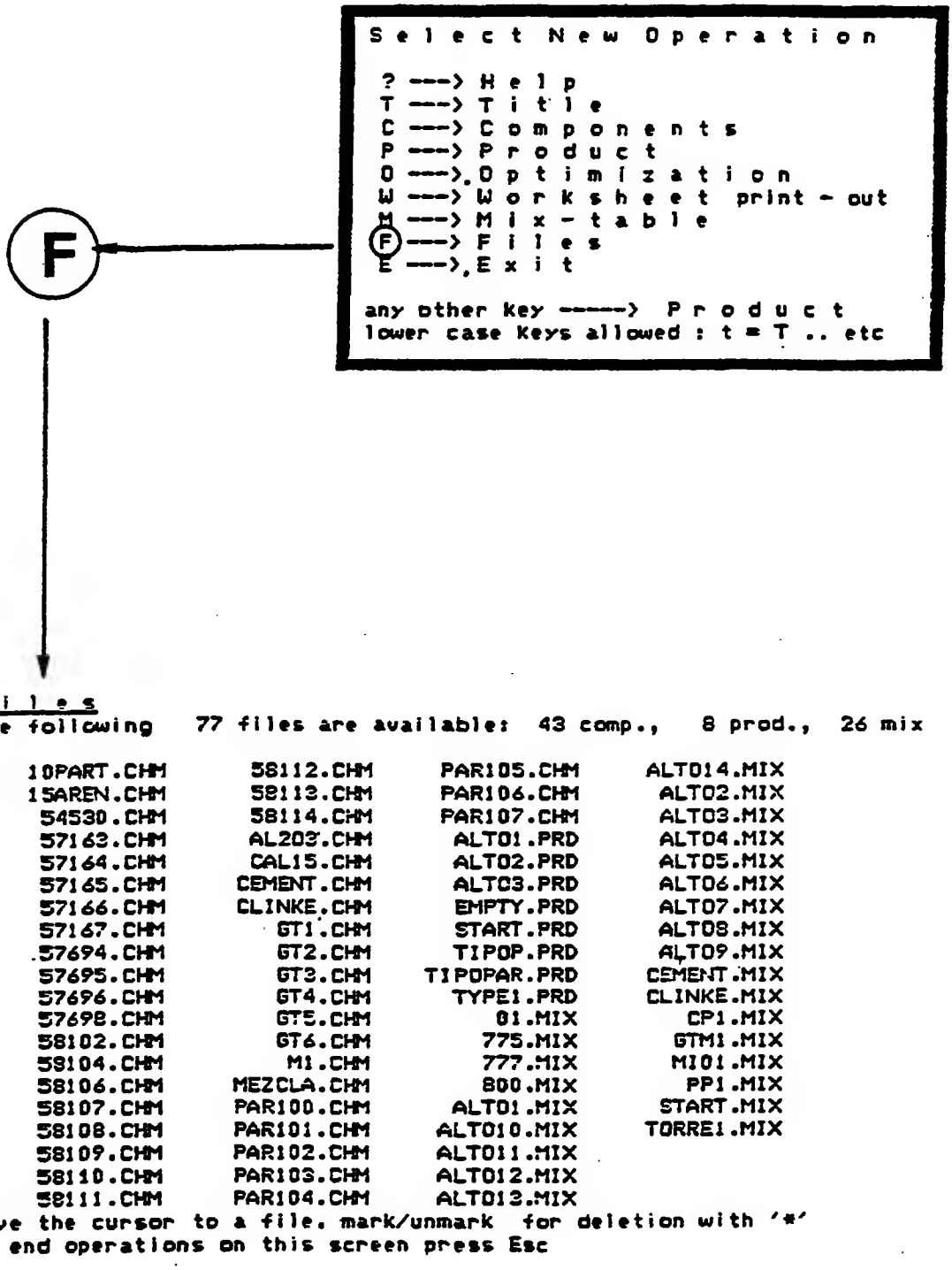


Fig. 5 Storage of Data, Constraints and Results.



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Chapter 14

Slurry Preparation

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Rheology 'Slurry preparation'

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Additional Information to Cement Course Lection "Slurry preparation"

1. INTRODUCTION

Several aspects to describe the slurry properties are known, some of them are mentioned below:

- Flow behaviour
- Granulometry
- Grindability
- Chemistry
- Pumpability
- Slump

In this paper the aspects of rheology are considered. Rheology describes the deformation of a body under the influence of stresses.

Ideal solids deform elastically. The energy of deformation is fully recovered when the stresses are removed.

Ideal fluids deform irreversibly - they flow. The energy of deformation is dissipated into the fluids in the form of heat, and it cannot be recovered just by releasing stresses.

Real bodies are neither ideal solids nor ideal fluids.

Only a few liquids of technical or practical importance come in their behaviour close to ideal liquids. The majority of liquids show rheologically a behaviour that classifies them to a region between the liquids and the solids: they are in varying extents both elastic and viscous and may therefore be named "viscoelastic".

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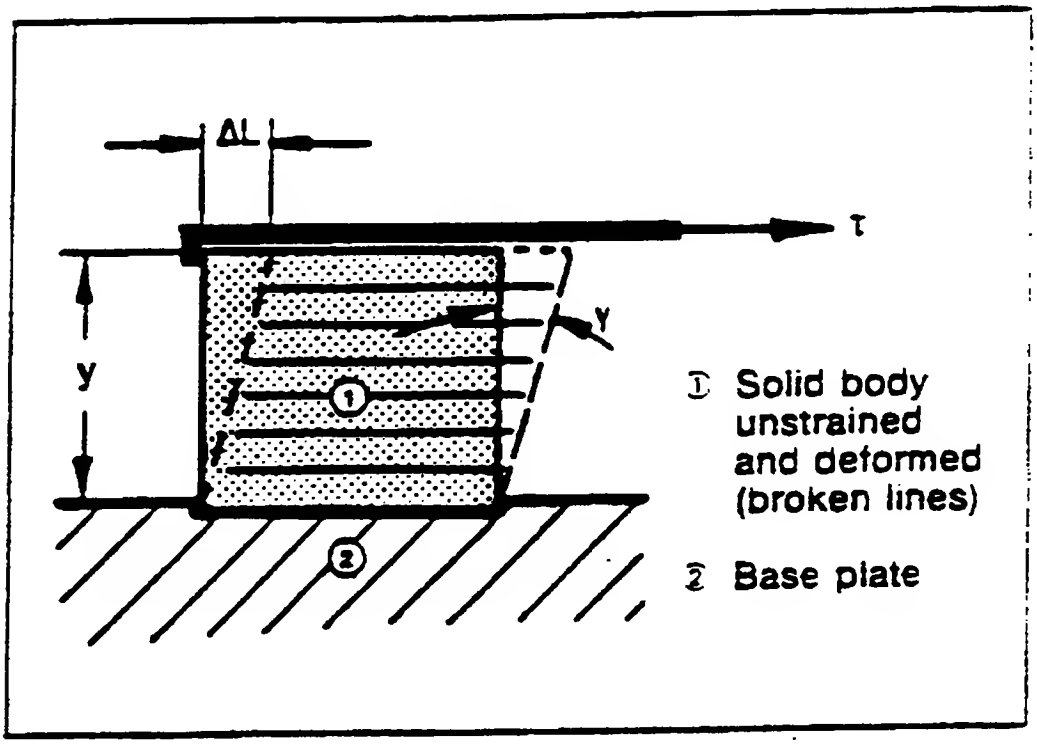
1.1 Ideal solids

Ideal Solids subjected to shear stresses react with strain.



$$\tau = G \cdot \frac{dL}{dy} = G \cdot \tan \gamma \approx G \cdot \gamma$$

τ = shear stress = force / area

G = Young modulus = correlation factor

γ = strain

y = height of the solid body

ΔL = deformation of the body as a result of shear stress

The Young's modulus in this equation is a correlating factor linked mainly to the chemical-physical nature of the solid involved. It defines the resistance of the solid against deformation.

1.2 Liquids

The resistance of a fluid against any irreversible positional change of its volume elements is called viscosity. To maintain flow in a fluid, energy must be added continuously.

Viscometry deals specifically with the measurement of the flow behaviour of liquids including those showing a viscoelastic behaviour.

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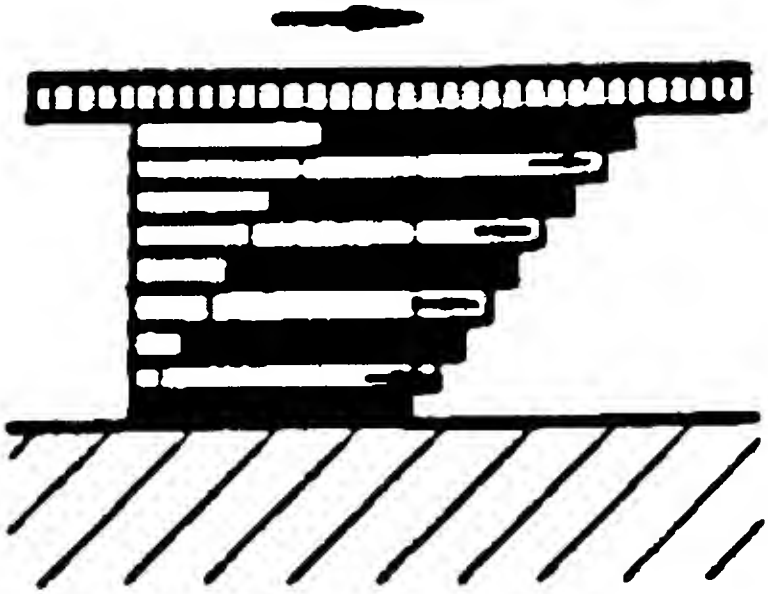
1.2.1 Flow Models

Shear induced flow in liquids can occur in 4 model cases:

1.2.1.1 Flow between two parallel flat plates

One plate moves and the other is stationary.

Flow between two parallel flat plates



1.2.1.2 Flow in the annular gap between two concentric cylinders

One of the two is assumed to be stationary while the other can rotate. This flow can be understood as the displacement of concentric layers sitting inside of each other. A Flow of this type is realised in rotational viscometers with coaxial cylinder sensor systems.

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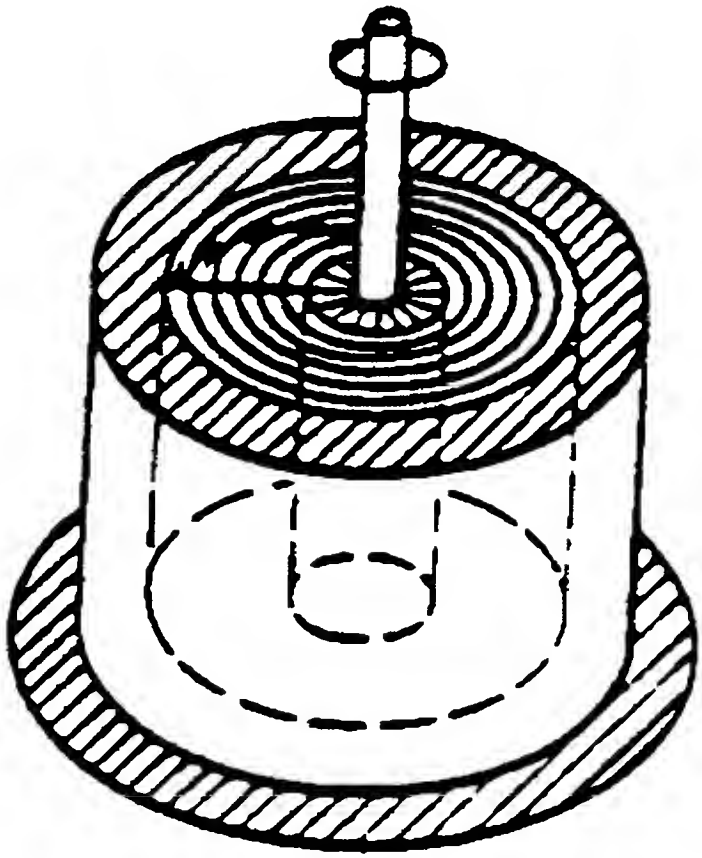
Flow in the annular gap between two concentric cylinders



1.2.1.3 Flow through pipes

A difference of pressure between their in and outlets forces the liquid to flow with a parabolic flow speed distribution across the diameter. This resembles a telescopic displacement of tube-like liquid layers sliding over each other.

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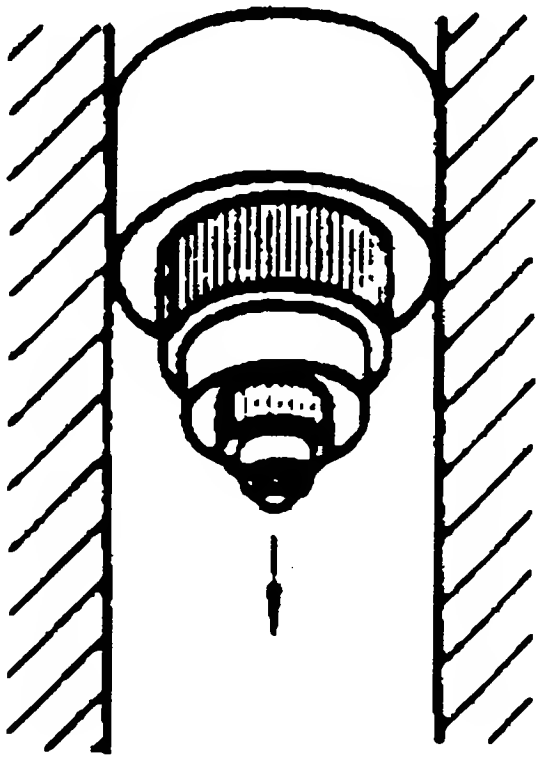
Flow through pipes



1.2.1.4 Flow between tow parallel plates or between a come and a plate

When one of the two is stationary and the other rotates. This model resembles twisting a roll of coins causing coins to be displaced by a small angle with respect to adjacent coins. This type of flow is realised in rotational viscometers with plate/plate or cone/Plate sensor systems.

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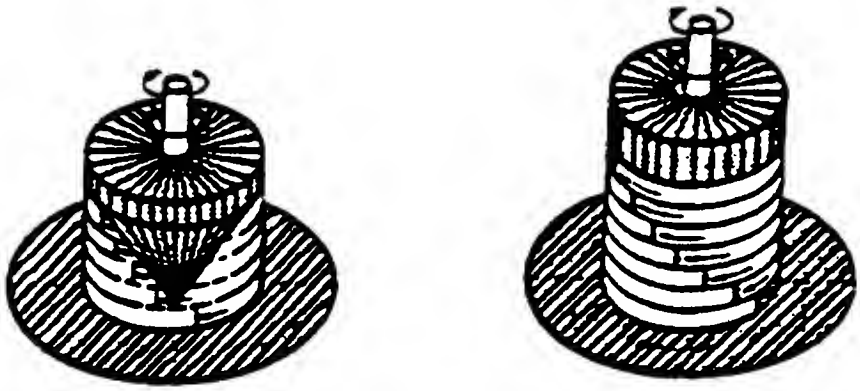
Flow between two parallel plates or between a come and a plate



2. LAW OF VISCOSITY

To measure the viscosity of liquids requires first the definition of the parameters which are involved in the flow. Then one has to find suitable test conditions which allow the measurement of flow properties objectively and reproducibly.

Isaac Newton was the first to find the basic law of viscometry describing the flow behaviour of an ideal liquid:

$\tau = \eta \times D$

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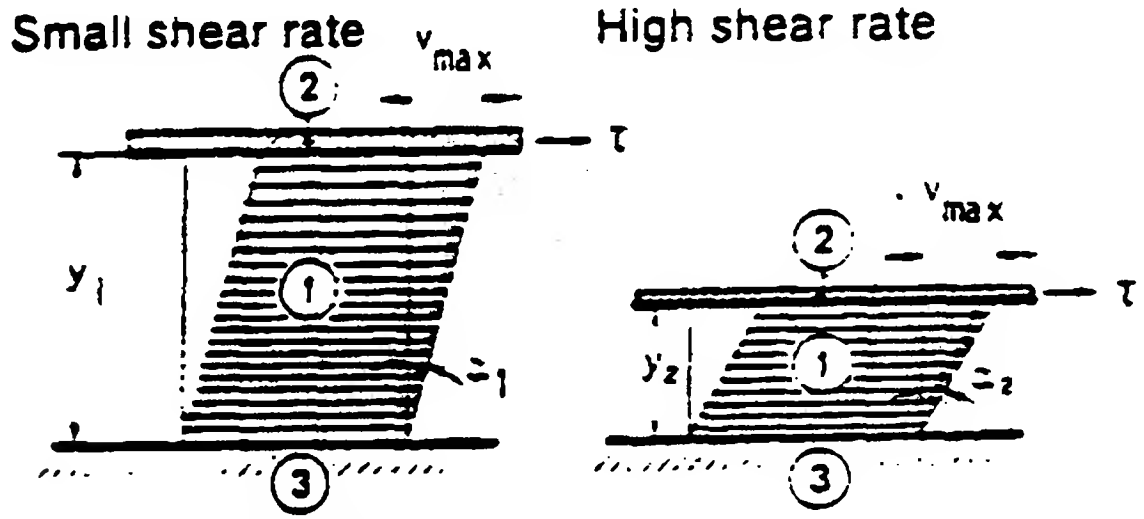
The parallel plate model helps to define both shear stress and shear rate:



① Liquid sheared
② Moving plate with surface area A to contact the liquid below
③ Stationary plate

2.1 Shear stress

A force F applied to an area A being the interface between the upper plate and the liquid underneath leads to a flow in the liquid layer. The velocity of flow that can be maintained for a given force will be controlled by the internal resistance of the liquid, i.e. by its viscosity.

$$\tau = \frac{F\,(\text{force})}{A\,(\text{area})} = \frac{N}{m^2}\,Pa\,(\text{Pascal})$$

2.2 Shear rate

The shear stress τ causes the liquid to flow in a special pattern. A maximum flow speed V_{max}. will be found at the upper boundary.

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The speed drops across the gap size y down to $V_{min} = 0$ at the lower boundary contacting the stationary plate. Laminar flow means that infinitesimally thin liquid layers slide on top of each other, similar to cards in a deck-of-cards. One laminar layer is then displaced with respect to the adjacent ones by a fraction of the total displacement encountered in the liquid between both plates.

In its general form the shear rate D is defined by a differential:

$$D=\frac{dv}{dx}$$

$$D=\frac{m/s}{m}=\frac{1}{s}[s^{-1}]$$

In case of a linear speed drop across the gap the differential in the equation above can be approximated by

$$D=\frac{V_{max}}{y}[S^{-1}]$$

2.3 Dynamic viscosity

Solving above equation for the dynamic viscosity η gives:

$$\eta=\frac{\tau}{D}$$

$$\eta=\frac{N}{m^2}\times s=[Pa\times s]$$

The unit of dynamic viscosity η is the "Pascal-second" [PA x s]. Often used is also the unit "milli Pascal x second" [mPa x s].

1 Pa x s = 1000 mPa x s

It is worthwhile noting that the previously widely used units of "cementPoise" (cP) of the dynamic viscosity η are interchangeable with the (mPa x s).

1 mPa x s = 1 cP

Typical viscosity values at 20° C:

Substance	mPa x s
petrol	0,65
water	1,0
mercury	1,5
blood (at 27°C)	4 - 15
honey	$\approx 10^4$
bitumen	$\approx 10^8$

2.4 Kinematic viscosity

When Newtonian liquids are tested by means of capillary viscometers, viscosity is determined in units of kinematic viscosity v. The force of gravity is acting as the force driving the liquid sample through the capillary. The density of the sample is one other additional parameter.

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Kinematic viscosity v and dynamic viscosity η are linked:

$$v = \frac{\eta}{\delta}$$

v = Kinematic viscosity given in the units of $\frac{mm^2}{s}$

Former units of kinematic viscosity were:

"Stokes" [St] or "centi Stokes" [cSt].

1 St = 100 cSt.

$$1\frac{mm^2}{s} = 1cSt$$

δ = density (kg/m^3)

3. FLOW AND VISCOSITY CURVES

3.1 Viscosity parameters

Viscosity describing the physical property of a liquid to resist shear induced flow may depend on up to 5 independent parameters:

$$\eta = f(S, T, P, D, t)$$

(S) This parameter denotes the physical-chemical nature of a substance being the primary influence on viscosity, i.e. whether the liquid is water, oil, honey, etc.

(T) This parameter is linked to the temperature of the substance. Viscosity is heavily influenced by changes of temperature. As an example: The viscosity of mineral oils drops typically by 10% for a temperature increase of only 1°C.

(P) This parameter "Pressure" is not experienced as often as the previous ones. Pressure compresses fluids and thus increased intermolecular resistance. Liquids are compressible under the influence of very high pressures - similar to gases but very much less. Increases of pressure tend to increase the viscosity.

(D) Parameter "shear rate D" is a decisive factor influencing the viscosity of very many liquids. Increasing shear rates may decrease or increase the viscosity.

(t) Parameter "time" denotes the phenomenon that the viscosity of some substances, usually dispersions, depends on the previous shear history, i.e. on the length of time the substance was subjected to continuous shear or was allowed to rest before being tested.

3.2 Flow Curve

The correlation between shear stress and shear rate defining the flow behaviour of a liquid is graphically displayed in a diagram of τ on the ordinate and D on the abscissa. This diagram is called the "Flow Curve".

The most simple type of a flow curve is shown below. The viscosity is assumed to be constant and independent of D.

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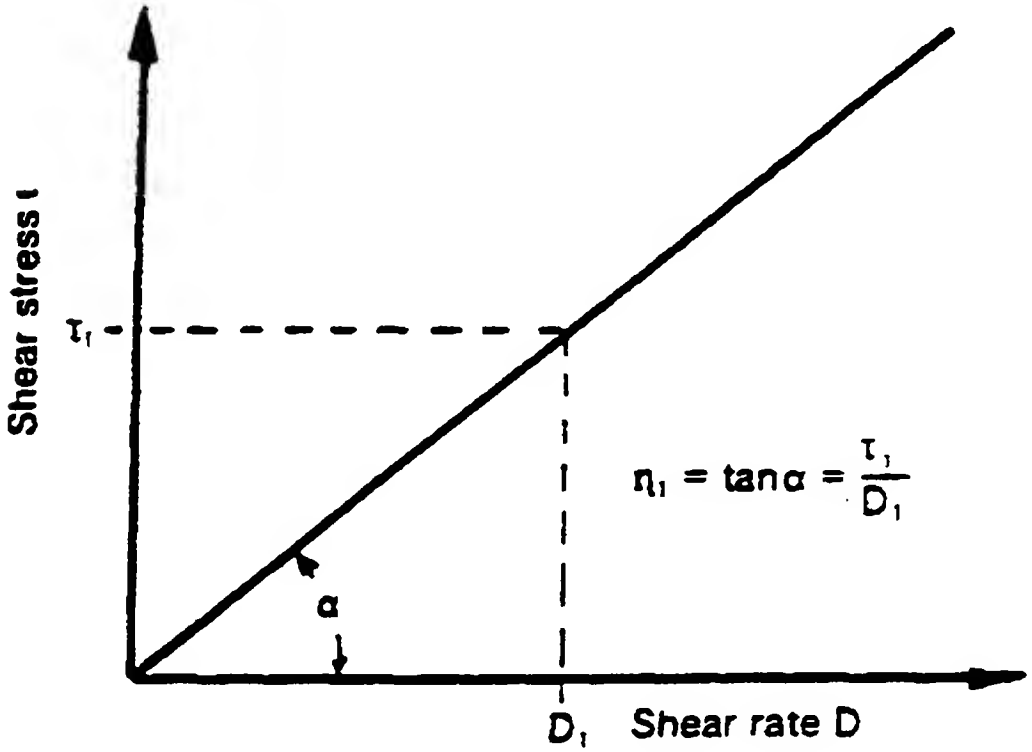
Simple Flow Curve



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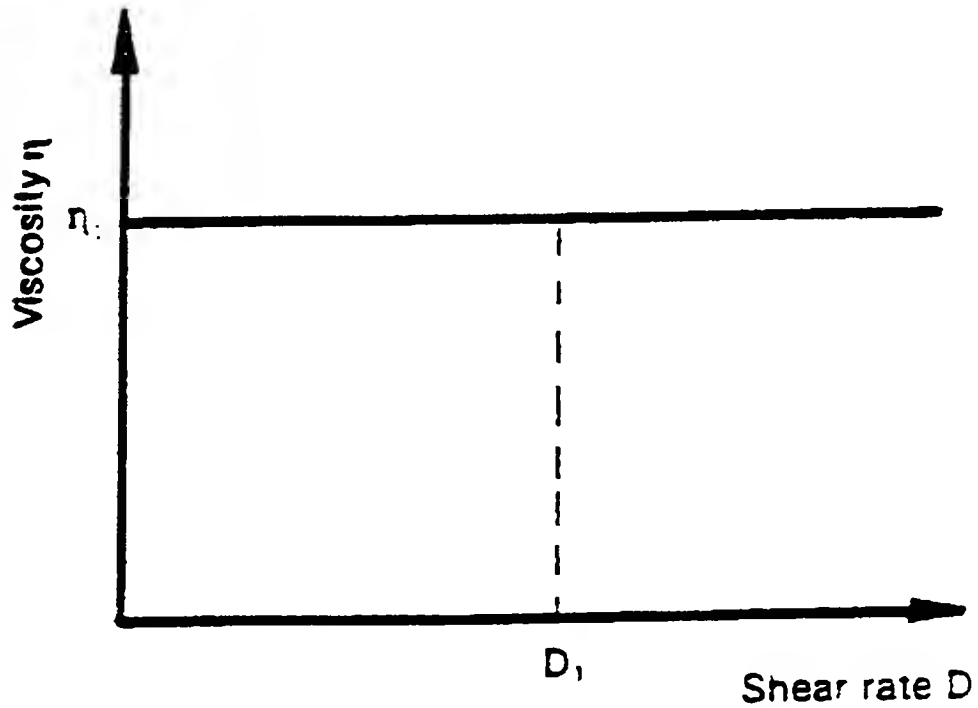
3.3 Viscosity Curve

Another diagram is very common: η is plotted versus D. This diagram is called the "Viscosity Curve":

Viscosity Curve



Viscosity measurements lead always first to the flow curve. Its results can then be rearranged mathematically to allow plotting the corresponding viscosity curve. The different types of flow curves have their counterparts in types of viscosity curves.

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4. SUBSTANCES

4.1 Newtonian liquids

Newton assumed that the graphical equivalent of his equation for an ideal liquid would be a straight line starting at the origin of the flow curve and would climb with a slope of an angle α. Any point on this line defined pairs of values for τ and D. Dividing one by the other gives a value of η. This value can also be defined as the tangent of the angle α.

Because the flow curve for an ideal liquid is straight, the ratio of all pairs of τ and D-values belonging to this line is a constant. This means, that η is not effected by changes of shear rate. All liquids for which this statement is true are called "Newtonian".

Examples: water, mineral oils, bitumen, molasses, etc.

4.2 Non-Newtonian liquids

All other liquids not showing this simple "ideal" flow behaviour are called "non-Newtonian". They outnumber the ideal liquids by far.

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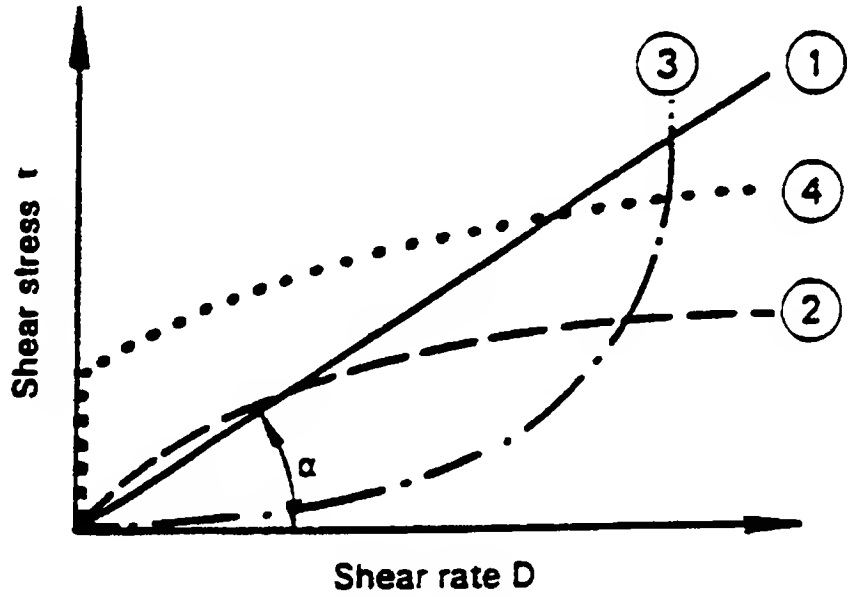
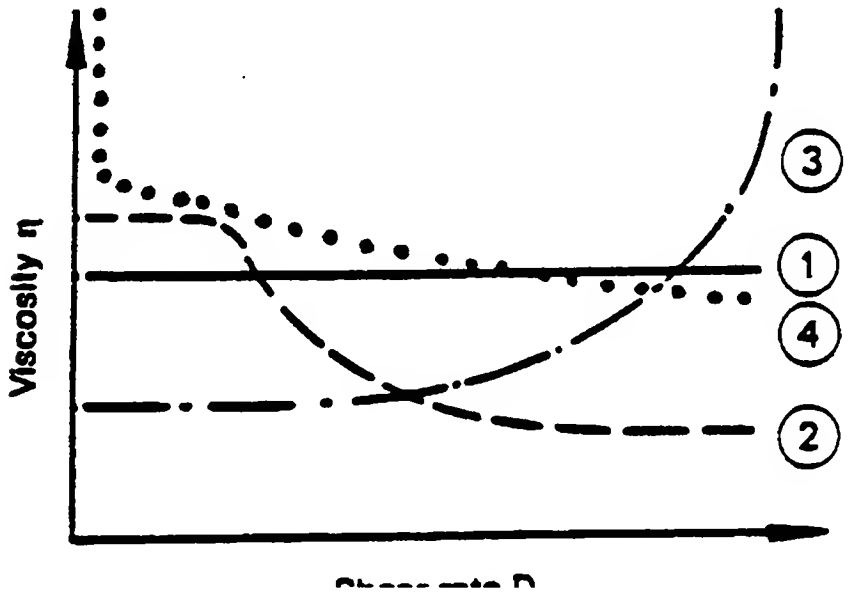
Flow and Viscosity Curves for non-newtonian liquids

Flow curves



① Newtonian liquid
Non-Newtonian liquids:
② Pseudoplastic liquid
③ Dilatant liquid
④ Pseudoplastic liquid with yield point = plastic liquid

Viscosity curves



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4.2.1 Pseudoasticity

Very many liquids show drastic viscosity decreases when the shear rates are increased from low to high levels.

Technically this can mean that for a given force or pressure more material can be made to flow, or the energy can be reduced to sustain a given flow rate. Materials which are thinning due to increasing shear rates are called "pseudoplastic". Very many substances such as emulsions, suspensions, or dispersions belong to this group.

4.2.1.1 Mechanism Leading to Pseudoplasticity

4.2.1.1.1 Entropy effect

Many liquid products that seem homogeneous throughout are in fact composed of several ingredients: particles of irregular shape or droplets of one liquid are dispersed in another liquid. On the other hand there are polymer solutions with long entangled and looping molecular chains. At rest all of these materials will maintain an irregular internal order and correspondingly a sizable internal resistance against flow, i.e. a high viscosity.

With increasing shear rates matchstick like particles suspended in the liquid will be turned lengthwise into the direction of the flow. Chaintype molecules in a melt or in a solution can disentangle, stretch, and orient parallel to the driving force. Particle or molecular alignments allow particles and molecules to slip past each other more easily and this shows up as reduces viscosity.

4.2.1.1.2 Disintegration

Shear can also induce irregular lumps of aggregated primary filler particles to break up and this also helps a material with such a filler to flow faster at a given shear stress.

4.2.1.1.3 Reversibility

For most liquid materials the shear thinning effect is reversible - often with some time lag - i.e. the liquids regain their original high viscosity when the shearing is slowed down or is even terminated: the chain-type molecules return to their natural state of non-orientation, deformed droplets return to ball-shape and the aggregates reform due to the Brownian motion.

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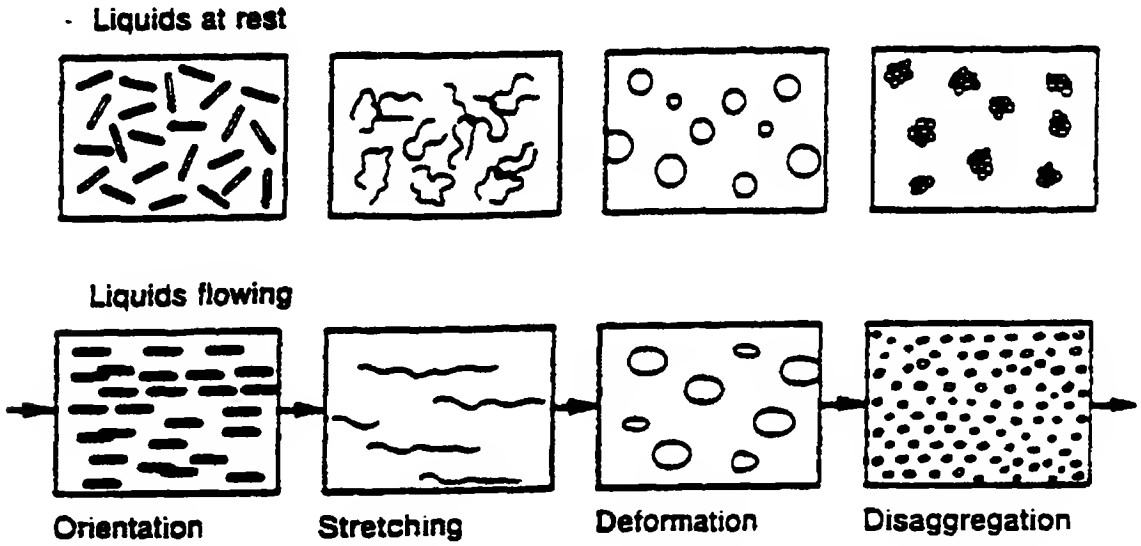
Reversibility of shear thinning effect



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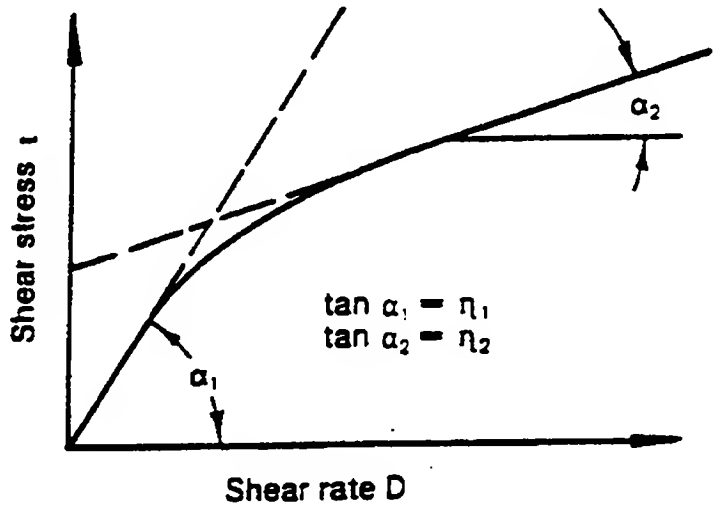
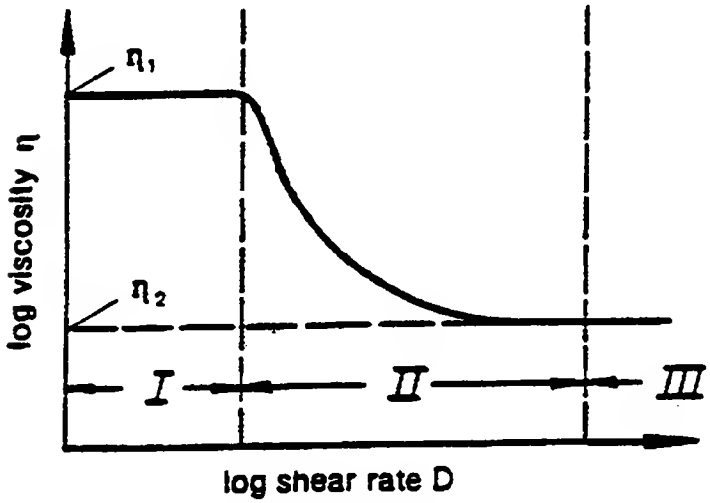
Shear thinning of pseudoplastic liquids is not uniform over the range of very low to very high shear rates:

Non-Uniform Shear thinning of pseudoplastic liquids

Flow curve



Viscosity curve



I: First Newtonian range – viscosity η_1 not yet dependent on shear rate.

II: In this range viscosity drops as the result of molecular or particle orientation.

III: Second Newtonian range – viscosity η_2 stays constant independent of further increases of shear rate.

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At the low shear rate range the Brownian movement of molecules keeps all molecules or particles at random in spite of initial effects of shear orientation. At very low shear rates pseudoplastic liquids behave similarly to Newtonian liquids having a defined viscosity independent of shear rate. Then follows a shear rate change when the shear rate induced molecular or particle orientation by far exceeds the randomising effect of the Brownian movement: the viscosity drops drastically. Finally the viscosity approaches a finite level. Going to even higher shear rates cannot cause further shear thinning: The optimum of perfect orientation has already been reached. In the low and in the high shear rate ranges - called the first and second Newtonian ranges - the viscosity even of non-Newtonian liquids is more or less independent of shear rate.

4.2.2 Dilatancy

There is one other type of material characterised by a shear rate dependent viscosity: "dilatant" substances increase their viscosity whenever shear rates increase.

Dilatant flow behaviour is found for example in highly concentrated suspensions in which solid particles are mixed with liquids such as plasticizers. The particles are densely packed and the amount of plasticizer added is just sufficient to fill the voids between the particles. At rest or at low coating speeds the plasticizer fully lubricates the particles surfaces and thus allows an easy positional change of particles when forces are applied: this suspension behaves as a liquid at low shear rates. At higher shear rates particles will wedge others apart causing a general volume increase of the plastisol. At this stage the plasticizer's share of the total volume decreases. No longer being sufficient to fill all voids and to keep particle surfaces fully lubricated the plastisol becomes more viscous.

Dilatancy in liquids is rare.

4.2.3 Plasticity

It describes pseudoplastic liquids which additionally feature a yield point. Plastic liquids can be classified with good reasoning to belong to both liquids and solids. They are mostly dispersions which at rest can build up an intermolecular/interparticle network of binding forces (polar forces, van der Waals forces, etc.). These forces restrict positional change of volume elements and give the substance a solid character with an infinitely high viscosity. Forces acting from outside, if smaller than those forming the network, will deform the shape of this solid substance elastically. Only when the outside forces become so big that they can overcome the network forces - surpass the threshold shear stress calls "yield point" - does the network collapse. Volume elements can now change positions irreversibly: the solid turns into a flowing liquid.

Typical substances showing yield points: oil well drilling muds, greases, lipstick masses, toothpastes. Plastic liquids have flow curves which intercept the ordinate not at the origin, but the yield point level of τ_0.

4.2.4 Thixotropy

This term describes a rheological phenomenon of great industrial importance. It calls for some explanations in simplified terms of an otherwise often very complex molecular of particle interaction:

In pseudoplastic liquids thinning under the influence of increasing shear depended mainly on the particle/molecular orientation or alignment in the direction of flow. This orientation will again be lost just as fast as orientation came about in the first place.

Plotting a flow curve with a uniformly increasing shear rate - the "up curve" - one will find that the "down-curve" plotted with uniformly decreasing shear rates will just be superimposed on the "up-curve".

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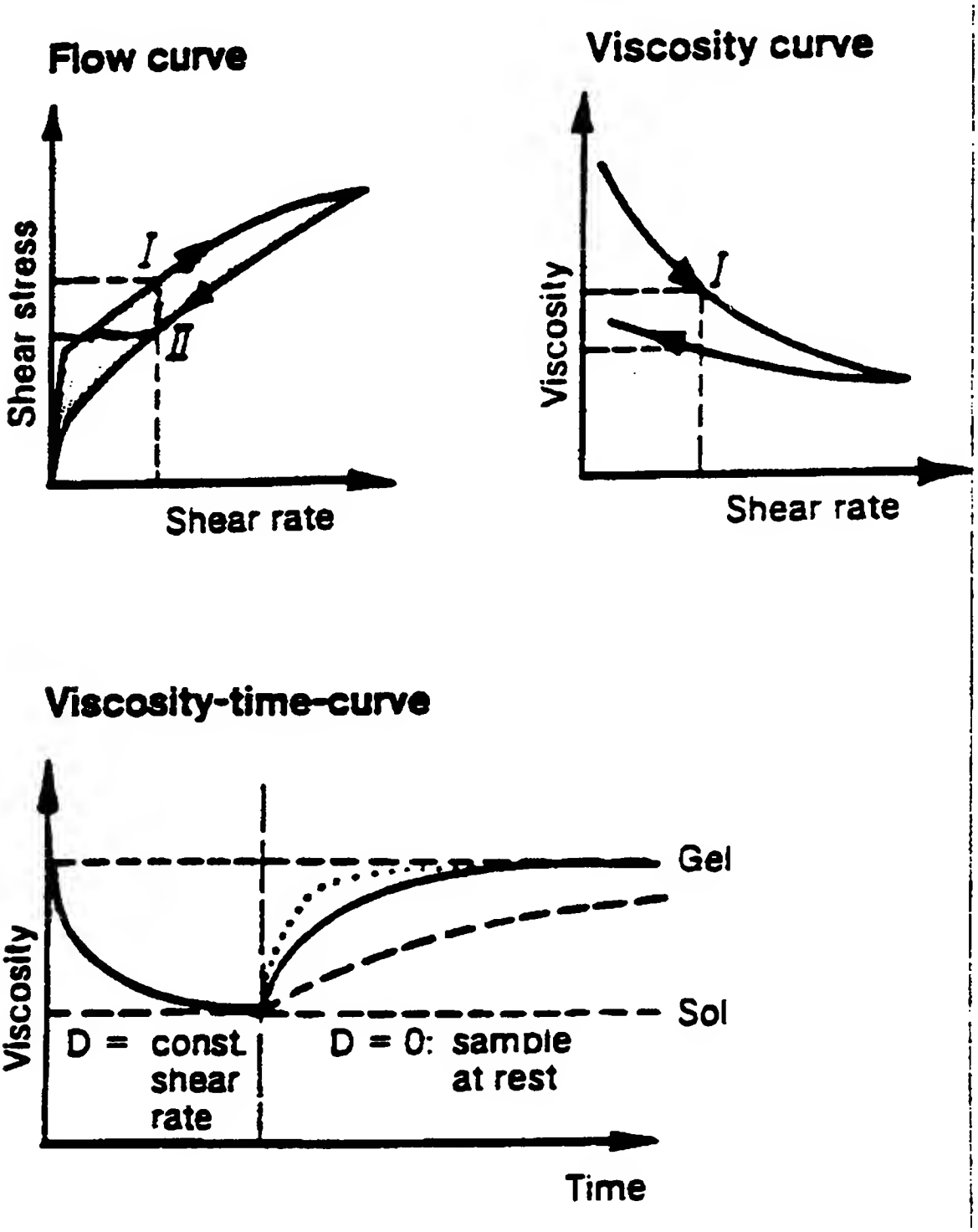
In the flow curve the "up-curve" is no longer directly underneath the "down-curve".

Flow and Viscosity Curves demonstrating Thixotropy

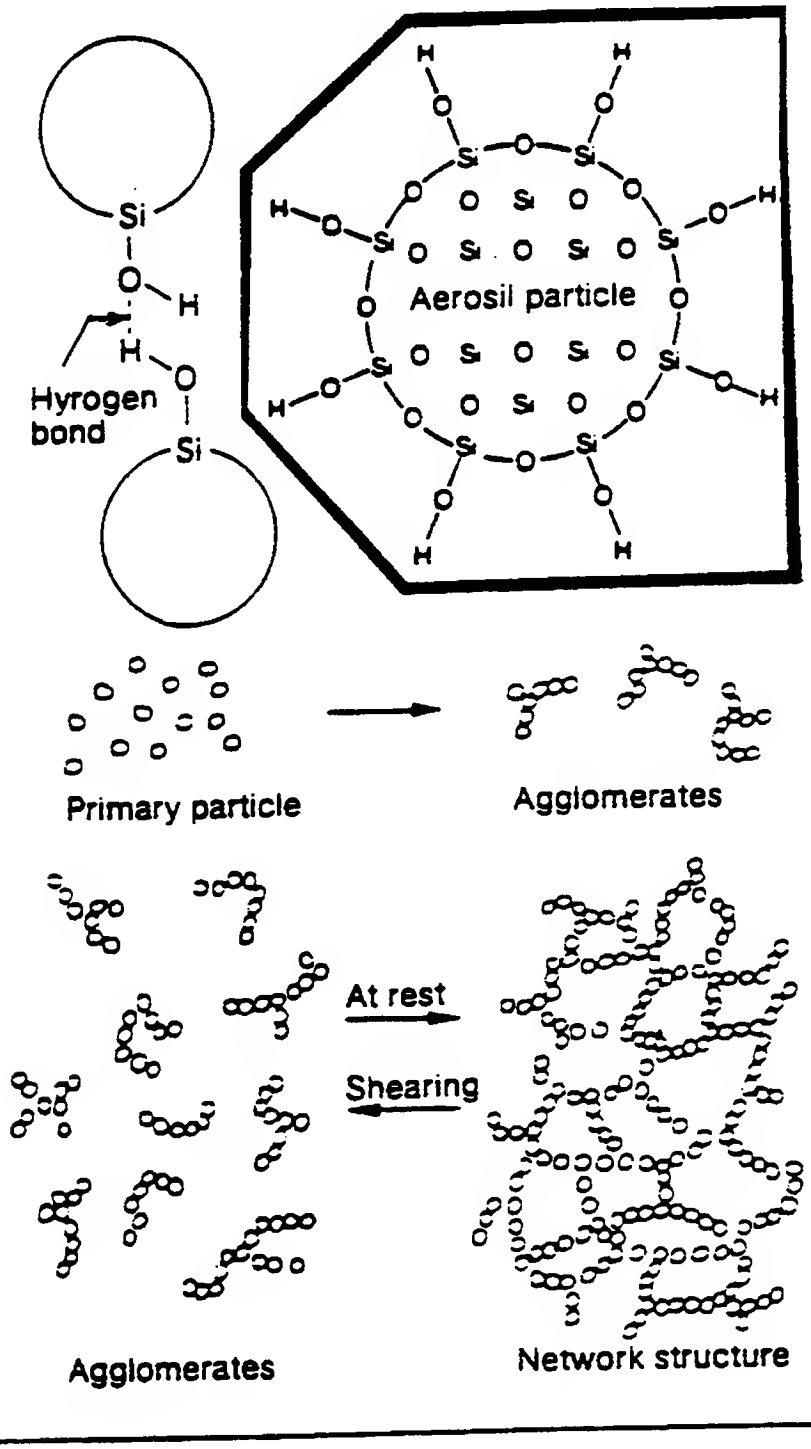


The hysteresis now encountered between these two curves surrounds an area "A" that defines the magnitude of this property called thixotropy. This area has the dimension of "energy" related to the volume of the sample sheared which indicates that energy is required to break down the thixotropic structure.

4.2.4.1 Mechanism

It is typical for many dispersions that they not only show this potential for orientation but additionally for a particle/molecule-inter-action. This will lead to bonds creating a three-dimensional network structure which is often calls a "gel". In comparison to the forces within particles or molecules, these bonds - they are often hydrogen or ionic bonds - are relatively weak: they rupture easily, when the dispersions is subjected to shear over an extended period of time.

Thixotropy Mechanism



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When the network is disrupted the viscosity drops until it asymptotically reaches the lowest possible level for a given constant shear rate. This minimum viscosity level describes the "sol"-status of the dispersion. A thixotropic liquid is defined by its potential to have its structure reformed, when the substance is allowed to rest for an extended period of time. The change of a gel to a sol and of a sol to a gel is reproducible any number of times.

4.3 Rheonexy

Rheopective liquids are characterised by a viscosity increase related to the duration of shear. When these liquids are allowed to rest they will recover the original - i.e. the low - viscosity level. Rheopexy and thixotropy are opposite flow properties.

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5. BOUNDARY CONDITIONS

The mathematical treatment of the physical problem of a fluid being deformed under the influence of forces will lead to very complex differential equations which in their general form cannot be solved. The problem can only lead to an acceptable mathematical solution when test conditions are kept within the framework of these restrictions the Newton's law of viscometry is valid.

The measurement of viscosity requires test conditions providing:

5.1 Laminar flow

The applied shear must lead only to laminar flow. In as much as laminar flow prevents exchange of volume elements between layers samples must be homogeneous to start with. A homogenising effect on heterogeneous samples during testing cannot be expected or permitted.

5.2 Steady state flow

In Newton's law of viscometry the shear stress applied was correlated to shear rate. The shear stress meant was the one that was just sufficient to sustain a constant flow rate. The additional energy required to accelerate or to decelerate the flow cannot be accounted for in the equation.

5.3 No slippage

The shear stress applied must be transmitted from the moving plate across the liquid boundary layer into the liquid. In case that adherence between the moving plate and the liquid is insufficient to transmit the shear stress - the moving plate slips above the non-moving liquid sample - any test results will be meaningless. Problems with slippage often arise with fats and greases.

5.4 Samples must be homogeneous

This requirement means that the sample must react to shear uniformly throughout. If samples are dispersions or suspensions then all ingredients, droplets, or bubbles have to be very small with respect to the thickness of the liquid layer sheared.

In viscometry really homogeneous samples are rare. If dispersions are considered homogeneous because every volume element contains an identical share of all the ingredients, then problems can still occur when the increasing shear encountered during the measurements leads to phase separation, i.e. the dispersion separates into a liquid and a solid layer.

5.5 No chemical or physical changes in the sample during testing

Changes due to processes such as hardening or degradation would be secondary influences on viscosity.

5.6 No elasticity

Samples should be purely viscous. Very many samples such as polymer melts or polymer solutions show a viscous and an elastic response to shear stresses or shear rates. When the latter becomes sizable test results can no longer be evaluated in terms of viscosity.

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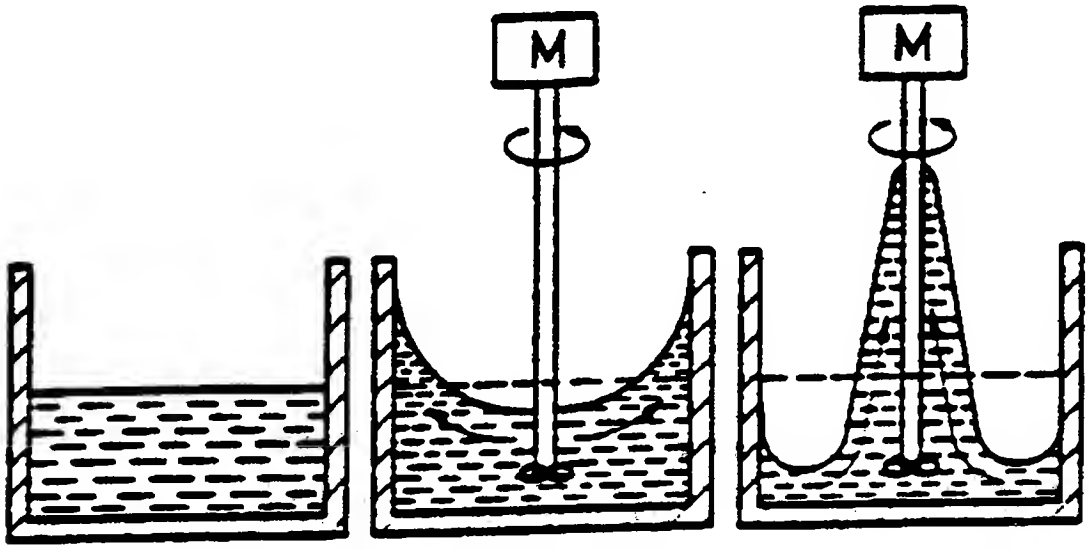
Comparison between Viscous and Elastic Liquids



Liquid at rest: rheological phenomena are not apparent.

Viscous liquid: rotating shaft creates centrifugal forces that throw volume elements outwards.

Elastic liquids: rotating shaft creates normal forces exceeding centrifugal forces which means that volume elements are pulled upwards.

In rotational viscometers with coaxial cylinder sensor systems for some samples the elastic response to shear may be so great as to "suck" all the substance out of the annular gap.

Elasticity is a characteristic property of a sample which cannot be subdued by any means. When measuring the viscosity of viscoelastic samples, the maximum of shear rate must be limited to keep the normal stresses from becoming too large so that they ruin the test results.

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6. ROTATIONAL VISCOMETERS

6.1 Indication of different models

The principle of rotational viscometers with coaxial cylinder sensor systems and with cone-and-plate sensor systems allows the design of excellent and versatile absolute viscometers. The range of rotational viscometers on the world market varies widely in sophistication and in price. The rheological criteria and boundary conditions mentioned before are being used to grade types of instruments and explain design features and resulting areas of application.

One might imagine that the coaxial cylinder sensor systems for rotational viscometers result from bending both flat plates of the Newton's parallel-plate-model into an inner and outer cyliner. A liquid sample filling the annular gap between the two cylinders can be exposed to shear. Conditions will lead to laminar flow and allow a mathematical treatment of the problem.

6.1.1 Preselect: t look for D

The inner or the outer cylinder is subjected to a defined shear stress or a defined torque while the other cylinder is held at rest. One can now measure the rate of rotation or shear rate resulting.

6.1.2 Preselect D: look for t

The inner or the outer cylinder rotates at a defined rotational speed while the other cylinder is held at rest. One can now measure the resulting shear stress or torque. This is the most common design for rotational viscometers. This type of viscometers can come in two versions:

6.1.2.1 Searle system

The inner cylinder - often called the "rotor" - rotates at a defined speed. The outer cylinder - often called the "cup" - is held at rest. The rotating inner cylinder forces the liquid in the annular gap to flow. The resistance of the liquid being sheared between the stationary and rotating boundaries results in a viscosity related torque working on the inner cylinder which counteracts the torque provided by the drive motor. A torque sensing element - normally a spring that twists as the result of the torque -. is placed between the drive motor and the shaft of the inner cylinder. The twist of the torque spring is a direct measure of the viscosity of the sample tested.

Most viscometer models made world-wide are Searle types. One reason for this tendency is that the good temperature control required for viscosity measurements is more easily accomplished when the outer cylinder does not rotate. Searle type viscometers are limited when low viscous samples have to be tested at high rotor speed. Centrifugal forces affecting the liquid can turn laminar flow to non-laminar flow and eventually to a fully developed turbulence.

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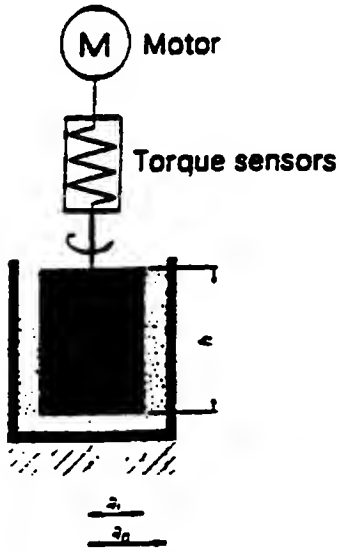
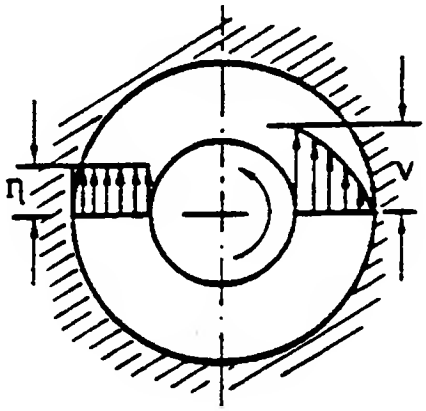
Searle Viscometers

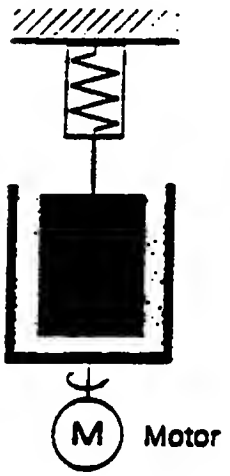


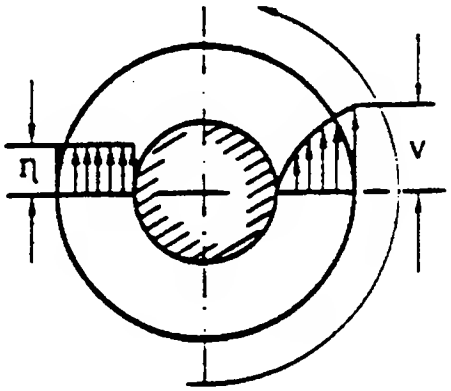


6.1.2.2 Couette system

The outer cylinder rotates at a defined speed. It forces the liquid sample in the annular gap to flow. The resistance of the liquid against being sheared transmits a viscosity related torque onto the inner cylinder, which would induce it also to rotate. This torque is measured by determining just what counteracting torque is required to hold the inner cylinder at stand still. Couette type viscometers are more stable with respect to centrifugal forces.

Couette Viscometers





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Materials Technology Slurry preparation

Memorandum Nr.: NIA 92/41 /E
Author: A. Edlinger
Date: April 24, 1992 (tg)
Order Nr.: 3026

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7. INTRODUCTION

Slurry preparation has technically and economically an influence in the so-called "Wet Process" and "Semi-wet" Process of clinker burning.

Economical Considerations

The "Holderbank" Group operates about 1/3 wet process of all plants world-wide (about 70 plants).

Subject of "Slurry preparation" therefore, has an outstanding importance on the economical situation of the whole group.

It is anticipated that optimisation and automatic control of slurry properties will reduce the water content at a minimum of 5% slurry as a long time average. The reduction will be in the range of 5%, as a long-term.

A reduction of 5% slurry water content saves about 400 kJ/kg cli. For the production of 11 Mio t of clinker per year this translates into a total reduction of 150'000 t coal per year which means about 10 Mio US$ saving potential per year for the whole group.

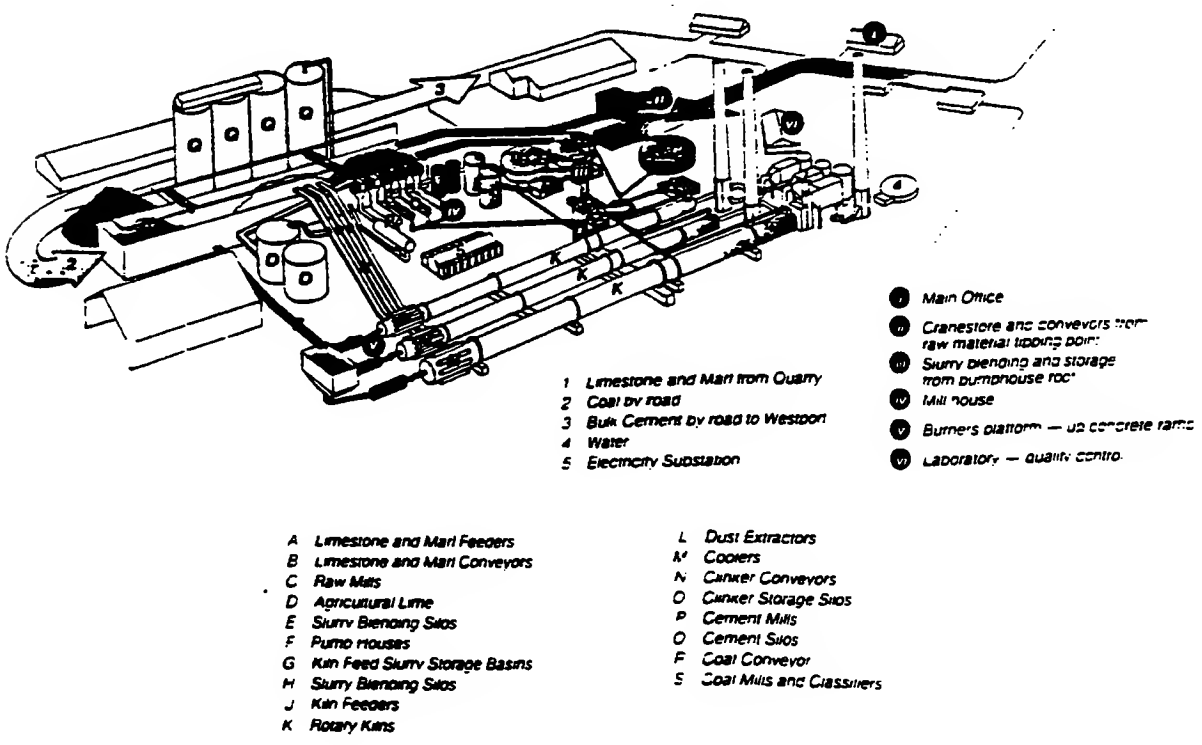
Wet Process

The wet process in the cement industry consists of finely grinding the raw materials, such as limestone, marl and clay with water in ball mills.

The fine slurry thus formed, which usually contains between 30 - 40% water, is then dried during the subsequent burning process in a long "wet" kiln in the so-called "Chain zone".

A typical plant layout (Milburn Cement, New Zealand) is shown in the following picture:

Fig. 1 Milburn Cement Plant.



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In a wet-process kiln, a reduction in slurry moisture results in a reduction in heat use, and since both the amount of water vapour and the amount of combustion products are decreased, this also results in lower exhaust fan power use. The water content of the slurry also influences the demand on pumping energy.

As a result of price increase in the past years of various types of energy carriers such as oil and coal, great efforts are being made to use as little water as possible in the grinding of raw slurries.

A minimum amount of water must be added because the slurry is transported by pumps and must therefore, demonstrate a sufficient flowability.

Process features

The advantages of the wet and semi-wet process are:

- low investment (fixed costs)
- simple and stabile process
- good compatibility with waste burning
- higher uniformity and quality of clinker
- simple operation and maintenance of the kiln.

The disadvantage is of course the higher thermal energy demand compared with the dry processes.

Slurry thinning

There are several possibilities for reducing slurry moisture while still maintaining pumpability. The most commonly used method is the addition of small amounts of viscosity-reducing agents. Other possibilities include carbonating the slurry with kiln exhaust gas, coarser grinding of the raw, material, preheating the slurry and, if alternative raw material is available, substituting shale for clay.

Target values

Slurry preparation controls following items:

- Heat consumption due to water evaporation
- Consumption of mechanical energy due to
- evaporated steam and waste gas ventilation
- pumping the raw slurry, etc.
- Dimensioning of Piping/Filtration processes
- Evaluation of pumping system
- Mixing/Blending/Feeding
- Sedimentation/Flotation
- Corrosion
- Filtration (semi-wet process)

Several aspects to describe the slurry properties are known, some important ones are mentioned below:

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- Water content
- Mineralogy
- Flow behaviour
- Granulometry
- Grindability
- Chemistry
- Pumpability
- Slump
- Specific grinding energy

8. RHEOLOGY

Rheology is the discipline describing the most important slurry property in a qualitative and quantitative manner.

There are off-line (laboratory) and on-line (process control) methods available.

Rheology describes the deformation of a body under the influence of stresses.

Ideal solids deform elastically. The energy of deformation is fully recovered when the stresses are removed.

Ideal fluids deform irreversibly - they flow. The energy of deformation is dissipated into the fluids in the form of heat, and it cannot be recovered just by releasing stresses.

Real bodies are neither ideal solids nor ideal fluids.

In their behaviour only a few liquids of technical or practical importance come close to ideal liquids. The majority of liquids show rheologically a behaviour that classifies them to a region between the liquids and the solids: they are in varying extents both elastic and viscous and may there-fore, be named "viscoelastic".

The resistance of a fluid against any irreversible positional change of its volume elements is called viscosity. To maintain flow in a fluid, energy must be added continuously.

Viscometry deals specifically with the measurement of the flow behaviour of liquids including those showing a viscoelastic behaviour.

Definition Viscosity

To measure the viscosity of liquids requires first the definition of the parameters which are involved in the flow. Then one has to find suitable test conditions which allow the measurement of flow properties objectively and reproducibly.

Isaac Newton was the first to find the basic law of viscosity describing the flow behaviour of an ideal liquid:

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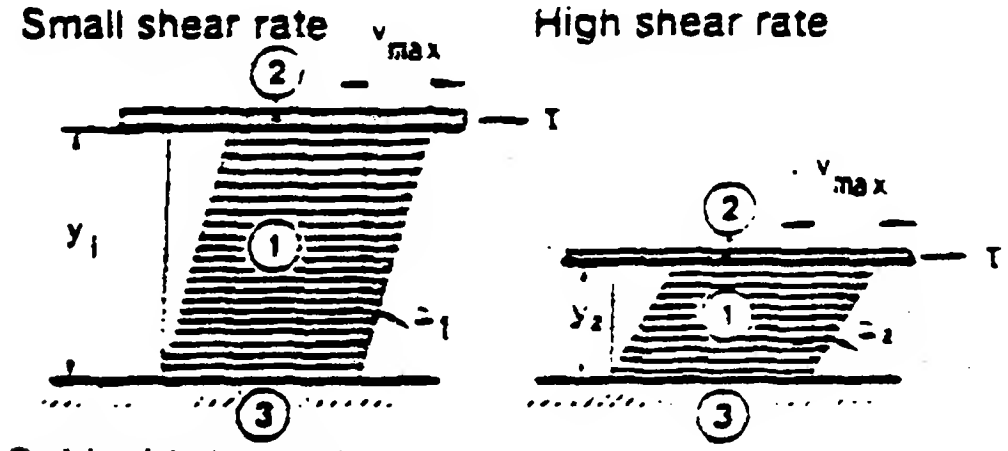
$\tau = \eta \times D$
η = Viscosity
τ = Shear stress
D = Shear rate

The parallel plate model helps to define both shear stress and shear rate:

Fig. 2 Shear rates.



Shear stress

A force F applied to an area A being the interface between the upper plate and the liquid underneath leads to a flow in the liquid layer. The velocity of flow that can be maintained for a given force will be controlled by the internal resistance of the liquid, i.e. by its viscosity.

$$\tau = \frac{F\,(\text{force})}{A\,(\text{area})} = \frac{N}{m^2} = Pa\,(\text{Pascal})$$

Shear rate

The shear stress r causes the liquid to flow in a special pattern. A maximum flow speed V_{max} will be found at the upper boundary.

The speed drops across the gap size *y* down to $V_{min} = 0$ at the lower boundary contacting the stationary plate. Laminar flow means that infinitesimally thin liquid layers slide on top of each other, similar to cards in a deck-of-cards. One laminar layer is then displaced with

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respect to the adjacent ones by a fraction of the total displacement encountered in the liquid between both plates.

In its general form the shear rate D is defined by a differential:

$$D=\frac{dv}{dx}$$

$$D=\frac{m/s}{m}=\frac{1}{s}[s^{-1}]$$

In case of a linear speed drop across the gap the differential in the equation above can be approximated by

$$D=\frac{V_{max}}{y}[S^{-1}]$$

Dynamic viscosity

Solving above equation for the dynamic viscosity gives:

$$\eta=\frac{\tau}{D} \qquad \eta=\frac{N}{m^2}\times s=[Pa\times s]$$

The unit of dynamic viscosity is the "Pascal-second" [Pa x s].

Viscosity Parameters

The viscosity of a slurry is a function of various factors

Viscosity describing the physical property of a liquid to resist shear induced flow may depend on up to 5 independent parameters:

$\eta = f(S, T, P, D, t)$

(S) This most important parameter denotes the physical-chemical nature of a substance being the primary influence on viscosity (see also next pages "Material parameter").

(T) This parameter is linked to the temperature of the suspension. Viscosity is heavily influenced by changes of temperature.

(P) This parameter npressure" is not experienced as often as the previous ones. Pressure compresses fluids and thus increase intermolecular re-sistance. Liquids are compressible under the influence of very high pressures - similar to gases but very much less. Increases of pressure tend to increase the viscosity.

(D) Parameter Ushear rate D" is a decisive factor influencing the viscosity of many liquids. Increasing shear rates may decrease or increase the viscosity.

(t) Parameter "time" denotes the phenomenon that the viscosity of some sub-stances, usually dispersions, depends on the previous shear history, i.e. on the length of time the substance was subjected to continuous shear or was allowed to rest before being tested.

Material Parameter (S)

These parameter (S) is to be differentiated into the following sub-parameters:

A) Water Content

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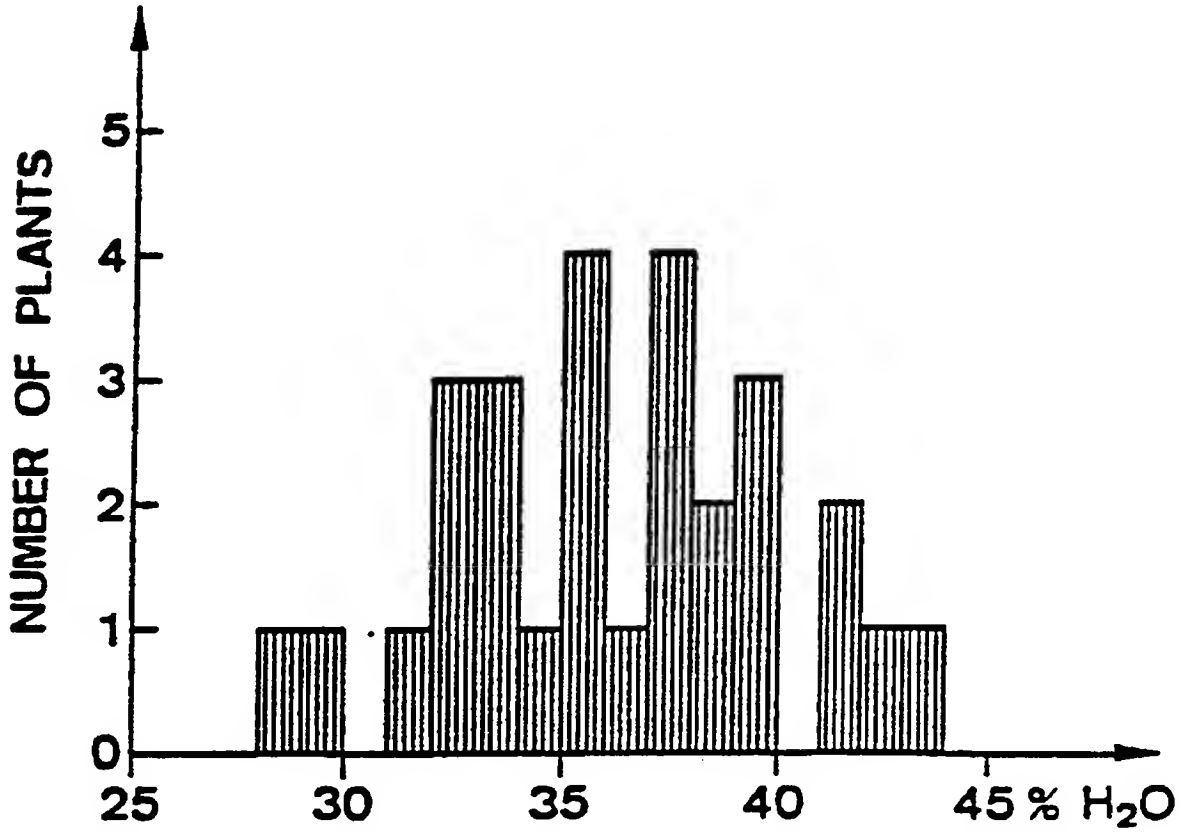
Approximately 30% of all "Holderbank"-Group plants (corresponding to approximately 35% of the total clinker production) are employing the wet-process for cement manufacture.

Fig. 3 Water content.



Generally it can be pointed out that the higher the water content, the lower the viscosity.

There are little restrictions with respect to the quality of the utilised water, i.e. water of many origin/source and quality can be used, such as:

- tap water
- industrial water/industrial waste liquids
- sewage water
- slightly brackish water.

The only quality aspects which have to be controlled are:

- content of corrosive elements (C1, 0)
- contents of harmful or poisoneous elements with regard to process, product and waste gases (C1, F, B, organic substances, etc.)
- content of smelling compounds in the waste water, etc.

B) Slurry Fineness

The fineness of slurry fluctuates in a wide range as demonstrated in the following diagram:

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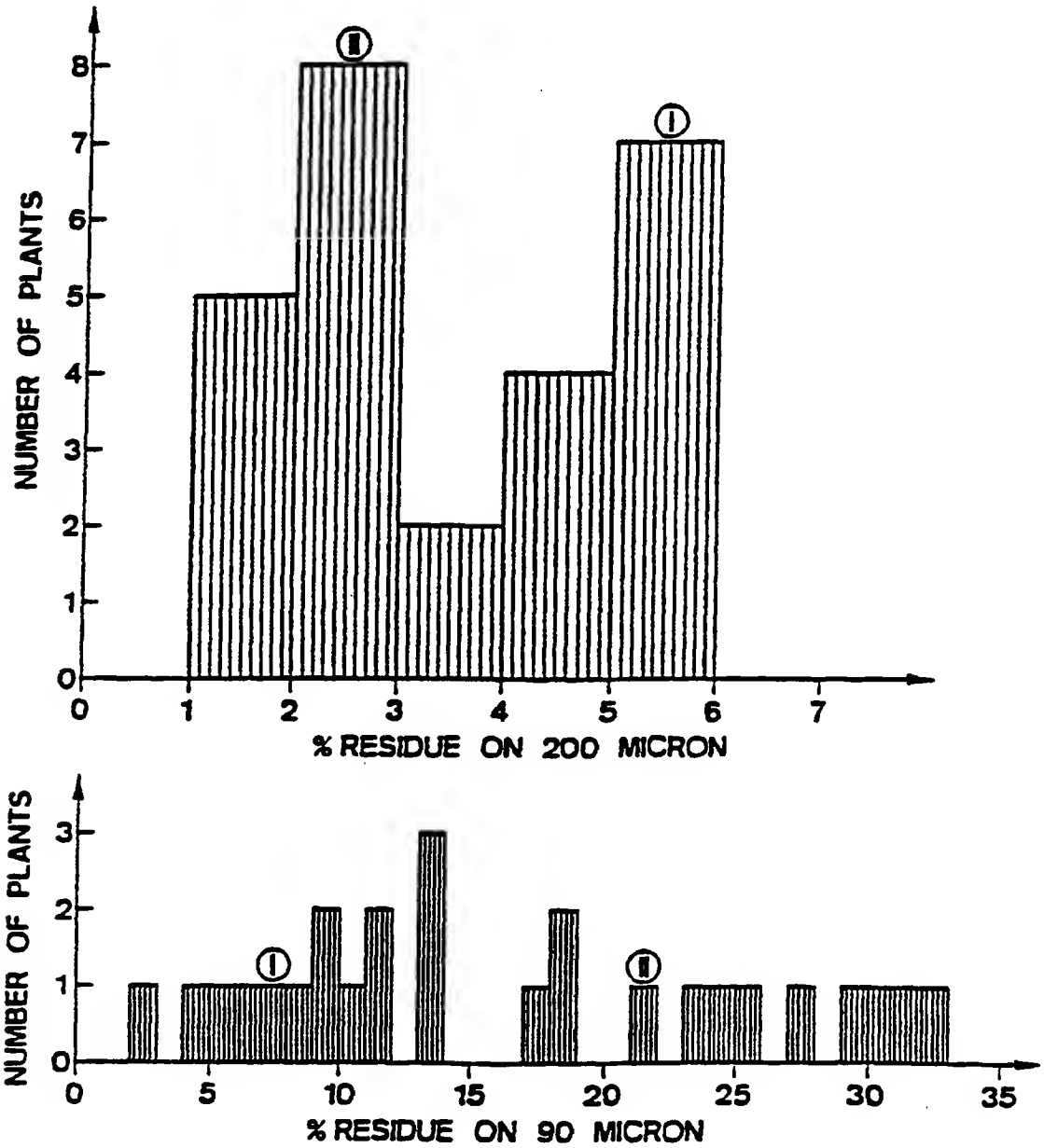
Fig. 4 Slurry Fineness.



The coarser the slurry fineness, the lower the viscosity.

On the average, cement raw slurries have the following fineness characteristics:

mesh sieve	Micron	Wet Process	Dry Process
R 900	200	1 - 6%	0.5 - 2%
R 4900	90	5 - 30%	5 - 25%

C) Water content and Plastic clay

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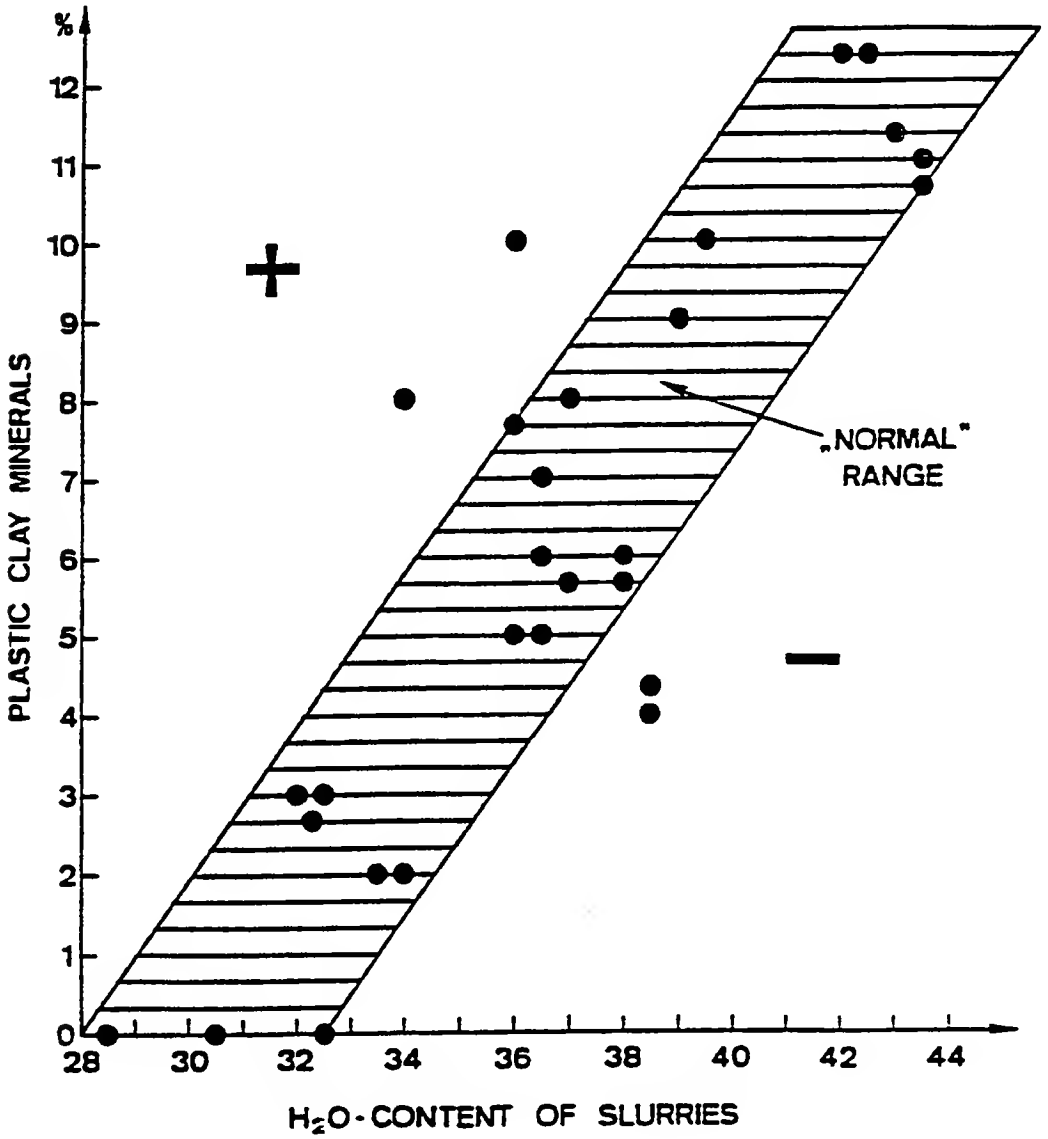
The relationship between slurry water-content and quantity of plastic clay minerals of some 25 cement plants, is displayed in the following figure which indicates the inter-dependence of these two parameters.

Fig. 5 Water content and Plastic clay.



The replacement of clay minerals, for instance with fly ash of thermal power plants, can reduce the water content significant by increased slurry flowability.

Other factors such as degree of fineness and grain size distribution of the raw mix, etc., also influence the slurry water-content, but the plastic clay mineral-content seems to be predominant.

Viscosity Diagrams

Flow Curve

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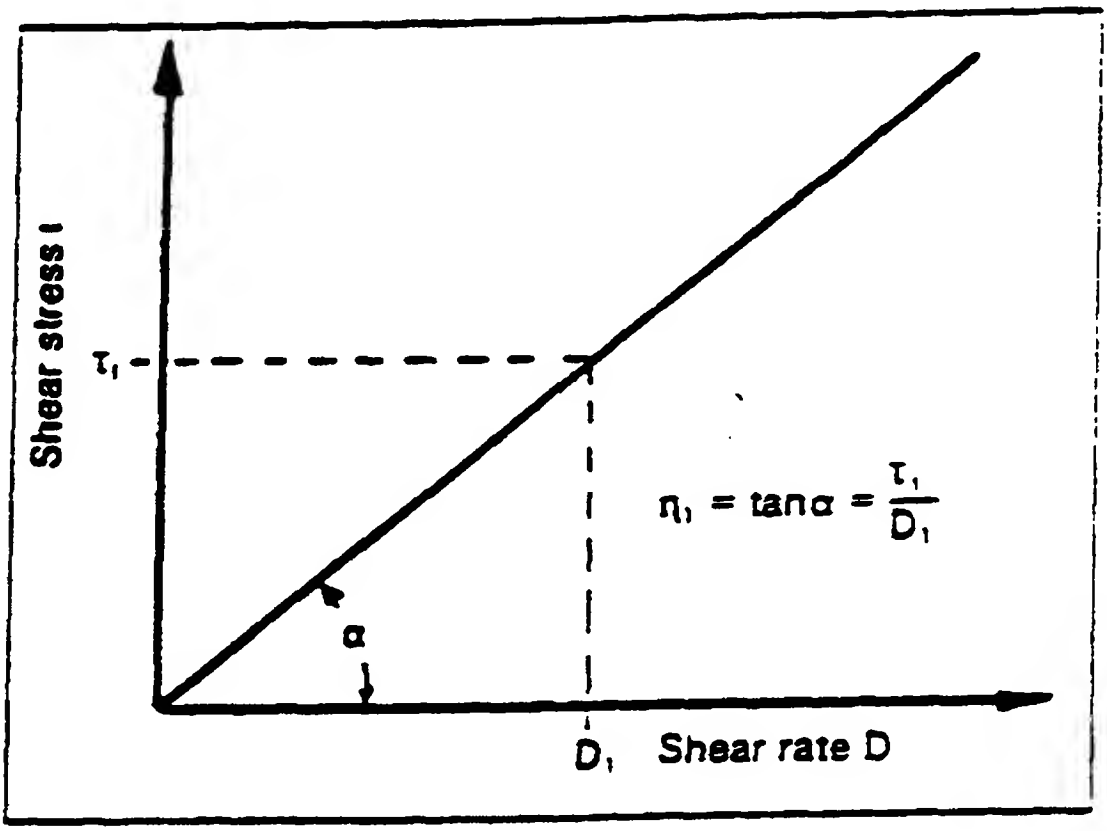
The correlation between shear stress and shear rate defining the flow behaviour of a liquid is graphically displayed in a diagram of T on the ordinate and D on the abscissa. This diagram is called the "Flow Curve".

The most simple type of a flow curve is shown below. The viscosity is assumed to be constant and independent of D.

Fig. 6 Flow Curve.



Viscosity Curve

Another diagram is very common: η is plotted versus D. This diagram is cal-led the " Viscosity Curve":

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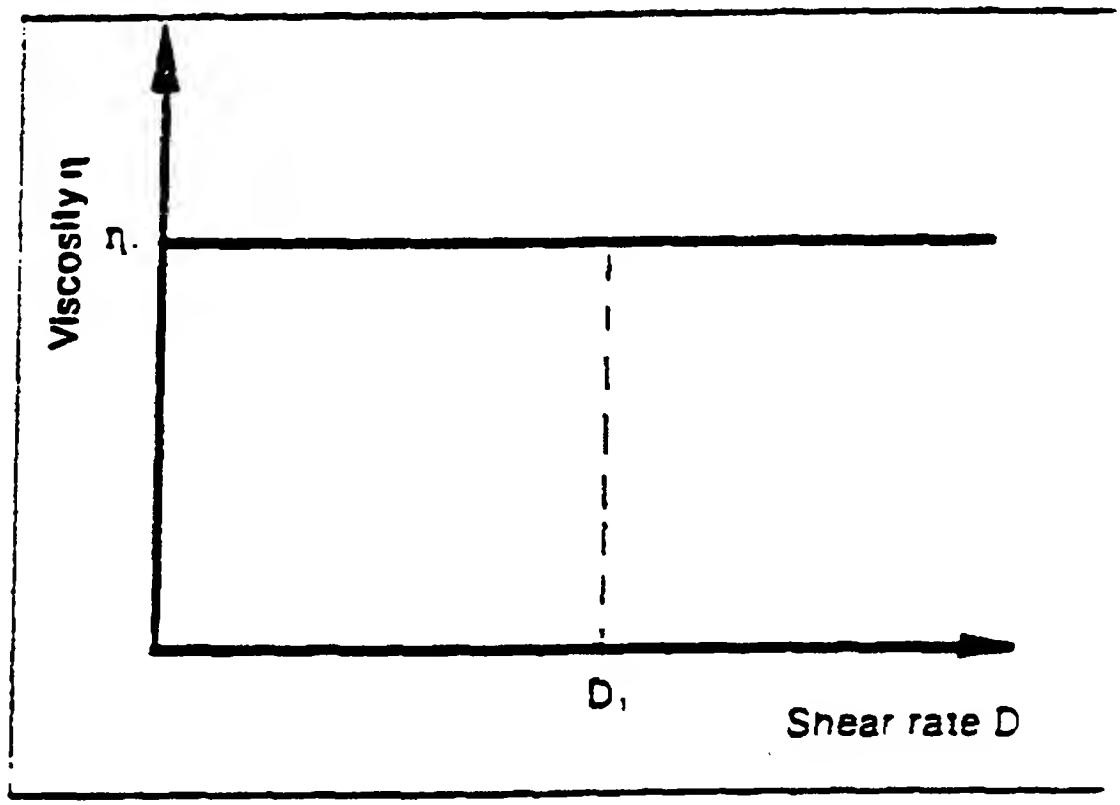
Fig. 7 Viscosity Curve.



Viscosity measurements lead always first to the flow curve. Its results can then be rearranged mathematically to allow plotting the corresponding viscosity curve. The different types of flow curves have their counterparts in types of viscosity curves.

8.1 Types of viscosity

Newtonian liquids

Newton assumed that the graphical equivalent of his equation for an ideal liquid would be a straight line starting at the origin of the flow curve and would climb with a slope of an angle α. Any point on this line defined pairs of values for τ and D. Dividing one by the other gives a value of n. This value can also be defined as the tangent of the angle α.

Because the flow curve for an ideal liquid is straight, the ratio of all pairs of τ and D-values belonging to this line is a constant. This means, that n is not effected by changes of shear rate. All liquids for which this statement is true are called "Newtonian".

Examples: water, mineral oils, bitumen, molasses, etc.

Non-Newtonian liquids

All other liquids not showing this simple "ideal" flow behaviour are called "non-Newtonian". They outnumber the ideal liquids by far.

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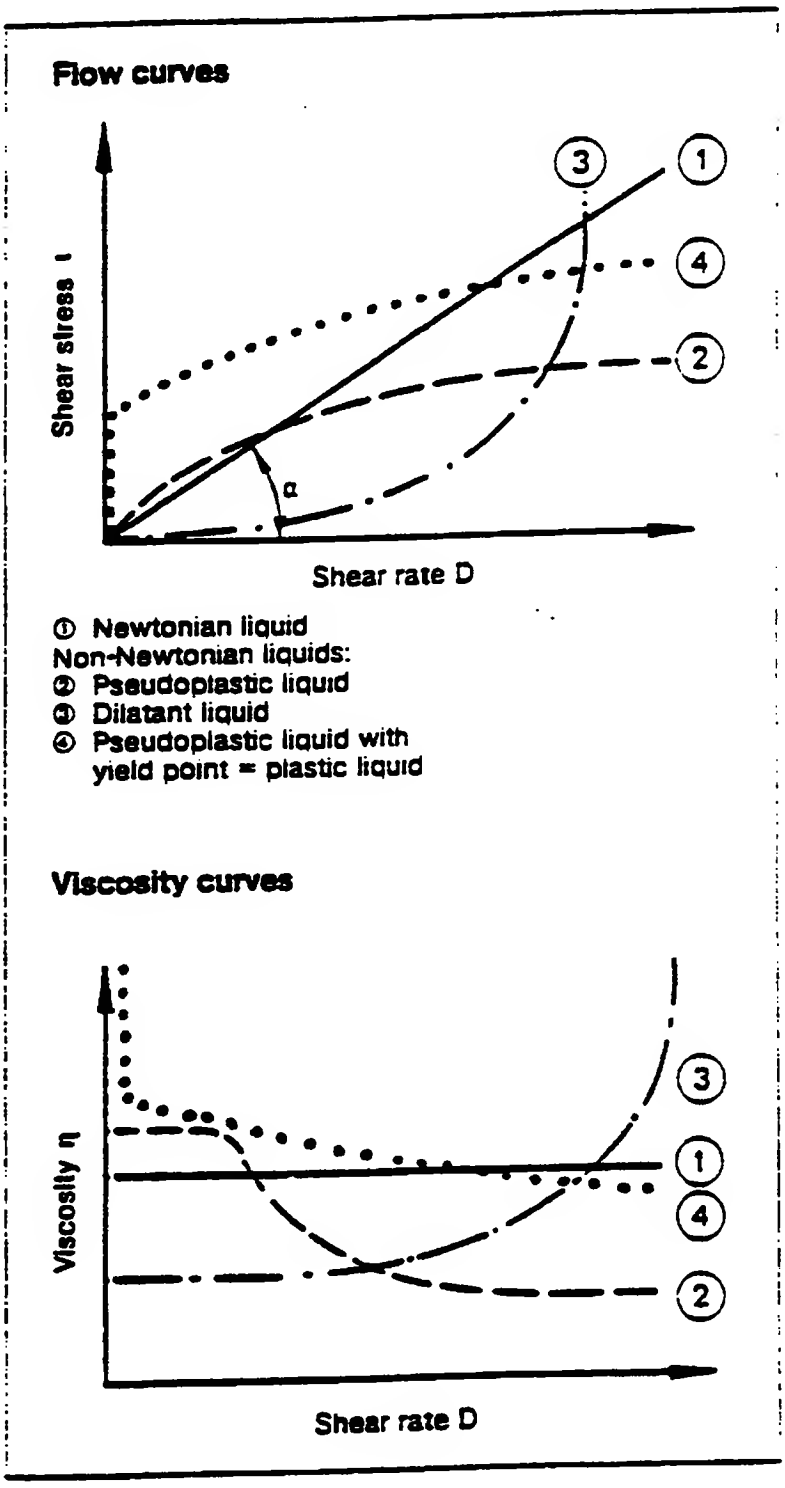
Fig. 8 Non-Newtonian liquids.



A) Pseudoplasticity

Many liquids show drastic viscosity decreases when the shear rates are increased from low to high levels.

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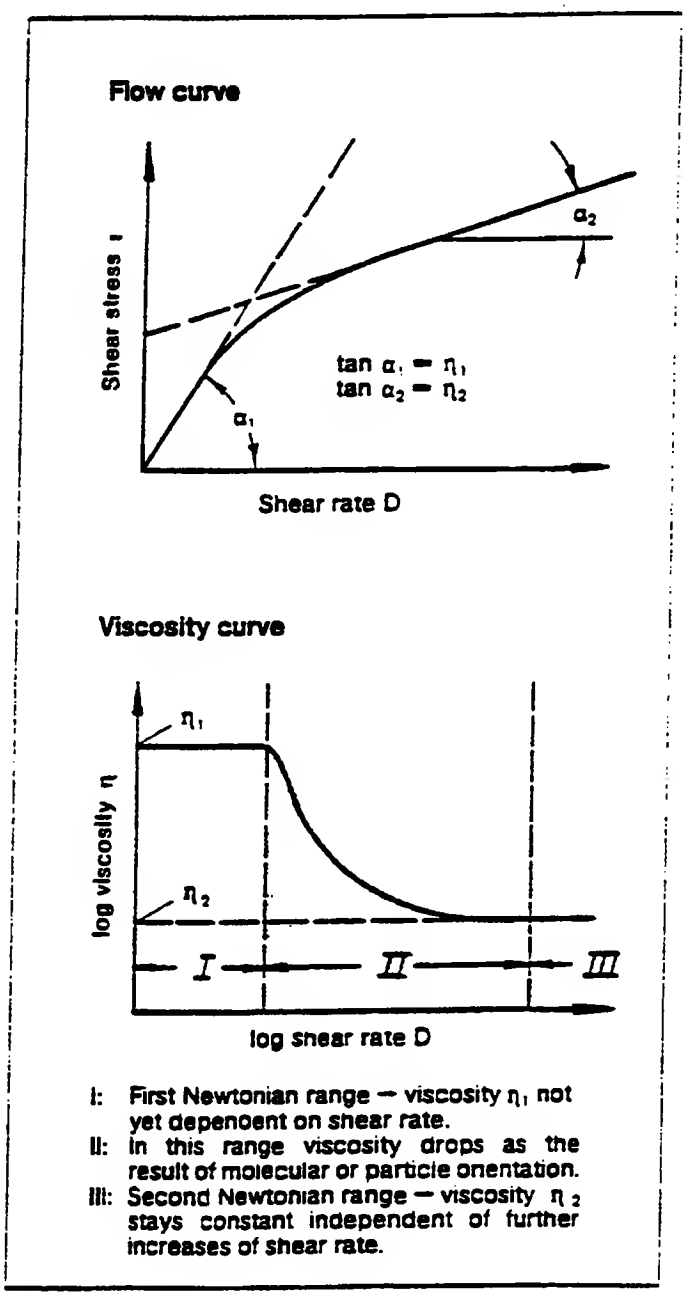
Technically this can mean that for a given force or pressure, more material can be made to flow or the energy can be reduced to sustain a given flow rate. Materials which are thinning due to increasing shear rates are called "pseudoplastic". Substances such as emulsions, suspensions, or dispersions belong to this group.

Shear thinning of pseudoplastic liquids is not uniform over the range of very low to very high shear rates:

Fig. 9 Pseudoplasticity.



At the low shear rate range the Brownian movement of molecules keeps all molecules or particles at random in spite of initial effects of shear orientation. At very low shear rates pseudoplastic liquids behave similarly to Newtonian liquids having a defined viscosity

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independent of shear rate. Then follows a shear rate change when the shear rate induced molecular or particle orientation by far exceeds the randomising effect of the Brownian movement: the viscosity drops drastically. Finally the viscosity approaches a finite level. Going to even higher shear rates cannot cause further shear thinning: The optimum of perfect orientation has already been reached. In the low and high shear rate ranges - called the first and second Newtonian ranges - the viscosity even of non-Newtonian liquids is more or less independent of shear rate.

B) Dilatancy

There is one other type of material characterised by a shear rate dependent viscosity: "dilatant" substances increase their viscosity whenever shear rates increase.

Dilatant flow behaviour is found for example in highly concentrated suspensions in which solid particles are mixed with liquids (sand slurry). The particles are densely packed and the amount of liquid added is just sufficient to fill the voids between the particles. At rest or at low coating speeds the liquid fully lubricates the particles surfaces and thus allows an easy positional change of particles when forces are applied: this suspension behaves as a liquid at low shear rates. At higher shear rates particles will wedge others apart causing a general volume increase. At this stage the liquids share of the total volume decreases. No longer being sufficient to fill all voids and to keep particle surfaces fully lubricated the slurry becomes more viscous.

Dilatancy in liquids is rare.

C) Plasticity

It describes pseudoplastic liquids which additionally feature a yield point.

Plastic liquids can be classified with good reasoning to belong to both liquids and solids. They are mostly dispersions which at rest can build up an intermolecular/interparticle network of binding forces (polar forces, van der Waals forces, etc.). These forces restrict positional change of volume elements and give the substance a solid character with an infinitely high viscosity. Forces acting from outside, if smaller than those forming the network, will deform the shape of this solid substance elastically. Only when the outside forces become so big that they can overcome the network forces - surpass the threshold shear stress calls "yield point" - does the network collapse. Volume elements can now change positions irreversibly: the solid turns into a flowing liquid. Plastic liquids have flow curves which intercept the ordinate not at the origin, but the yield point level of τ_o.

D) Thixotropy

This term describes a rheological phenomenon of great industrial importance in the cement industry. It calls for some explanations in simplified terms of an otherwise often very complex molecular of particle interaction:

In pseudoplastic liquids thinning under the influence of increasing shear depended mainly on the particle/molecular orientation or alignment in the direction of flow. This orientation will again be lost just as fast as orientation came about in the first place.

Plotting a flow curve with a uniformly increasing shear rate - the "up-curve" - one will find that the "down-curve" plotted with uniformly decreasing shear rates will just be superimposed on the "up-curve".

In the flow curve the "up-curve" is no longer directly underneath the "down-curve.

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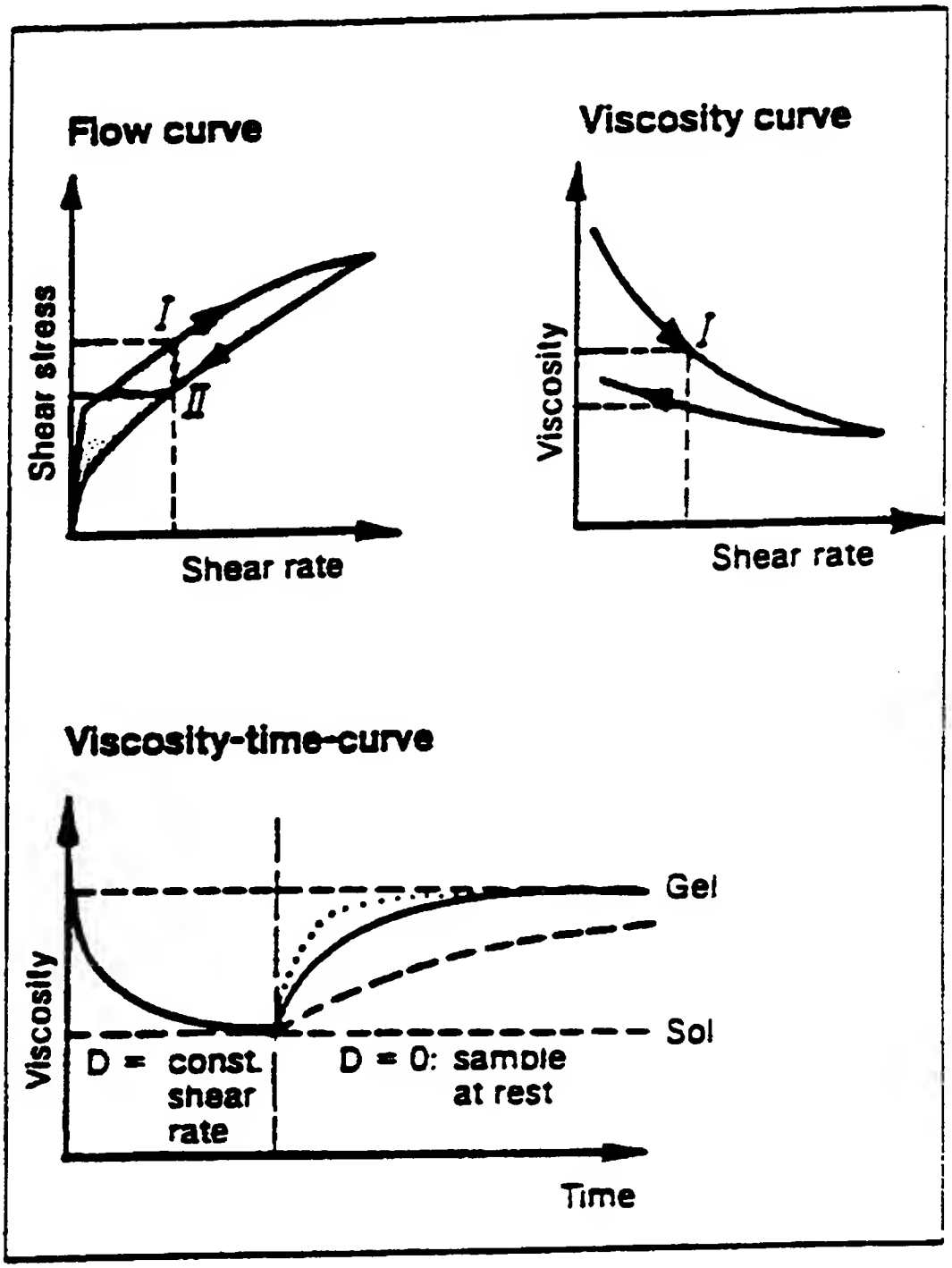
Fig. 10 Thixotropy.



The hysteresis now encountered between these two curves surrounds an area "A" that defines the magnitude of this property called thixotropy. This area has the dimension of "energy" related to the volume of the sample sheared, which indicates that energy is required to break down the thixotropic structure.

Mechanism

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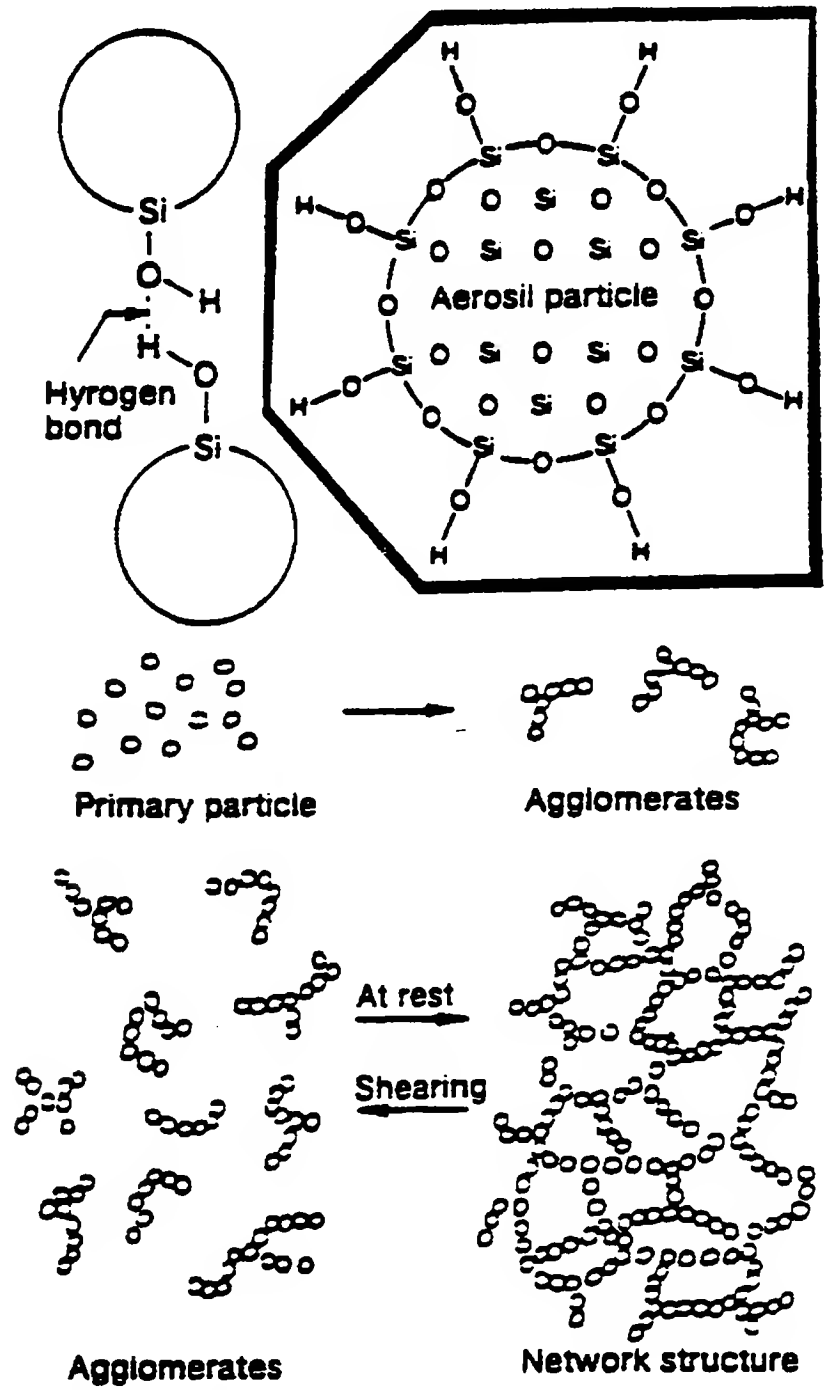
It is typical for many dispersions that they not only show this potential for orientation but additionally for a particle/molecule-inter-action. This will lead to bonds creating a three-dimensional network structure which is often called a "gel". In comparison to the forces within particles or molecules, these bonds - they are often hydrogen or ionic bonds - are relatively weak: they rupture easily, when the dispersions are subjected to shear over an extended period of time.

Fig. 11 Mechanism.



When the network is disrupted the viscosity drops until it asymptotically reaches the lowest possible level for a given constant shear rate. This minimum viscosity level describes the

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"sol"-status of the dispersion. A thixotropic liquid is defined by its potential to have its structure reformed, when the substance is allowed to rest for an extended period of time. The change of a gel to a sol and of a sol to a gel is reproducible any number of times.

E) Rheopexy

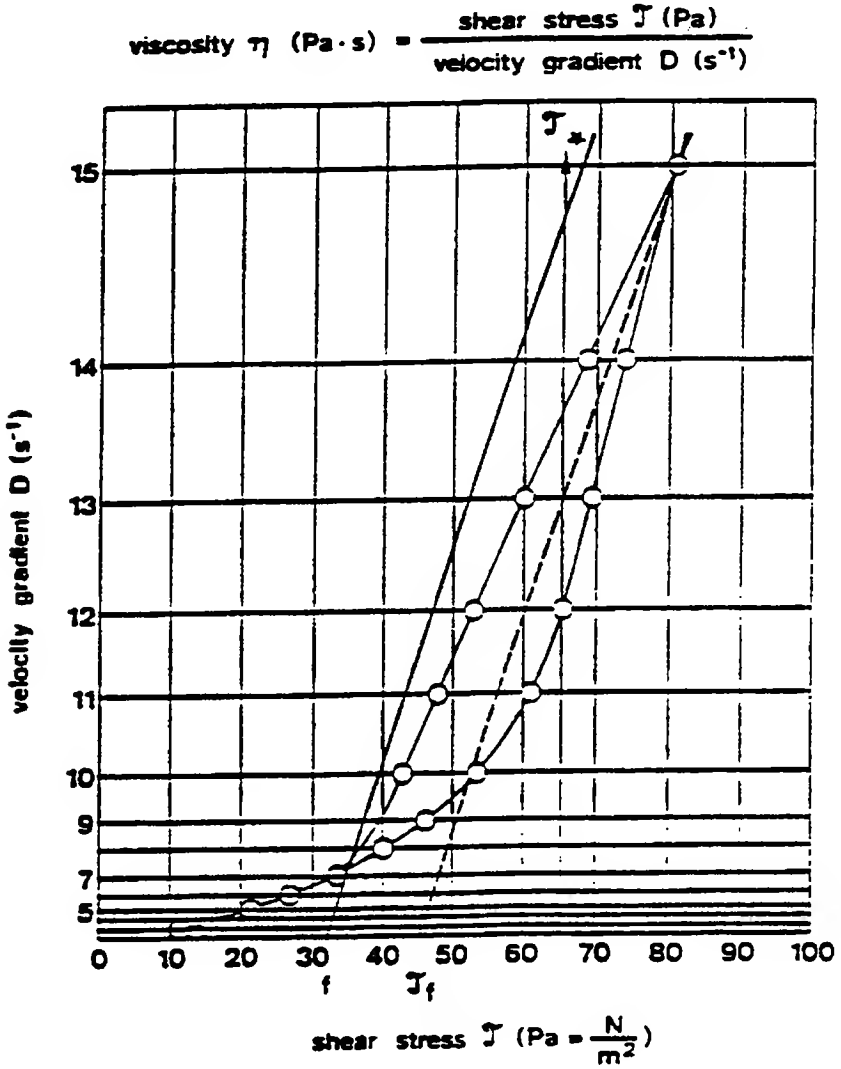
Rheopective liquids are characterised by a viscosity increase related to the duration of shear. When these liquids are allowed to rest they will recover the original - i.e. the low - viscosity level. Rheopexy and thixotropy are opposite flow properties.

Slurry Flow Curve

A typical cement raw slurry flow curve is shown below.

This curve clearly demonstrates the plastic thixotrope characteristics of a typical cement raw slurry.

Fig. 12 Slurry Flow Curve.



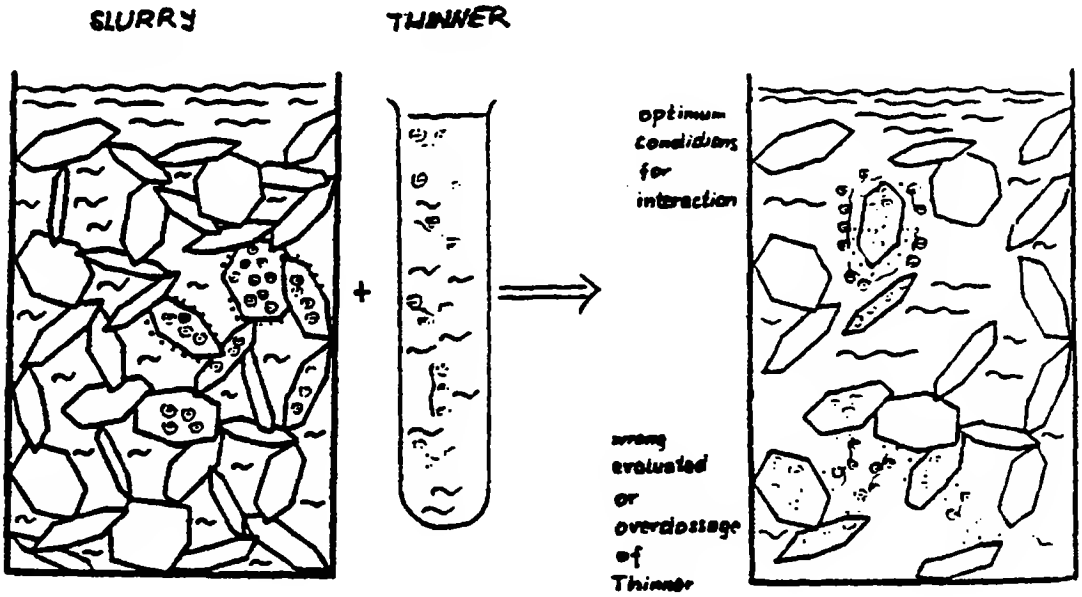
8.2 Slurry Thinners

Very small additions of thinning agents have a marked reducing effect on the water content of a slurry. However, a minimum fluidity and thus a minimum pumpability of the slurry should be maintained, in order to prevent difficulties with thickening of slurry in the chain system.

Normally, the yield values of pumpable raw slurry range between 30 and 60 Pa with a respective slurry water-content of 32 to 38%.

For a better understanding of the mechanism of slurry thinning effect, one of the many interaction models is explained:

Fig. 13 Slurry Thinners.



The slurry particles consists of small flat plates. The areas are occupied with negative electronic change, the corners are positively charged. Stable electrostatic bridges between areas and corners reduces the flowability of the slurry .

Negatively charged thinners (e.g. polyelectrolyts) forms a protective layer around the positively charged particle corners and lowers the slurry viscosity. In case of overdossage or wrong evaluated thinner the opposite effect can occur: the polyelectrolyte forms bridges between the particles and increase the stickiness of the slurry (flocculation).

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Single thinning agents

Sodium salts	Phosphates
hydroxide	sodium or potassium metaphosphate
carbonate	mono-sodium phosphate
monovinyl-sulfonate	all-sodium phosphate
carboxylmethyl-cellulose	tri-sodium phosphate
chloride	sodium hexametaphosphate
oxalate	sodium pyrophosphate
fluoride	polyphosphoric acids
silicate (water glass)	

Vegetable decomposition products	Sulfite lye, lignin-sulfonate
peat and "boggy components"	sulfite lye
peat extract	sulfite alcohol slop
	sulfite spirit slop
	protein
	tanning substances (extracts)
	calcium ligno-sulfonate
	lignin-sulfonate

Surfactants with acidic groups	Organic acid salts
Polyacid salts	citrate
polysilicic acids	tartrate
polymolybdenic acids	
polyborophosphoric acids	

- Carbon dioxide
- Tensides
  - anion-active
  - non-ionogenic
  - cation-active

Combinations of thinning agents

- Phosphates + sodium or calcium salts
  - phosphate + sodium carbonate
  - phosphate + sodium hydroxide
  - phosphate + calcium hydroxide
- Water glass + soluble salts of humic or lignic acid
- Citrate + tartrate
- Carbon dioxide + sodium tripolyphosphate (STPP)
- Triethanolamine + gallic acid

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- Sulfite lye + soda

Experience with Slurry Thinning

From tests on thinning of both industrial and laboratory raw slurries, the following conclusion can be drawn :

- A large number of thinning agents produce the desired effect.
- Not every thinning agent is suited for every type of slurry.
- Under certain conditions, a combination of thinning agents added as a mixture or successively produce a better effect than one single thinning agent.
- The actual effect of a thinning agent for specific conditions cannot be predicted in advance so that tests in laboratory and in plant have to be conducted (synergism-effect, non-linearity).
- The adding quantities vary from case to case.
- The water reduction usually varies in absolute terms between 2 and 7%.
- Slurries with an initial water content between 33 and 43% have been most frequently treated with thinners.
- The mineralogical composition of a raw slurry plays an important role.
- The pH value may be a decisive factor determining the thinning effect.
- Due to local availability and for economical reasons, the most frequently used thinners are soda, water glass, STPP, sulfite lye and lignin-sulfonate.
- The fineness of a raw slurry is of little or no importance as regards the action of the thinners.
- The existing kiln systems usually require no adaptations with regard to chains and filters provided that the water content of the slurry remains above 33 - 35%. However, some plant adaptations are usually necessary, if the water content of thinned industrial raw slurries is low (30 - 33%).
- Under certain conditions, piston pumps should be installed instead of centrifugal pumps and as many bends as possible should be eliminated from the slurry piping, especially if the cement raw slurry shows rheopexic behaviour.
- For each % of water saved. 60 - 100 kJ are economised during the burning process.
- A sequential addition is important for a combination of thinners. First of all, a thinner must be added to adjust the pH and once this effect is reached, a second thinner can be added; afterwards, a third thinning agent can be used if necessary. If all of the thinners are added at the same time and place, generally the effect is not the same as with a sequential addition.

The use of a thinner should be economical, that is, the price of a thinner plays a decisive role (see also chapter "Cost Analysis). It also has to be taken into consideration that specific thinners are, or should be preferred depending on the country or location of the plant (molasses from the sugar industry, sulfite lye from the paper mills).

8.3 Viscosimetry

Laboratory methods

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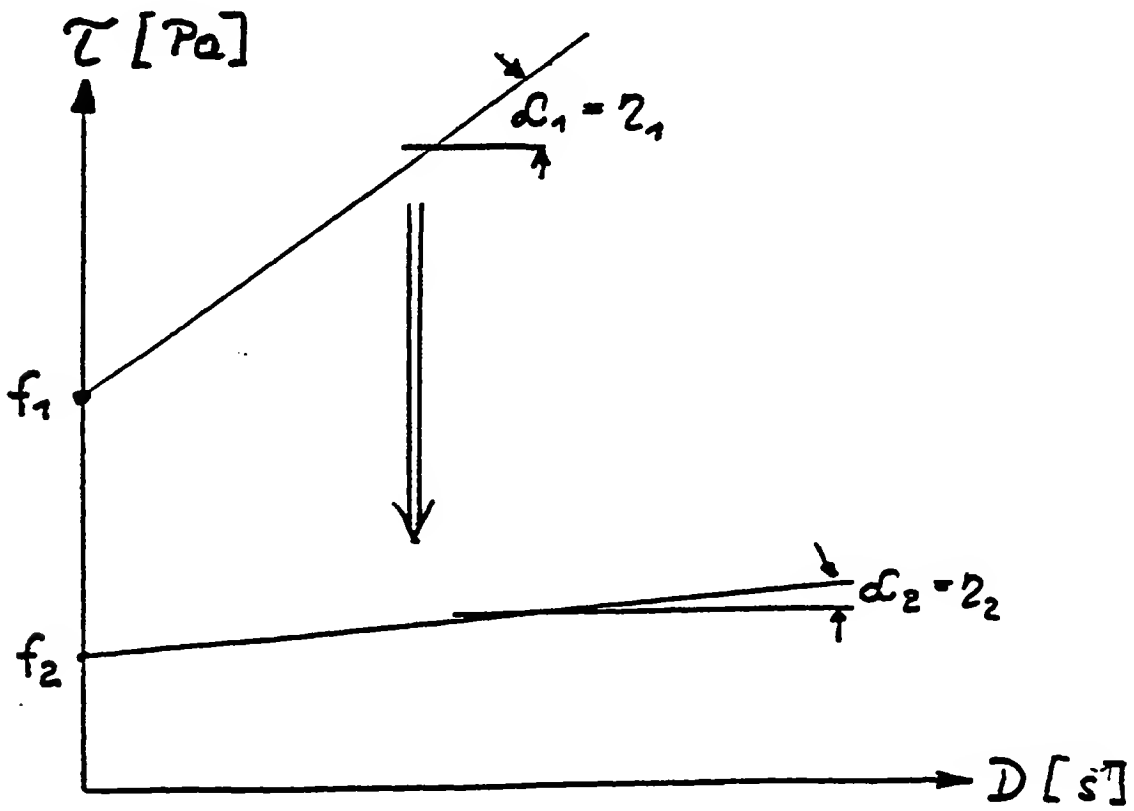
Target of the laboratory investigation is the reduction of the water content and/or the reduction of the yield value (f1 to f2) and the viscosity (71 to 22). A target could also be to decrease the slurry water content and to keep yield value and viscosity constant.

Fig. 14 Laboratory methods.



As mentioned before, reduction of the water content leads to lower thermal and mechanical energy demand, reduction of yield value and viscosity to better pumpability and dewatering property (half wet process).

For the purpose of optimizing the slurry viscosity it is indicated to make trials with different thinner admixtures, concentrations, retention time and temperature levels in the laboratory.

Recent developed method of statistical experimental planning helps to minimise laboratory work drastically.

The main 6 influencing parameters to investigate are
- water content of slurry
- thinner quality
- thinner concentration
- quality of raw material (depending on clinker quality)
- retention time
- slurry temperature

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The target is to find out a quantitative relationship between viscosity and above mentioned parameters. The assumption of 3 experimental level of above 6 influencing parameters means

36 = 729 experiments

This number is, of course, unacceptably high. By application of statistic methods 45 relevant experiments can be evaluated.

After the experiments are done, the quantitative relationship is generated by a regression function. This regression function leads to the optimum parameter constellation and can also be implemented into a PLC ("Model! based Process Control ") .

Rotational Viscosimeter

The instrument of the laboratory investigation is the so-called "Rotational Viscosimeter". State of the art is the automated unit.

The principle of rotational viscometers with coaxial cylinder sensor systems and with cone-and-plate sensor systems allows the design of excel-lent and versatile absolute viscometers. The range of rotational viscometers on the world market varies widely in sophistication and in price. The rheological criteria mentioned before are being used to grade types of instruments and explain design features and resulting areas of application.

One might imagine that the coaxial cylinder sensor systems for rotational viscometers result from bending both flat plates of the Newton's parallel-plate-model into an inner and outer cylinder. A liquid sample filling the annular gap between the two cylinders can be exposed to shear. Conditions will lead to laminar flow and allow a mathematical treatment of the problem.

Two alternatives are open to turn the above geometry into an absolute viscometer:

a) Preselect τ: look for D

   The inner or the outer cylinder is subjected to a defined shear stress or a defined torque while the other cylinder is held at rest. One can now measure the rate of rotation or shear rate resulting.

b) Preselect D: look for τ

c) The inner or the outer cylinder rotates at a defined rotational speed while the other cylinder is held at rest. One can now measure the resulting shear stress or torque. This is the most common design for rotational viscometers.

d) This type of viscometer can come in two versions:
   * Searle system

     The inner cylinder - often called the "rotor" - rotates at a defined speed. The outer cylinder - often called the "cup" is held at rest. The rotating inner cylinder forces the slurry in the annular gap to flow. The resistance of the slurry being sheared between the stationary and rotating boundaries results in a viscosity related torque working on the inner cylinder which counteracts the torque provided by the drive motor. A torque sensing element - normally a spring that twists as the result of the torque - is placed between the drive motor and the shaft of the inner cylinder. The twist of the torque spring is a direct measure of the viscosity of the slurry tested.

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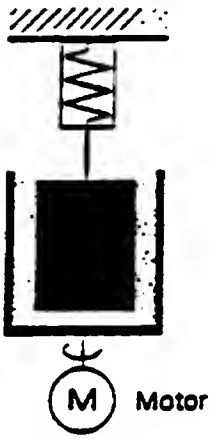
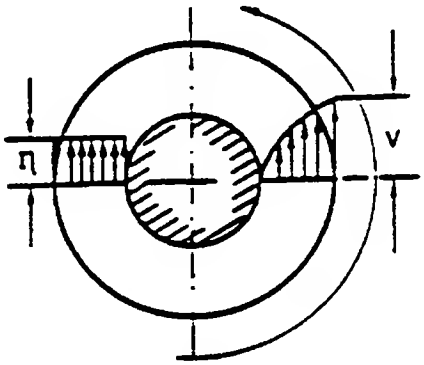
Most viscometer models made world-wide are Searle types. One reason for this tendency is that the good temperature control required for viscosity measurements is more easily accomplished when the outer cylinder does not rotate. Searle type viscometers are limited when low viscous samples have to be tested at high rotor speeds. Centrifugal forces affecting the liquid can turn laminar flow to non-laminar flow and eventually to a fully developed turbulence.

Fig. 15 Searle System.





* Couette system

The outer cylinder rotates at a defined speed. It forces the slurry sample in the annular gap to flow. The resistance of the slurry being sheard, transmits a viscosity related torque onto the inner cylinder, which would induce it also to rotate. This torque is measured by determining just what counteracting torque is required to hold the inner cylinder at stand still. Couette type viscometers are more stable with respect to centrifugal forces.

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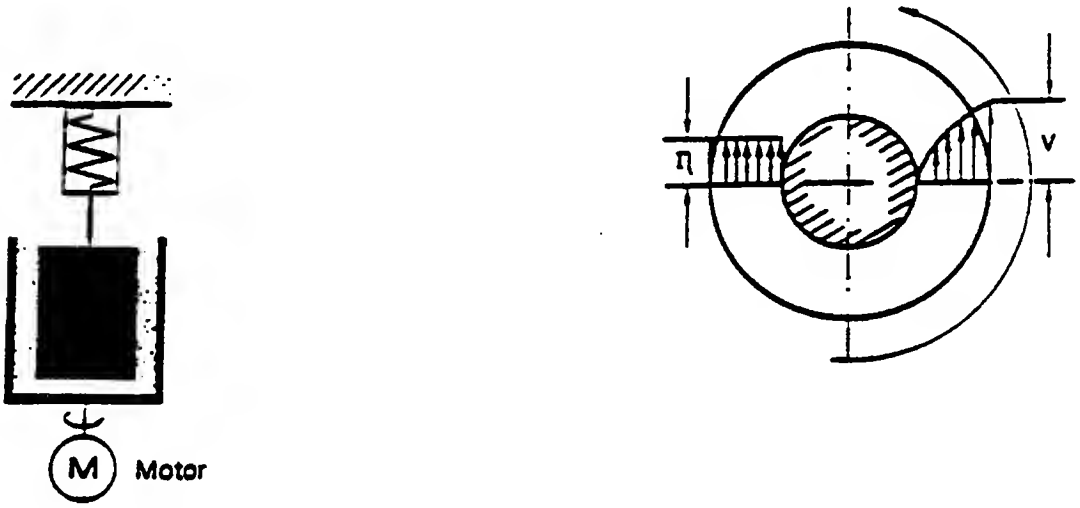
Fig. 16 Couette System.



9. PRODUCTION COST ANALYSIS

Basic is to describe the process in mathematical "Cost Analysis" is to find out the minimum of all specific costs as a function of the raw slurry water content (viscosity).

Approach

1) In a first step a qualitative technical relationship has to be established. This relationship describes the demand on thermal and mechanical energy in function of the water content (viscosity) of the raw slurry.
2) The technical functions are to be evaluated with the specific factor prices, resulting in "Partial Variable Cost Functions".
3) The addition of the "Partial Variable Cost Functions" gives the a Total Variable Cost Function".
4) The minimum of the "Total Variable Cost Function" shows the optimum water content (viscosity).
5) To evaluate new investments it is possible to combine the "Total Variable Cost Function" with a II Fixed Cost Function" to a "Total Cost Function" .

9.1 Technical functions

Several important aspects of slurry preparation are seen in a wet/semi-wet plant. Some of them are:

- Pumping energy demand
- Waste gas ventilation demand
- Pump evaluation
- Filtration energy demand
- Thermal energy (enthalpy) demand

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As examples short, simplified overviews are given. Basic consideration is to show the specific energy demands as a function of the raw slurry water content.

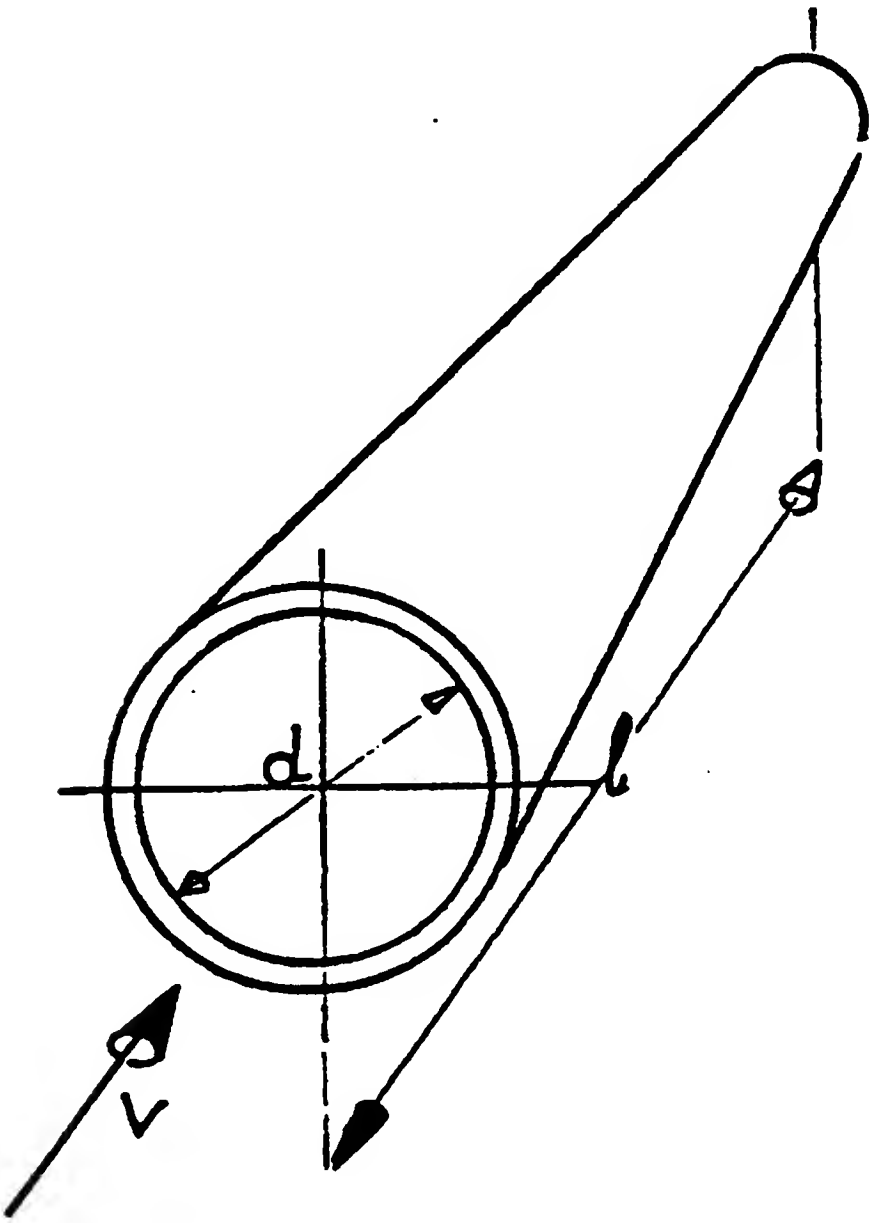
9.1.1 Piping system

There are several parameters known with regard to flow resistivity of a piping system:

- length of piping
- diameter of piping (flow-velocity)
- inner surface quality of piping
- amount and design quality of bends, valves, fittings, ducts

In a simplified form the approach to achieve the flow resistivity function is shown on a simple pipe piece:

Fig. 17 Simple pipe.



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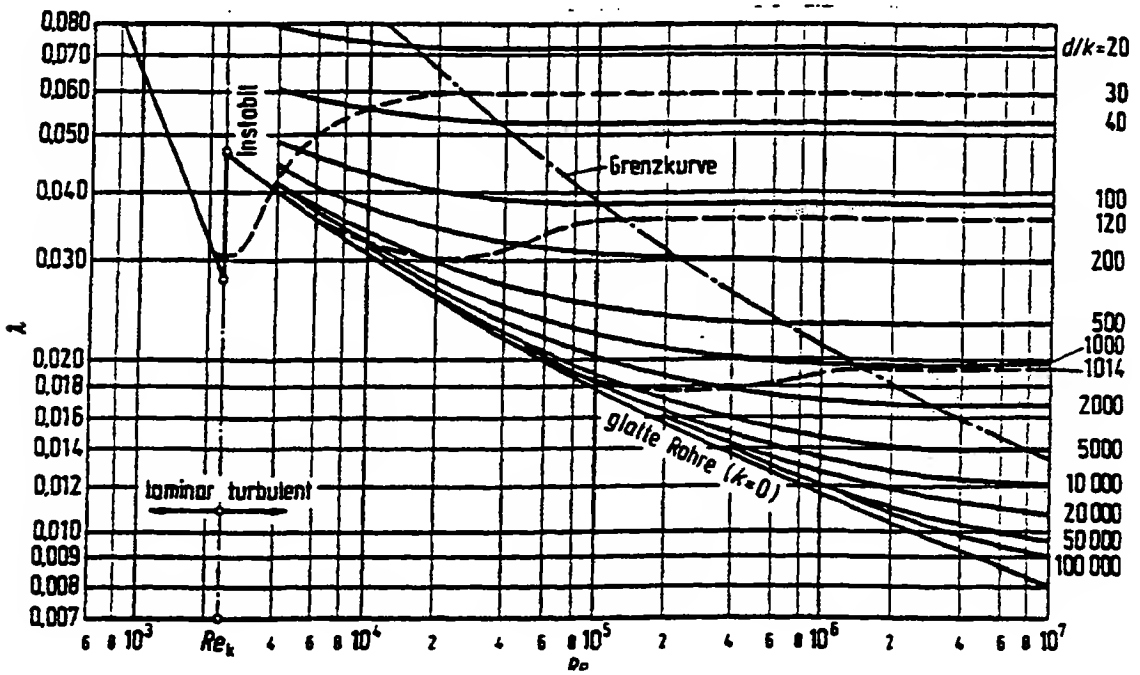
First step is to outnumber the "Reynols Number" Re

$$Re = \frac{v \times d \times \delta}{\eta}$$

Second step is to evaluate the friction number λ as a function of Re a quality of the pipe surface (d/k). The following diagram also shows the interface between laminar and turbulent flow.

Fig. 18 Laminar and Turbulent Flow.



In the laminar flow zone no relation between pipe surface quality and resistivity number exists, whereas the influence in the turbulent flow zone increase drastically.

In the laminar flow zone the simplified approach to evaluate the flow resistivity number is possible by applying the "Hagen Poiseulle's low":

$$\lambda = \frac{64}{Re}$$

With the flow resistivity number it is easily possible to compute the pressure drop of a simplified piping system as follows:

$$\text{Pressure drop} = \lambda \times \frac{1}{d} \times \delta \times \frac{v^2}{2g}$$

Pressure drops results from flow resistivities, elevation differences (geodatical) and velocity difference between input and output of the piping system.

Total pressure drop

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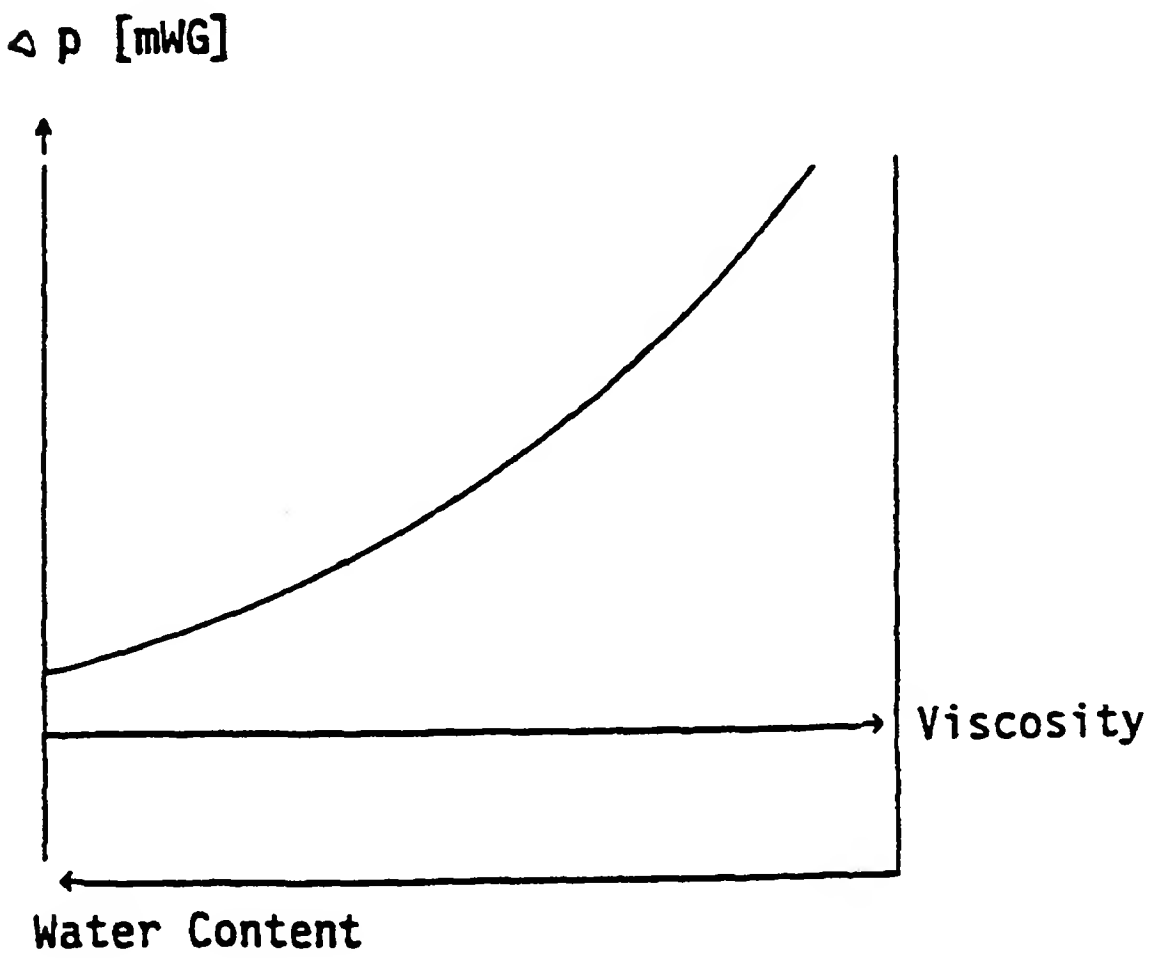
$$\sum \Delta p = \sum \lambda \times \frac{L}{d} \times \delta \times \frac{V^2}{2g} + \sum \xi \times \frac{v^2}{2g} + \frac{V_{(ou)} - V_{(IN)}}{2g} \times \delta$$

With this relationship the function of "Pressure drop" depending on slurry viscosity (water content) can be given quantitatively:

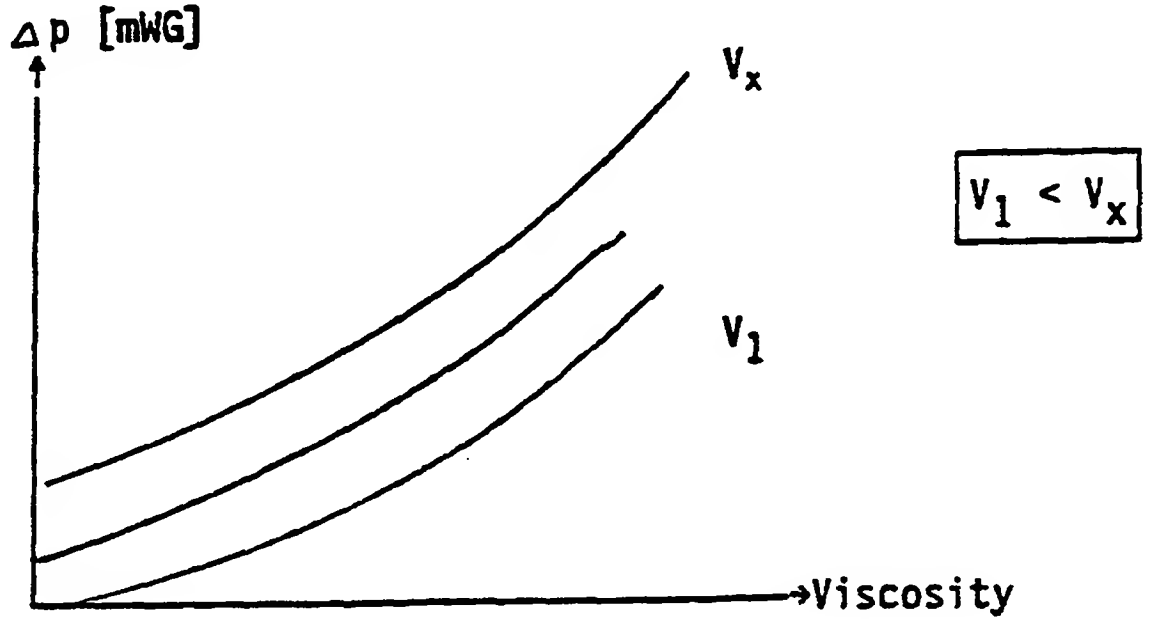
Fig. 19 Pressure Drop.



9.1.2 Pump evaluation

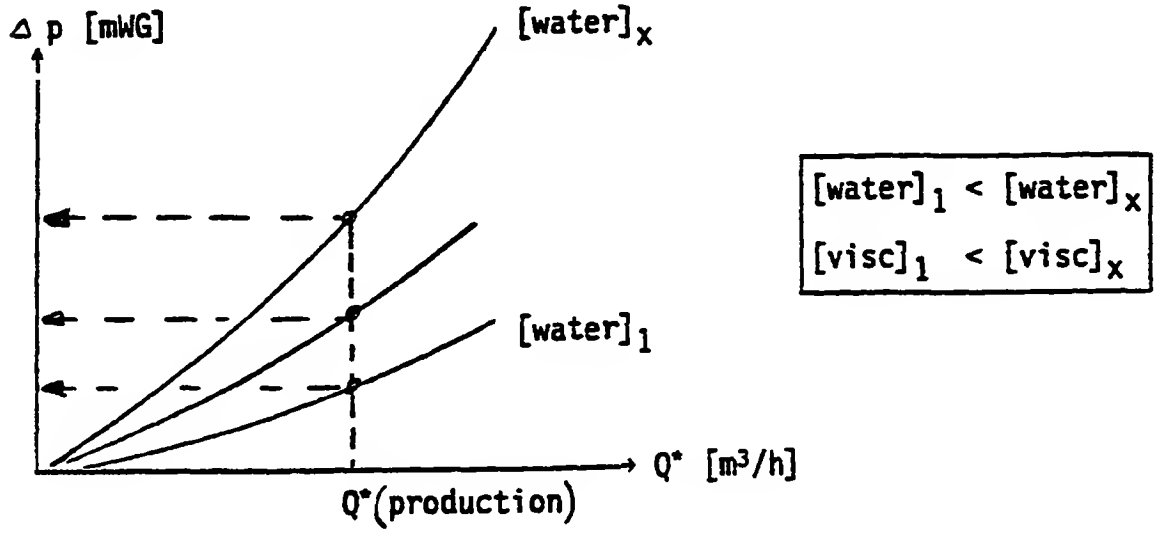
As shown before, the pressure drop of a piping system depends on viscosity (η) and on velocity (v):

Fig. 20 Viscosity against Velocity.



Viscosity and Velocity depends on water content of slurry, so the following relation can be formed:

Fig. 21 Water content.



Curve parameter is Water Content of Cement Raw Slurry

Pump power demand = Δ p x Q* (Production)

Optimisation

Qualitatively the correlation before show the influence of Water Content Viscosity to the Pump power demand. To evaluate the optimum pump, the pump characteristic in relation to the efficiency must be known:

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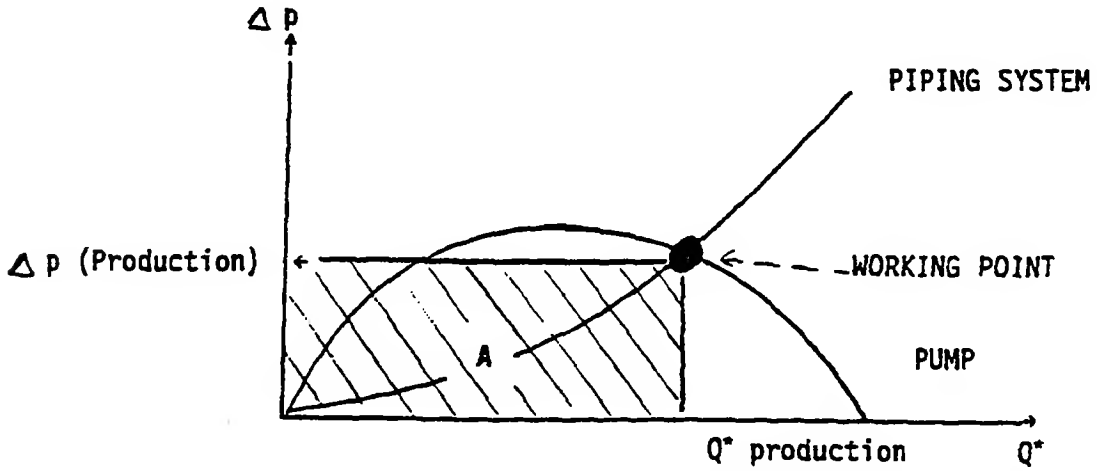
Fig. 22 Optimisation.



Q* production, pump efficiency and pump characteristic dictates the pump evaluation and the power demand. The area "A" represents the pumping power demand, the value has to be minimised.

This consideration leads to a complex, multivariable optimisation. It is worthwhile doing it in face of the great saving potential of investment cost and mechanical energy.

9.1.3 Filtration

Filtration is defined as a "solid - liquid - separation - technique", i.e. separation of a cement raw slurry into a liquid phase (the filtrate) and a solid phase (filter cake) with the aid of a filter which can only be penetrated by liquids. This process is applied in the so called "semi-wet process".

Kozeny - Carman's law

$$Q^* = \frac{1}{\eta} \times k \times \frac{\Delta p}{S^2} \times \frac{E^3}{(1-E)^2} \times \frac{A}{e}$$

k System constant
A Filtration Area
S Specific surface of solids
E Porosity

The equation demonstrates that - in order to achieve a short filtration time - the working pressure (P) should be high (up to 25 bar uncle' industrial conditions), the total filter area large (up to 100 plates per filter press), porosity as high as possible, but the viscosity low.

9.1.4 Thermal Energy Demand

The main reason to decrease the raw slurry water content is to reduce the thermal energy demand of the drying phase prior the clinker burning in the wet kiln. The simple function show the linear relationship between water content and thermal evaporation enthalpy:

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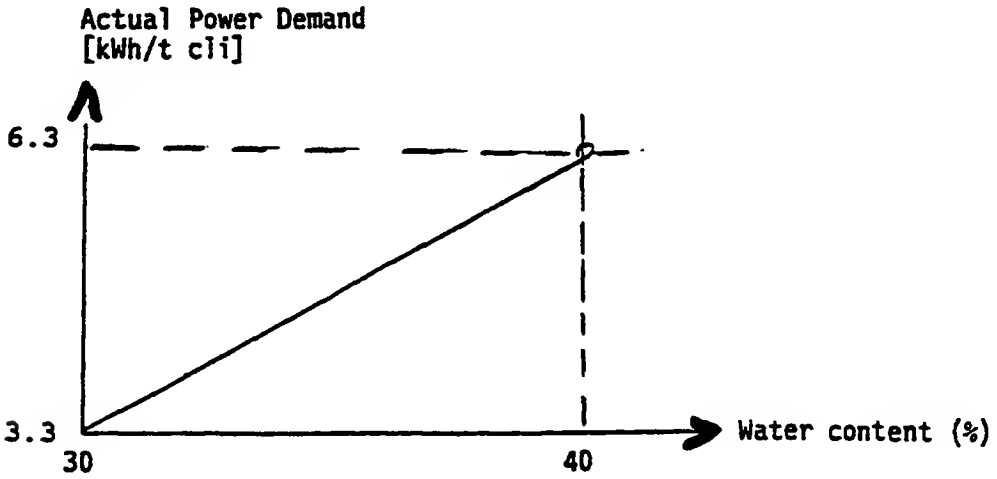
$$H = \frac{\frac{[H_2O] \times 10}{1-[H_2O]}}{100} \times F_{cli} \times 2450 \text{ [kJ / kg cli]}$$

H Specific Evaporation Enthalpy
$[H_2O]$ Water Content of Slurry (%)
f_{cli} Clinker Conversion Factor

9.1.5 Exhaust Fan Energy

The simplified relationship between demand on mechanical exhaust fan energy and water content of raw slurry is shown below. Pressure drop through kiln varies between 100 - 150 mm WG, the kiln is coal fired (25'000 kJ/kg). Fan efficiency = 0.8, no oxygen content of waste gas.

Fig. 23 Exhaust Fan Energy.



9.2 Optimisation

The given technical functions are now to be evaluated by the specific prices in order to get "Cost-functions".

This means the multiplication of specific demands with the specific prices of

- Pumping energy
- Thermal energy
- Exhaust fan energy
- Filtration
- Thinners
- etc.

Qualitatively the "Cost-Function" is shown in relation to the water content of the slurry $[H_2O]$%.

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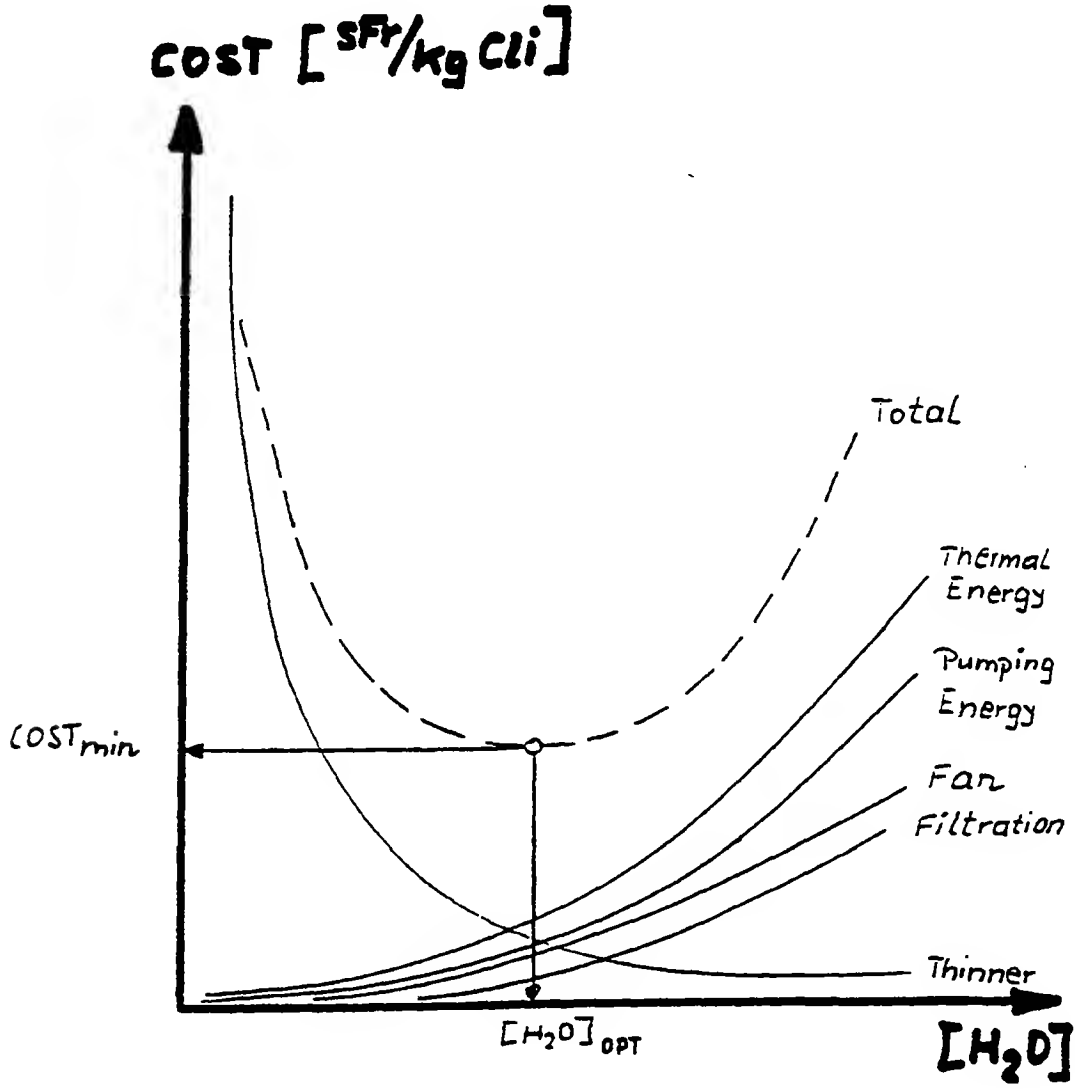
Fig. 24 Cost Function.



The addition of all specific cost functions leads to the "Total Cost Function". This function has the minimum point (optimum operational point) of course on the lowest operation cost. On this point of the function the economical best water content of the slurry is to find.

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Chapter 1

Clinker Burning: Material Technological Aspects

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Clinker Burning - Material Technological Aspects

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1. INTRODUCTION

In the production of Portland cement a mixture of natural minerals is transformed into an intimate mixture of synthetic minerals having the required hydraulic properties.

$$\text{Natural minerals} \xrightarrow[\text{time (t), pressure (p)}]{\text{Temperature (T)}} \text{Synthetichydraulic minerals}$$

This process is similar to that occurring during the transformation of sedimentary and igneous into metamorphic rocks, but with the exception that the time (t) is much shorter and that the pressure (p) is only 1 bar. Consequently, a high temperature is required.

During the transformation into clinker, two principal steps may be differentiated, namely the disintegration of the original rock structures followed by the formation of new ones. Even these categories can be subdivided as follows:

Disintegration of original structure

- mechanical crushing and grinding
- thermal decomposition
- structural rearrangement on heating(e.g. polymorphism)
- melting

Formation of new structures

- occurrence of intermediate products
- genesis and growth of final clinker minerals
- crystallization of liquid phase

1.1 Important Features

Several important features characterize the clinker formation process. These are:

- the clinker burning system is complex and occurs via a series of diverse mechanisms
- the transformation requires mechanical, thermal and electrical energy
- the reaction rate is slow in comparison with homogenous reactions (i.e. gas or liquid phase reactions). This leads to the necessity of high temperatures, extended reaction times, and finely dispersed materials
- the desired hydraulically active clinker minerals are not all stable at normal temperature, and so the hot clinker must be rapidly cooled
- the quality of the product is determined by the

⇒ clinker chemistry

⇒ clinker microstructure

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1.2 Control of the burning process

The transition into clinker is influenced by both material and process technological aspects, namely:

- Material technological aspects
  - raw meal burning behavior including burnability, dust formation, coating behavior, granulation of clinker, etc.
  - The quantity and properties of the liquid phase
- Process technological aspects
  - temperature profile
  - kiln atmosphere
  - fuel type
  - flame characteristics
  - etc.

1.3 Basic Aspects of the Material Changes Involved in Clinker Formation

To properly understand the material changes occurring, it is an advantage to have a good knowledge of the most basic characteristic aspects associated with clinker formation. They are listed in the following table:

Table 1 Important aspects of the understanding of chemical reactions

Aspect	Characteristic Features
Reaction Pathway	indicates the intermediate products occurring between reactants and products
Reaction Mechanism	type(s) and reaction(s) taking place
Reaction Kinetics	indicates rate at which the final products are produced
Reaction Thermodynamics	dictates whether reaction will at all be possible, and what the heat and temperature requirements will be

It is therefore logical, when considering the production of Portland cement clinker, that consideration should be given to these and earlier mentioned points. In the following pages, a review will be given of the principal material aspects associated with clinker formation.

2. REACTION PATHWAYS ENCOUNTERED DURING CLINKER FORMATION

2.1 General Considerations

Cement clinker is an intermediate product in the manufacture of the hydraulic binding agent known as Portland cement. By means of a single thermal treatment, a mixture of the non-hydraulic, naturally occurring minerals calcite, quartz, clays and feldspars, it is transformed into the intimate mixture of hydraulically active minerals alite, belite, aluminate and ferrite, called Portland cement clinker. Like other products such as ceramics, refractories, bricks, etc., cement clinker can be considered as being synthetically produced rock. **The thermal treatment of heating and cooling**, responsible for this "metamorphosis", **is termed "clinkering"** and, with the exception of a small number of shaft kilns, is carried out in one of the several types of rotary kilns. As has already been discussed, only a short specific range of mixtures of calcareous and siliceous minerals can be transformed into Portland cement clinker.

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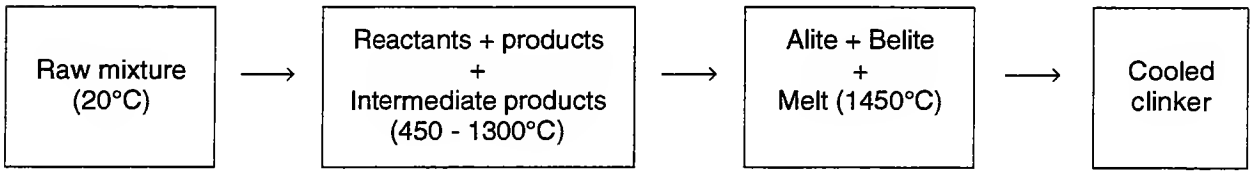
To fully describe the pathway of clinkering, it is necessary to consider the following aspects:

- the chemical and mineralogical content of the raw mix
- the overall sequence of reactions
- the chemical and mineralogical nature of the intermediate products

2.2 Basic Sequence of Reactions

During the heating and cooling of the clinker formation process, a complex series of reactions takes place within the rotary kiln, exhibiting a wide spectrum of reaction mechanisms. Concerning first the reactions involved, the following summarizes the general sequence of events taking place during the formation process.



As can be seen, the raw mixture is not transformed into the final cement clinker by rapid subjection to the required clinkering temperature. The process of clinker formation, as demanded by the construction of the rotary kiln, takes place over a wide range of temperatures, during which a complex mixture of reactants, intermediate and final products often coexist. Even after reaching the maximum temperature (approx. 1450 °C), a certain time must also elapse before a clinker of acceptable quality is produced.

In somewhat more detail, the principal sequence of events occurring and the temperature range over which they take place, can be represented as shown in Table 2:

Table 2 Sequence of reactions occurring in a rotary kiln

Temperature range (°C)	Type of reaction
Heating:	
20 - 100	Evaporation of H_2O
100 - 300	Loss of physically adsorbed water
400 - 900	Removal of structural H_2O (H_2O and OH groups) from clay minerals
>500	Structural changes in silicate minerals
600 - 900	Dissociation of carbonates
>800	Formation of belite, intermediate products, aluminate and ferrite
>1250	Formation of liquid phase (aluminate and ferrite melt)
approx. 1450	Completion of reaction and recrystallization of alite and belite
Cooling	
1300 - 1240	Crystallization of liquid phase into mainly aluminate and ferrite

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From Table 2, several points can be observed, namely:

- that the dissociation of carbonate minerals does not take place at any single temperature but occurs over a temperature range (600 - 900 °C)
- that, with respect to the formation of clinker minerals, three important stages can be distinguished:
- formation of belite, aluminate and ferrite over a wide temperature range starting at approx. 800 °C
- start of appearance of liquid phase and commencement of formation of principal clinker phases (alite) at approx. 1250 °C
- that the aluminate and ferrite found in the final clinker are formed by crystallization of the liquid phase during the cooling cycle. The individual aluminate and ferrite crystals formed during the heating cycle at >800 °C loose their identity by melting (> 1260 °C) to form the liquid phase

2.3 Mineralogical and Chemical Characteristics of Raw Mixes

During a detailed study performed by the Materials Division of the raw meals and slurries from all the plants of the "Holderbank" Group, a mineralogical and chemical analysis of a large number of mixes was undertaken. The following Tables 3 and 4 list the large variety of observed minerals and the range of raw mix chemistry.

Table 3 Minerals frequently found in cement raw mixes

Type	Example
Carbonates	calcite $CaCO_3$, aragonite $CaCO_3$, dolomite $CaCO_3 \cdot MgCO_3$, ankerite $CaCO_3 \cdot (Mg, Fe)\ CO_3$, magnesite $MgCO_3$, siderite $FeCO_3$
Simple Oxides	quartz SiO_2, cristobalite SiO_2, hematite Fe_2O_3, magnetite Fe_3O_4
Feldspars	potassium feldspars (Na, K) Si_3O_8 and plagioclase series (Na, Ca) (Si, Al) $Al_2Si_2O_8$
Sheet silicates*	minerals of the mica and chlorite groups (e.g. biotite, muscovite, chlorite), clay minerals (e.g. kaolinite, montomorillonite, illite, palygorskite) and other sheet silicates (e.g. pyrophyllite) *
Hydroxides*	Al-hydroxides (e.g. boehmite), Fe-hydroxides (e.g. goethite, limonite)
Sulfides and sulfates	pyrites FeS_2, anhydrite $CaSO_4$, gypsum $Ca\ SO_4 \cdot 2H_2O$
Fluorides	fluorspar CaF_2

*see chapter 4 for chemical formulae

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Table 4 **Range of chemical compositions (dry basis) from which OPC clinker is normally produced**

	Parameter	$\bar{x}$	x min.	x max.
Chemistry (wt %)	L.o.I.	35.5	33.8	37.3
	SiO_2	14.4	12.8	16.0
	Al_2O_3	3.2	2.4	4.6
	Fe_2O_3	1.8	1.0	3.8
	CaO	42.4	39.0	44.2
	MgO	1.3	.6	4.0
	SO_3	.37	.08	1.1
	K_2O	.60	.08	1.4
	Na_2O	.17	.04	.58
	TiO_2	.17	.10	.34
	Mn_2O_3	.12	.07	.82
	P_2O_5	.11	.04	.50
	Cl	.03	.01	.19
Ratios	SR	2.9	1.8	3.9
	AR	1.9	0.7	3.2
	LSF	94.0	85.4	103.4
Bogue Mineralogy	C_3S	59	39	84
	C_2S	19	0	41
	C_3A	8	7	13
	C_4AF	9	5	19

From the preceding data it is evident that the various mixes are composed of very different minerals and in addition vary within a certain range in their chemical composition. In spite of these differences, Portland cement clinkers composed of the same hydraulically active minerals are produced from all these mixes. The fact that such a diversity of mixes results in the same end product, underlines the complexity of the transformation process.

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2.4 Intermediate Products

The clinkering reaction does not take place in a single direct step from the natural to the final minerals, but proceeds via intermediate products. These intermediate products can be observed by heating up raw mixes in the laboratory or by withdrawing samples from various points along the kiln. This latter method, although more difficult to carry out, has the advantage over the laboratory method in that those intermediate minerals, whose presence is also dependent on the kiln gas atmosphere, are observed. The following table lists the most commonly encountered intermediate minerals.

Table 5 Intermediate minerals encountered during clinker production

Type	Mineral Name	Chemical Composition
Simple Sulfates	anhydrite	$CaSO_4$
	arcanite	K_2SO_4
Compound Sulfates	"sulfate"-spurrite	$2(C_2S) \cdot CaSO_4$
	"calcium"-langbeinite	$K_2Ca_2(SO_4)_3$
Compound Carbonates	spurrite	$2(C_2S) \cdot CaCO_3$
Simple Chlorides	sylvite	KCl
Calcium Aluminates	mayenite	$12\ CaO \cdot 7\ Al_2O_3$
		$CaO \cdot Al_2O_3$
Calcium Ferrites		$2\ CaO \cdot Fe_2O_3$
Calcium Alumino-Silicates	gehlenite	$2\ CaO \cdot Al_2O_3 \cdot SiO_2$

That these materials are formed at all, when it is considered that the final equilibrium products are alite, belite, aluminate, and ferrite, can be explained as follows:

Table 6 Reasons for the formation of the intermediate products

- the intermediate products are preferentially formed by kinetically faster reaction rates
- the intermediate products are the reaction products of localized zones in the meal charge, i.e. local equilibrium but not overall equilibrium was reached (for example gehlenite formation)
- the intermediate products are really the equilibrium products at the given temperature and gas atmosphere, but not at the final clinkering temperature (the formation of spurrite is a typical example)

The most important intermediate product, however, is not a crystalline product at all, but is the liquid phase first formed above ca. 1250 °C, and which only forms crystalline products (mainly aluminate and ferrite) after cooling.

2.5 Liquid Phase

The composition of the raw mix determines

- temperature at which liquid will first be formed
- amount of liquid formed at any given temperature
- the physical properties of the liquid at any particular temperature, especially its viscosity

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2.5.1 Temperature of Liquid Formation

The following are the lowest melting temperatures of various mixtures pertinent to Portland cement production:

Components	Temperature °C of liquid formation
C - S	2065
C - S - A	1458
C - S - A - F	1338
C - S - A - F - M	1300
C - S - A - F - M - N	1280

Due to the inclusion of additional oxides (e.g. K_2O, TiO_2, Mn_2O_3, etc.) the lowest formation temperature of the liquid phase in clinker is in reality around 1250°C.

2.5.2 Quantity of Liquid

Although most raw mixes show about the same minimum temperature of liquid formation, the quantity of liquid formed at this and progressively higher temperatures varies according to the raw mix chemistry.

In the Portland cement relevant parts of the system C - S - A - F, in which melting begins at 1338 °C, the composition of the liquid is:

CaO	-	55 %	
SiO_2	-	6 %	Alumina ratio
Al_2O_3	-	22 %	(AR) = 1.38
Fe_2O_3	-	16 %	

The percentage liquid forming at various temperature (1338, 1400, and 1450 °C) is given in Table 7.

Table 7 Formulae used for estimating the quantity of the liquid phase according to Lea & Parker

Temperature °C	Quantity of liquid phase
1338	**AR > 1.38:** 6.1 Fe_2O_3 + MgO + Na_2O + K_2O **AR < 1.38:** 8.5 Al_2O_3 - 5.22 Fe_2O_3 + MgO + Na_2O + K_2O
1400	2.95 Al_2O_3 + 2.2 Fe_2O_3 + MgO + Na_2O + K_2O
1450	3.0 Al_2O_3 + 2.25 Fe_2O_3 + MgO + Na_2O + K_2O

Because MgO has limited solubility in the liquid, there is a restriction of 2 % maximum MgO which can be incorporated into these equations.

For the majority of clinkers, the quantity of liquid formed at 1450 °C varies according to the above formula from approx. 20 to 30 %. Figure 1 shows the quantity of liquid formed at the given temperature for several Group plant clinker, the influence of MgO and alkalis being included.

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Figure 1 **Quantitative change of liquid phase with temperature (INFLUENCE OF MgO, Na_2O AND K_2O INCLUDED)**

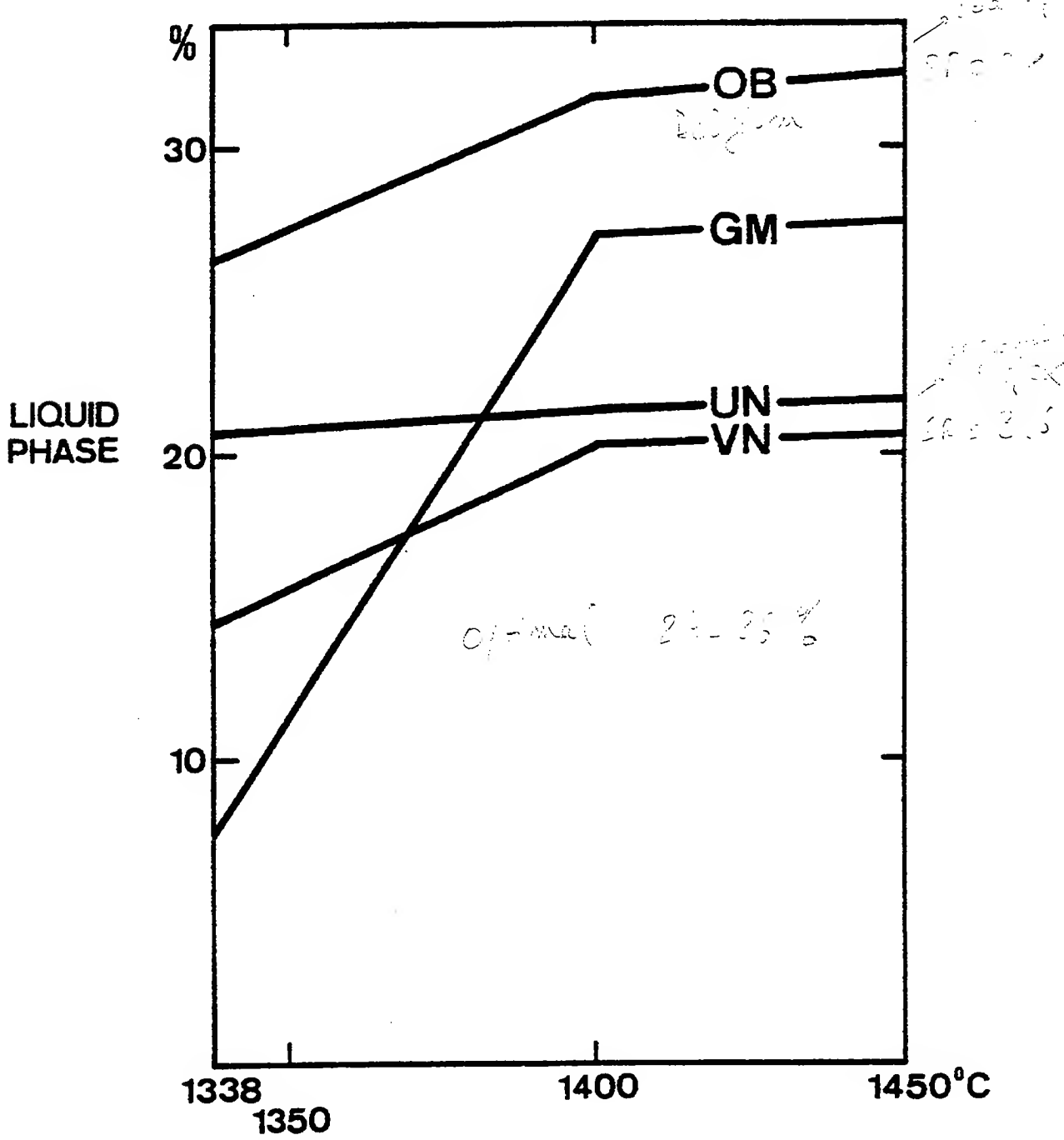

Figure 2 displays graphically the influence of Al_2O_3 and Fe_2O_3 alone on the quantity of liquid formed at **1338 °C**, the solid contours representing the percentage of liquid formed. It can be seen that the effect of adding Fe_2O_3 or Al_2O_3 on the proportion of liquid formed at 1338 °C is influenced strongly by the alumina ratio. If in a clinker composition containing 4.5 % Al_2O_3 and 2 % Fe_2O_3, for example, the Fe_2O_3 content is held constant and the Al_2O_3 content is increased, the percentage of liquid remains unchanged at 14 %, because the iso-Fe_2O_3 contour lies parallel to the iso-liquid contour in this region. If, however, the iron content is increased and the Al_2O_3 content remains constant, the proportion of liquid at first increases rapidly until at 3.3 % Fe_2O_3 it attains a maximum of 21 %. With further increase in Fe_2O_3 the

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quantity of liquid decreases. Therefore, the most effective use of Al_2O_3 and Fe_2O_3 - with respect to liquid formation at 1338 °C - occurs when the two are used in the weight ratio of 1.38.

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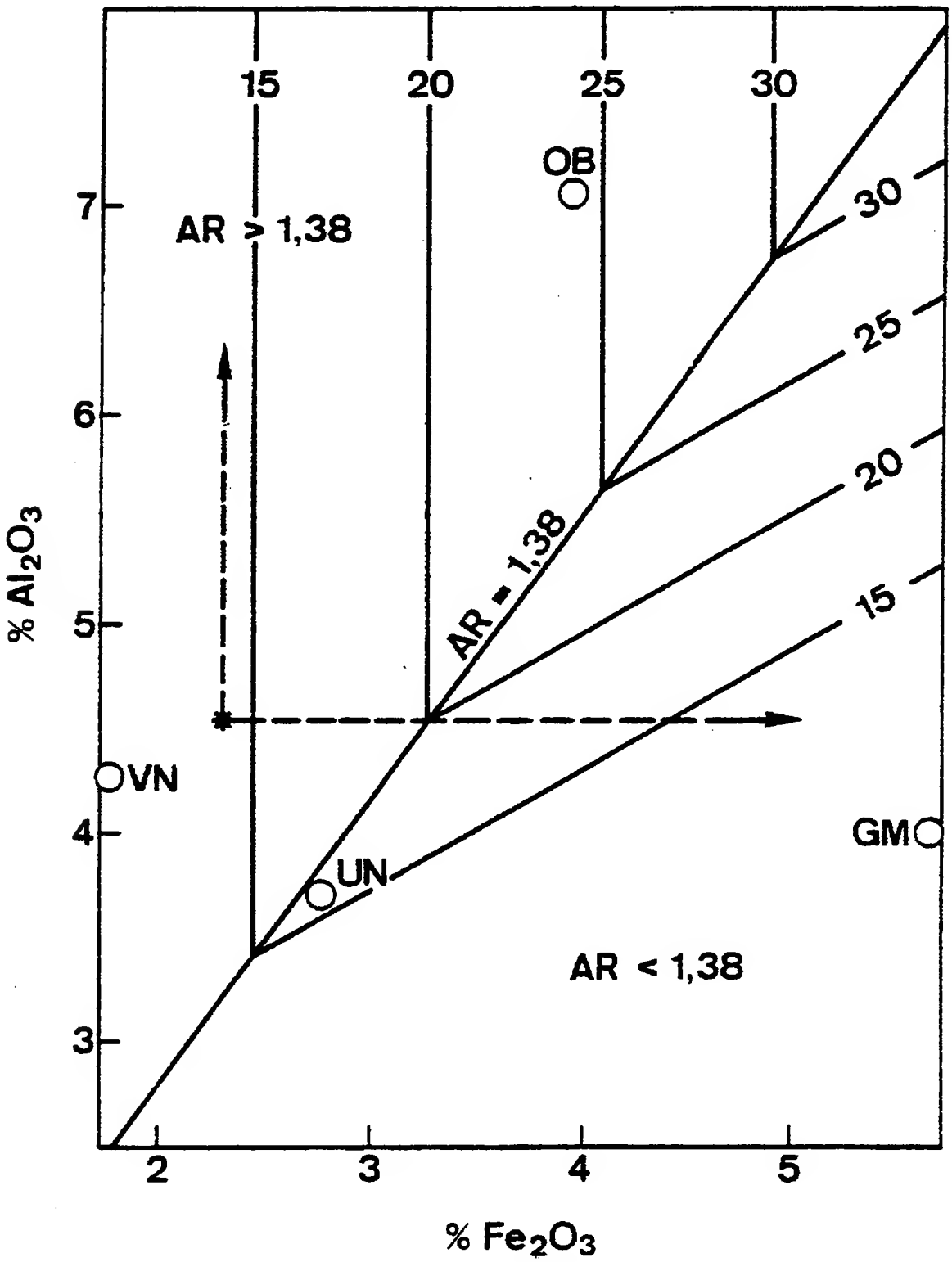
Figure 2 Quantity of liquid phase at 1338 °C



AR = Alumina ratio

OB, GM, UN, VN = Different Plants

At **1400 °C and above** increasing Al_2O_3 and / or Fe_2O_3 always result in a larger quantity of liquid phase, Al_2O_3 being the one with the larger influence.

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2.5.3 Viscosity of Liquid Phase

The viscosity of the liquid phase diminishes exponentially with increasing temperature and at 1400 °C is reduced by addition of fluxing components in the following order:

$$Na_2O < CaO < MgO < Fe_2O_3 < MnO$$

With increasing SiO_2 content of the melt and to a lesser extent with increasing Al_2O_3, appreciable increases in viscosity occur.

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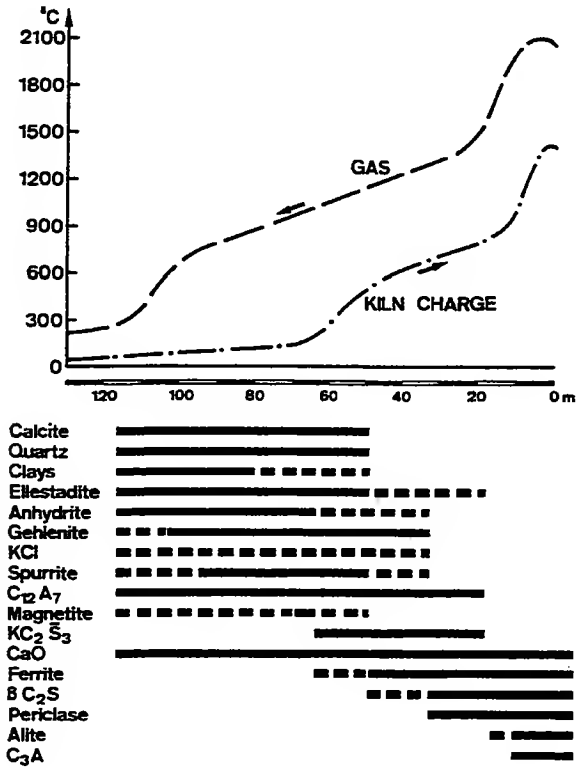
2.6 The Overall Reaction Sequence

Several different approaches have been used in the past to display the sequence of events taking place during the formation of cement clinker. These differ mainly in the aspect chosen, the following examples giving emphasis to differing features.

2.6.1 Display Based on Qualitative Change of Minerals from Samples taken from an Operational Kiln

The minerals existing at various points during the production process in the kiln can be seen in Figure 3. It is worth noting that the feed contains several minerals (e.g. KCl, spurrite, $C_{12}A_7$, CaO) that were not present in the freshly milled raw mix. This is due to the return of dust to the slurry feed.

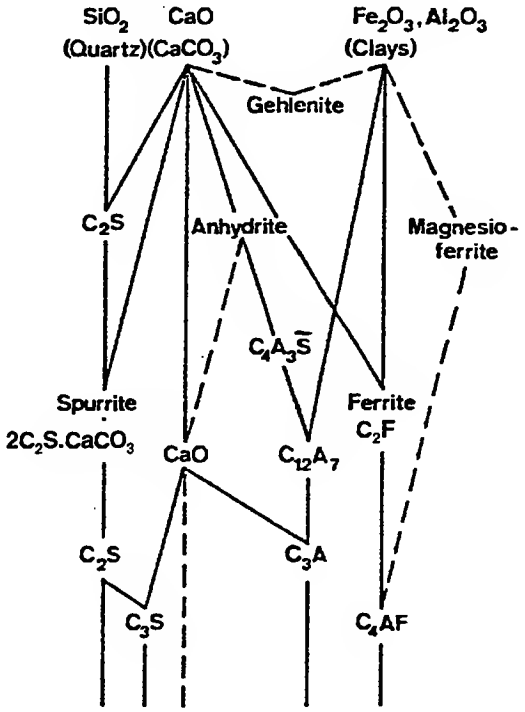
Figure 3 Minerals identified at different locations (long wet kiln)



2.6.2 Schematic Display according to Chemistry

A recent example of this approach to clinker formation is proposed in Figure 4.

Figure 4 Sequence of compound formation according to chemical composition



In order to allow an overall picture to be quickly gained, it must be mentioned that the SiO_2 content of the clay minerals and the pressure of feldspars have been omitted. Of interest in the above display is that the presence of spurrite is considered as being a normal intermediate product and not only a component found solely in kiln deposits.

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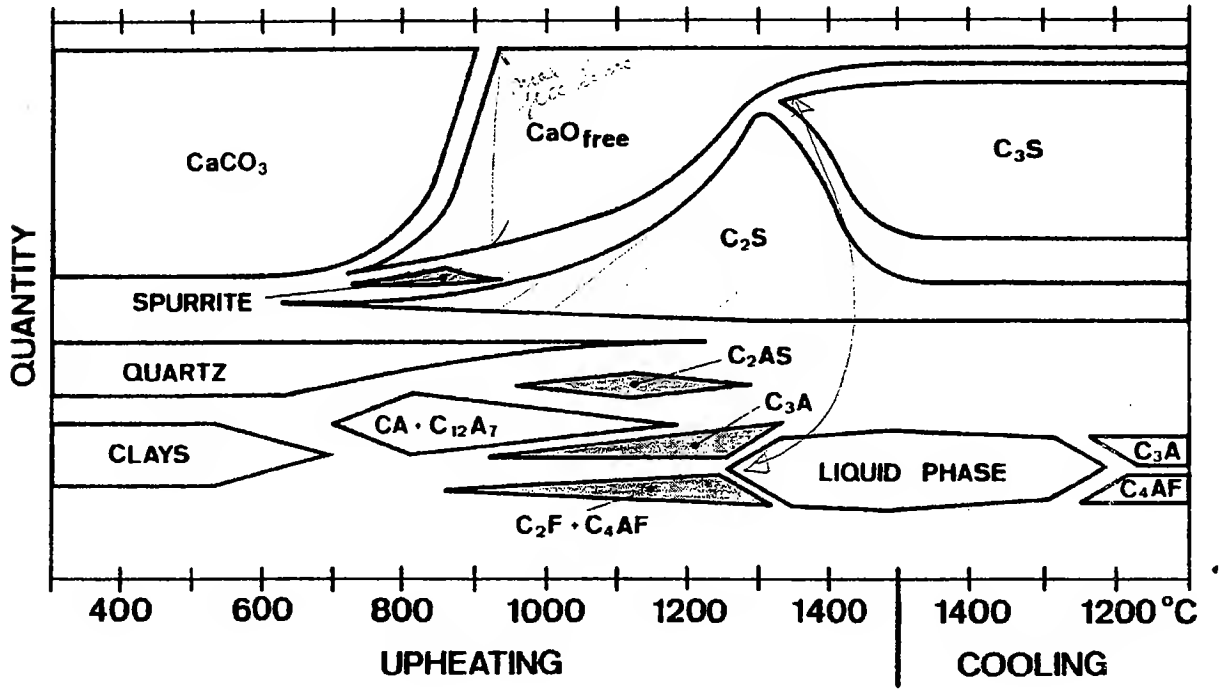
2.6.3 Quasi-Quantitative Display of Minerals Present at Various Temperatures

An informative schematic display based mainly on laboratory studies depicts in a quasi-quantitative manner changes of the various minerals with temperature. This is shown in Figure 5

Figure 5 Quasi-quantitative variation of minerals with temperature



3. REACTION MECHANISMS

3.1 Definitions

For the discussion of reaction mechanisms the following concepts with regard to states of matter and types of reaction will be employed.

3.1.1 States of Matter

All matter can be assigned at any specific temperature to at least one of the three defined states of matter, namely to the solid, liquid, or gaseous state. Each of these states has specific properties which allow a simple distinction between them to be made as follows:

State of Matter	Properties
solid	definitive volume and definite shape
liquid	definitive volume, assumes shape of container
gaseous	neither definitive volume nor definite shape

Under specific conditions (of temperature and pressure) it is possible for more than one state to coexist with another. An example demonstrating this effect is, of course, water which at various temperatures and pressures can coexist in the liquid/gas, solid/liquid and solid/liquid/gas states.

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3.1.2 Classification of Reactions

Chemical reactions may be classified according to their type, namely:

Type	Example from clinker burning
structural change	high quartz ↔ low quartz
decomposition	$CaCO_3 \leftrightarrow CaO + CO_2$
combination	$2CaO + SiO_2 \leftrightarrow C_2S$

A further differentiation of these reaction types may be made:

- According to the state of matter

reaction type	Example from clinker burning
solid - solid	quartz and free CaO → belite
solid - liquid	liquid phase → crystallization of aluminate + ferrite
solid - gas	$CaCO_3 \leftrightarrow CaO + CO_2$
liquid - liquid	-
liquid - gas	drying process, volatilization of alkalis, etc.
gas - gas	$CO + 1/2\ O_2 \leftrightarrow CO_2$

- according to rate controlling step (kinetics of reaction)

Type	Example from clinker burning
diffusion	formation of alite
phase boundary	quartz + free CaO → belite (initial reaction)
nucleation	liquid phase → crystallization of aluminate + ferrite
	alite formation

For many reactions the rate controlling pressure may not be known.

Since clinker burning is a most complex chemical process, it is not surprising that most of the mentioned reaction types are operative during the process. In the following, examples of these reaction types taking place in the kiln during production will be considered.

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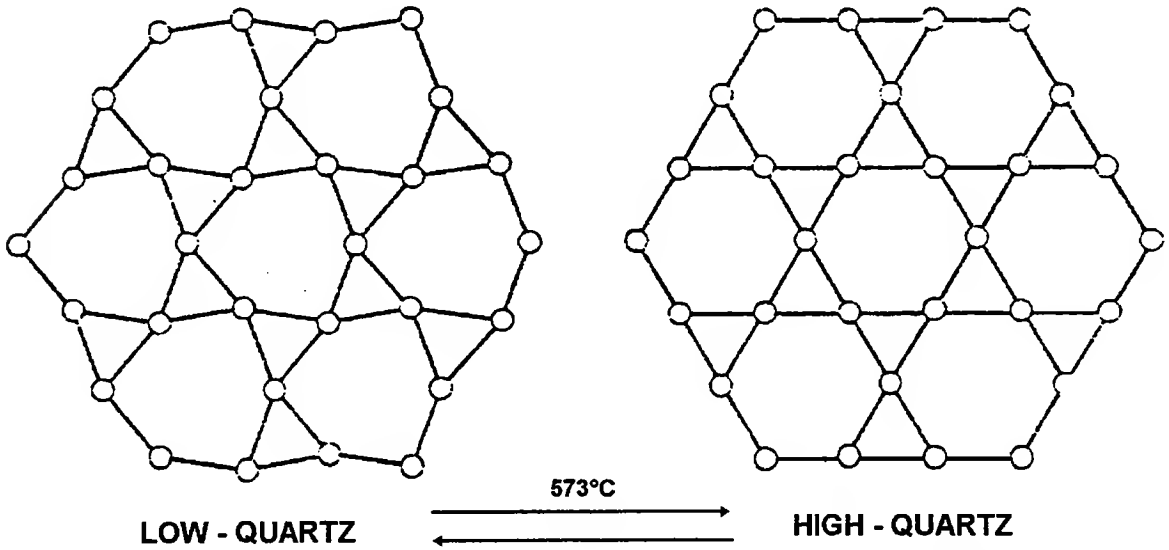
3.2 Examples

3.2.1 Structural Changes

This type of change occurring within single minerals in the solid state is also called a polymorphic transition (see paper 4). In this process the chemical composition of the minerals remains constant, but the spatial arrangement of the component atoms alters, usually upon reaching a specific temperature. Relevant examples of such polymorphic transition in clinker burning include the following:

a) The low - high quartz inversion (both have composition SiO_2)

Figure 6 Arrangement of the atoms in low and high quartz



In this transition the distorted low from, stable under 573 °C, changes into the more ordered high modification, the stable species above the transition temperature.

b) aragonite ➔ calcite inversion (both have composition $CaCO_3$)

Aragonite present in the raw meal transforms on heating into calcite. Consequently, kiln dust always contains calcite and not aragonite. The reason for this transition is, that under normal atmospheric pressure calcite and not aragonite is the stable from of $CaCO_3$. This can be easily seen in Figure 7, showing the stability conditions of calcite and aragonite.

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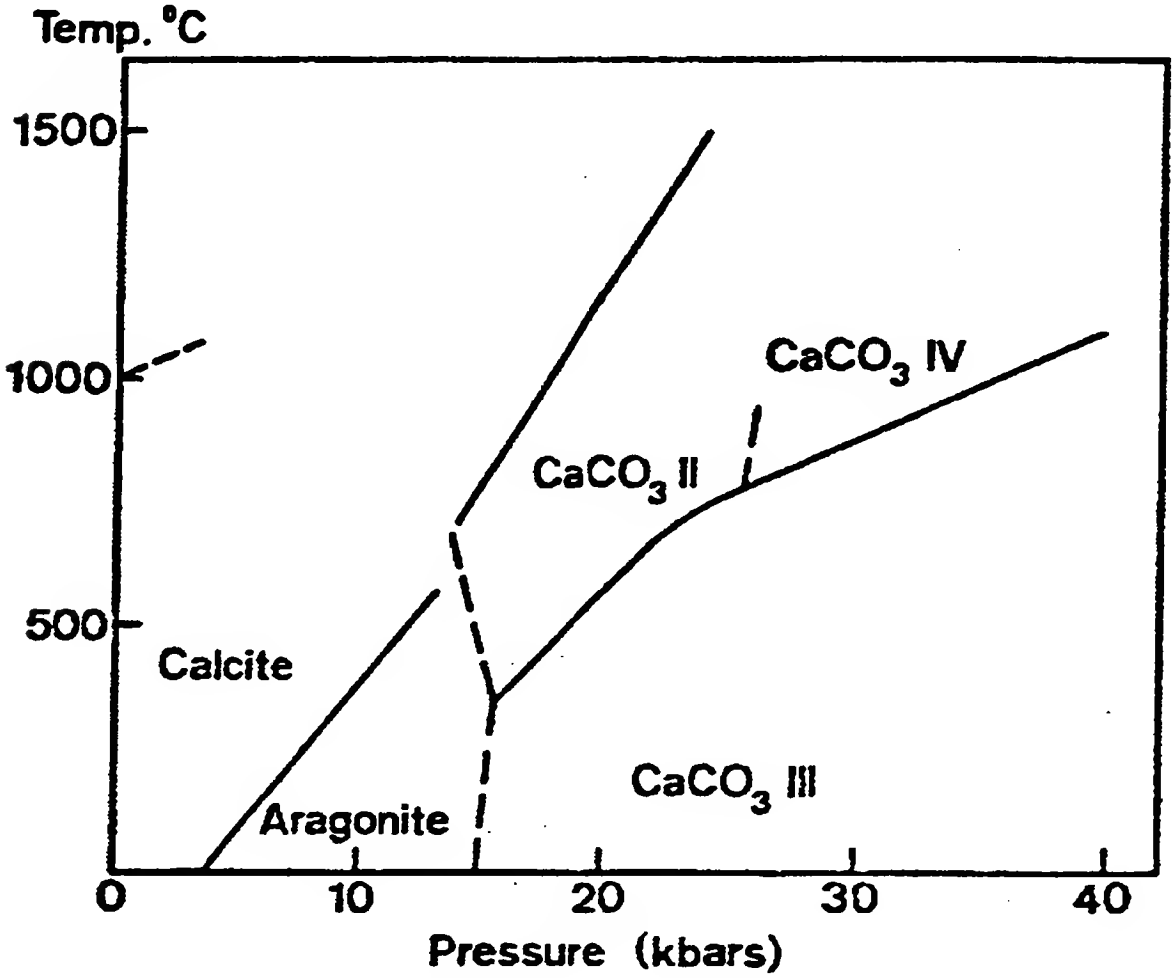
Figure 7 Stability conditions of the various forms of $CaCO_3$



c) alite and belite transformations

Even more important than the polymorphic transitions in the raw materials are the structural changes occurring in the principal clinker minerals alite and belite. These occur during both formation and cooling of the clinker, and contribute to the complexity of the clinker system. These transitions will be discussed in more detail in paper 13, pages 11 - 13.

3.2.2 Decomposition Reactions

The principal decomposition reactions encountered during clinker production include the following:

solid / gas type

- dehydroxylation of the clay minerals (kaolinite, etc.)
- decarbonation of the carbonate minerals (magnesite, dolomite, calcite, spurrite)

solid / solid type

- decomposition of alite

Characteristic of this reaction type is that the single reactant is transformed into two products.

a) Dehydroxylation of the clay minerals

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The removal of structural water from the clay minerals is a complex decomposition, as it further involves structural changes of the dehydrated product. This may be demonstrated for one of the simplest of the clay minerals compounds kaolinite, $2SiO_2, \cdot Al_2O_3 \cdot 2\ H_2O$.

There are two alternative theories about the changes that kaolinite undergoes when it looses water at about 500 - 600 °C. One theory is that the anhydrous alumino silicate $2SiO_2 \cdot Al_2O_3$ is formed, the other than an intimate mixture of amorphous silica and alumina results

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b) Decarbonation of the simple carbonate minerals

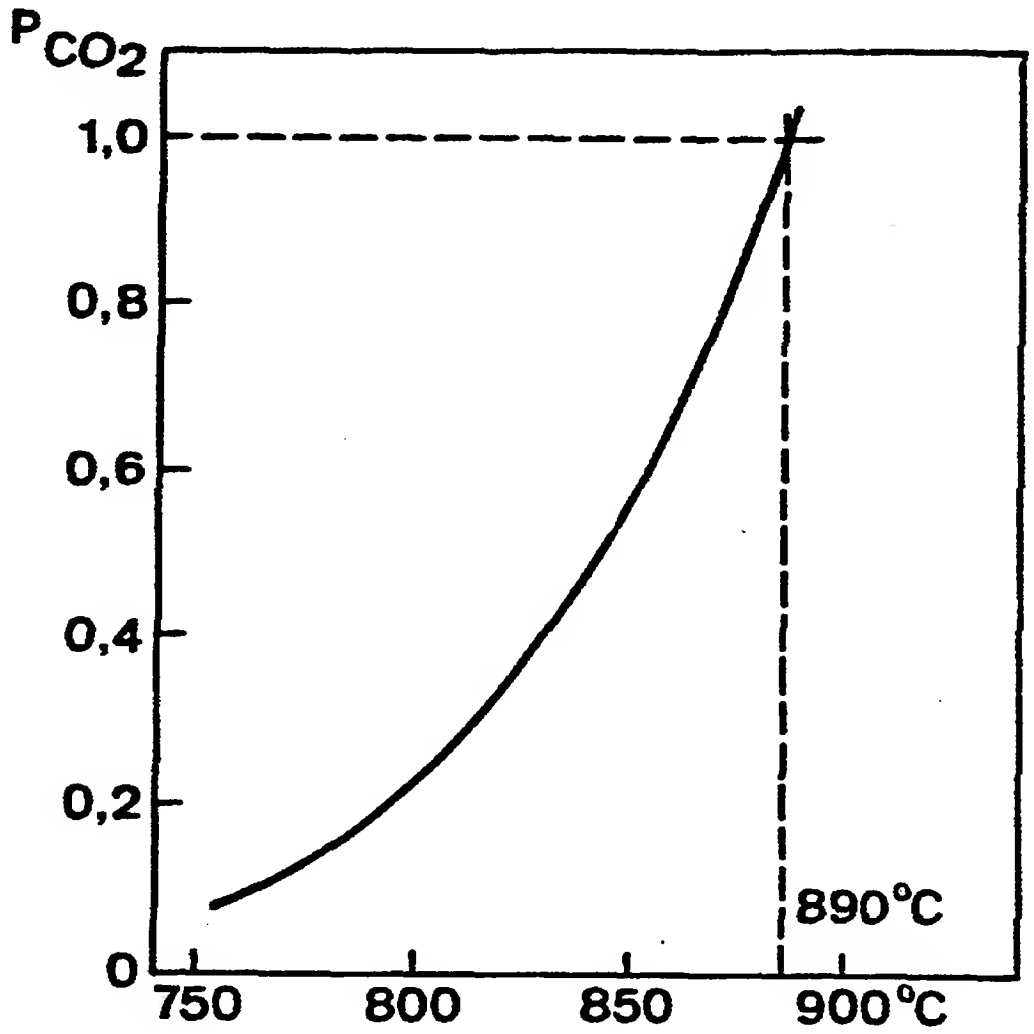
These reactions are of the type

MCO_3	$\leftrightarrow$	MO	+	CO_2
(solid)		(solid)		(gas)

where M = Ca (Calcite)
= Ca/Mg (Dolomite)
= Mg (Magnesite)

in which, if the CO_2 is not removed, there exists an equilibrium between the reactant and the reaction products. At any single temperature, both the degree of decarbonisation of the carbonate and the partial pressure of CO_2 (i.e. pCO_2) are fixed, providing equilibrium has been achieved. With increasing temperature the pCO_2 value increases until a certain given temperature the value equals atmosphere pressure as shown in Figure 8, so that the carbonate will completely decompose.

Figure 8 Equilibrium dissociation pressure of calcite with temperature



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Under conditions in which the CO_2 is constantly removed (e.g. in a flowing gas atmosphere at reduced pressure in which the pCO_2 value is lower than the equilibrium level) decomposition of the carbonate will take place at a reduced temperature.

In the simple carbonates found in cement raw mixes, the temperature at which the equilibrium pCO_2 level attains 1 bar is as follows:

Mineral	**Temp (°C)**	**pCO_2 = 1 bar**
Calcite	890	
Dolomite	745 and 890	
Magnesite	650	

c) Decomposition of compound carbonates (e.g. spurrite)

The decarbonisation of spurrite $2C_2S \cdot CaCO_3$ proceeds according to the equation

$2C_2S \cdot CaCO_3$	$\leftrightarrow$	$2C_2S$	+ CaO	+	CO_2
spurrite (solid)		belite (solid)	free lime (solid)		carbon dioxide (gas)

Just like the simple carbonates, the above equilibrium is reversible, and its extent depends on the temperature and pCO_2 value. Unlike the simple carbonates, two solid reaction products are formed, belite and free CaO, which at the temperature associated with spurrite decomposition under normal pressure, cannot react with each other.

The equilibrium pCO_2 value for spurrite is given in Figure 9 along with that pertaining to $CaCO_3$.

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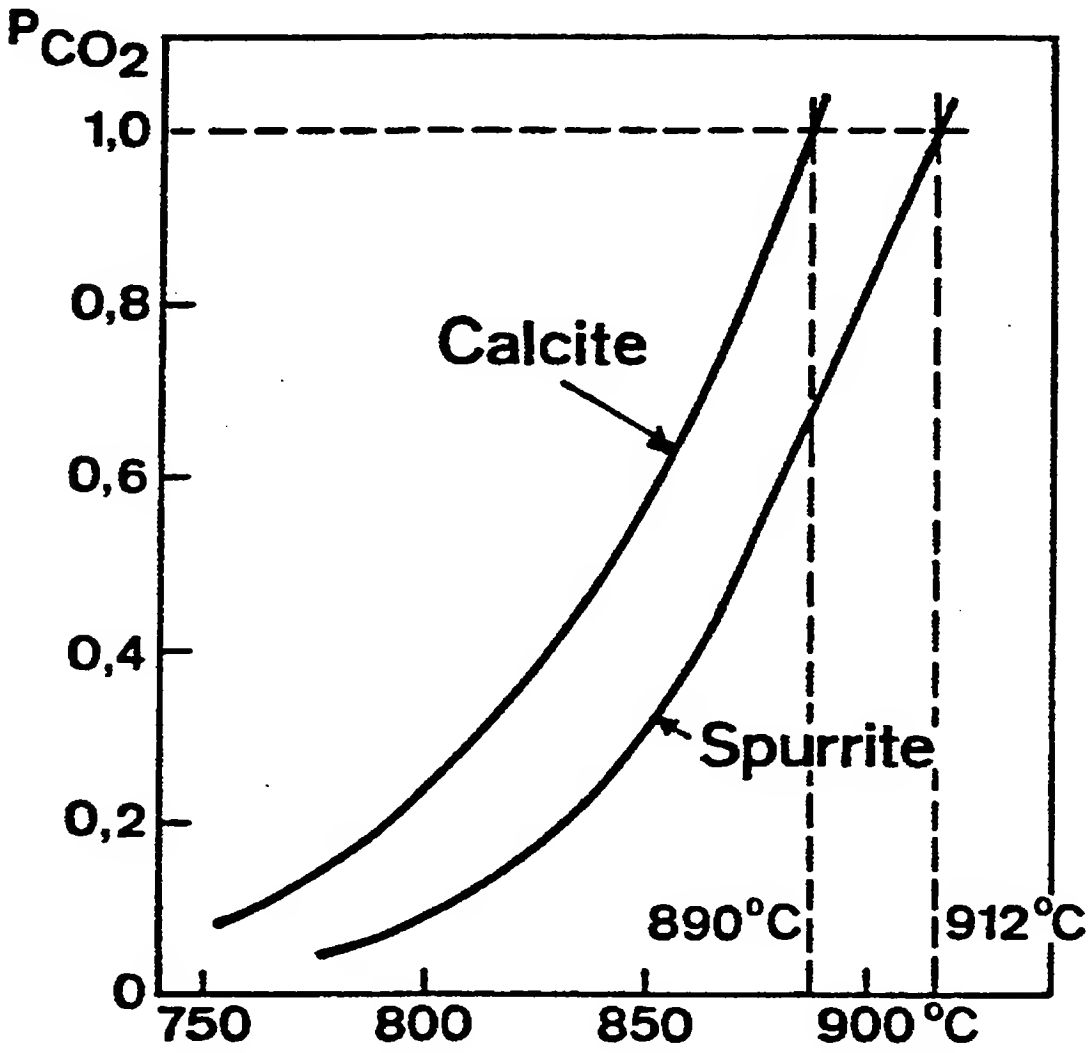
Figure 9 **Equilibrium dissociation pressure of calcite and spurrite with temperature**



From the figure it is evident that **spurrite is thermally more stable than calcite.** As a consequence of the greater stability of spurrite, it may form in cement kilns either by direct reaction of calcium carbonate and silica, or by reaction of CaO, Ca_2SiO_4 and gaseous CO_2.

The formation of spurrite with the latter mechanism may occur only in a small range of temperatures above the $CaCO_3$ decomposition temperature, for example, between 890 °C and 912 °C at 1 bar CO_2 pressure.

d) Decomposition of alite

A reaction of the solid-solid decomposition type which can, under certain circumstances, occur during clinker cooling, is the decomposition of alite. From the phase diagram for the system CaO - SiO_2 it can be deduced that C_3S has a lower limit of stability at 1250 °C, below which it will decompose into two solids, C_2S and CaO. The maximum rate of decomposition occurs at 1175 °C, at which temperature the S-shape decomposition curve (Figure 10) indicates that the initial decomposition is controller by nucleation of the products on the C_3S surface.

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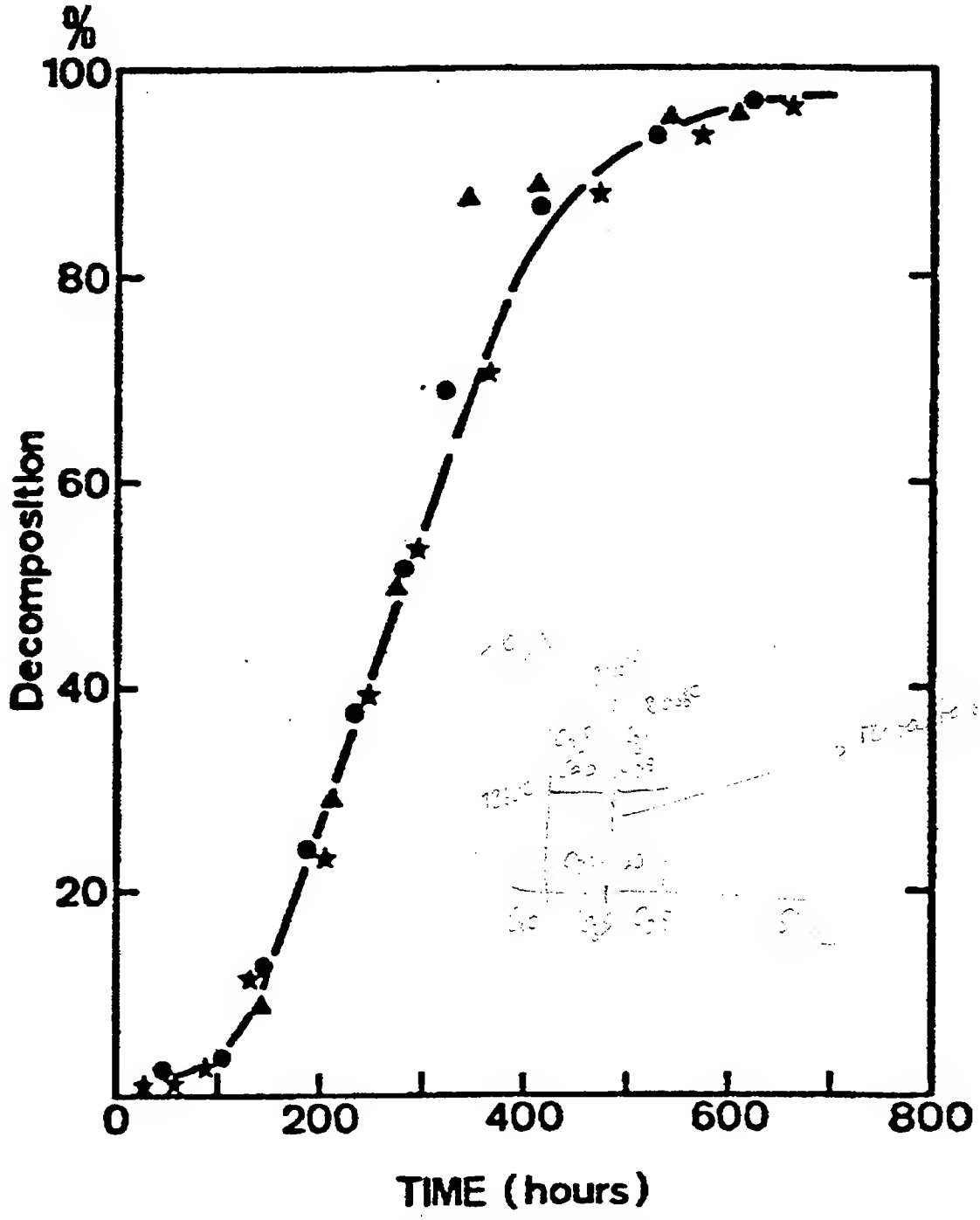
Figure 10 Decomposition of C_3S at 1175 °C



As can be deduced from figure 10, the rate of decomposition is so slow that for the cooling rates encountered during clinker cooling, the decomposition of C_3S should not be observed.

In the case of impure C_3S, i.e. clinker alite, the rate of decomposition is appreciably accelerated by:

- the presence of lime and C_2S nuclei
- the presence of Fe^{2+} , H_2O and K_2SO_4 / $CaSO_4$ melts

This aspect will be further considered in paper 13.

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3.2.3 Combination Reactions

The main combination reactions are the formation of the two major phases, the calcium silicates alite and belite, the only principal phases remaining unmelted at the highest clinkering temperature. The other major phases, aluminate and ferrite, although formed by combination reactions during heating, are melted at the highest temperatures and are present in the final cooled clinker as crystallization products of the melt.

a) Belite formation

The formation of belite starts at ≈ 800 °C and is mainly formed by solid-state reaction. In many raw mixes it is formed in quantities exceeding those found in the final clinker (see Figure 5) and then further reacts with free lime (via the liquid phase) to yield alite, the quantity of belite thus falling back to that finally observed.

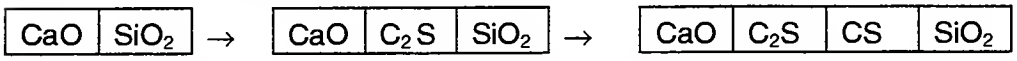
Belite formation (see Figure 4) is the result of a combination between the calcite and silica components of the raw mix.

$$\underset{\text{calcite}}{2\,CaCO_3} + \underset{\text{e.g. quartz}}{SiO_3} \xrightarrow{\approx 800°C} \underset{\text{belite}}{2\,C_3S} + CO_3$$

Although the pCO_2 value of calcite does not reach 1 bar until 890 °C, the initial formation of belite does take place at lower temperatures, due to the stability of the C_2S phase.

During the solid state reaction of silica and calcite (or CaO) the first reaction product is always C_2S. This can bee seen in Figure 11.

Figure 11 Sequence of appearance of phases in the solid state reaction of CaO and SiO_2



showing the order of appearance of phases at a CaO / SiO_2 interface. Only later does the most silica rich phase CS appear as a result of reaction s between C_2S and SiO_2. This demonstrates the remarkably strong nature of the belite formation reaction. At temperatures at which solid state reactions occur, the formation of C_3S does not take place due to its thermal instability.

The rate limiting mechanism by which belite is formed (after an initial phase boundary controlled reaction) depends on the diffusion of ions through the solid state. The rate of this reaction is thus dependent on:

- the path distance that the diffusing species have to travel
- defects in the reactant's crystal lattices.

The former factor is dependent on both the size of the CaO and SiO_2 particles, and the effective contact area between them. It is this easy to understand why finely milled and/or densely intergrown powder mixtures react faster than large-grained coarse ones, and thus, why cement raw meals must be finely ground and intimately mixed.

From the results of many studies, the main diffusion species is known to be the Ca^{2+} ion which migrates into the solid SiO_2, although evidence does exist to indicate that the diffusion of Si^{4+} into CaO can occur to a lesser extent.

That the Ca^{2+} ion is the major diffusing species is confirmed by the well known fact in clinker formation that the quartz must be finer ground than the limestone if complete reaction is to take place.

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b) Formation of alite

The formation of alite commences only when the temperature is above its lower stability limit of ≈ 1250 °C. At that temperature, the liquid phase is also starting to form. Consequently the formation of alite is a liquid - solid reaction, as reaction via a liquid medium is faster than by solid state processes. In the laboratory, C_3S can be formed by the solid state reaction of CaO and SiO_2, but this requires many hours at temperatures above those encountered during clinker formation. The resulting finely crystalline product must be rapidly quenched, at much faster rate than can be achieved in a production kiln, to prevent its decomposition at normal temperatures.

In the industrial production of clinker, the formation of alite and its stabilization is thus wholly dependent on the presence of the liquid phase.

At the temperature at which alite formation begins (≈ 1250 °C), the material consists mainly of free lime, belite and liquid phase. The formation reaction is, thus, of the type

$$C_2S + CaO \xrightarrow[> 1250\ °C]{\text{liquid phase}} C_3S$$

From the results of many laboratory experiments, the rate controlling step is considered to be the movement (diffusion) of dissolved CaO through the melt. This is supported by the following observations:

- alite particles often contain belite inclusions (see paper 13)
- when free lime is added to an already formed clinker, and the mixture again heated, pores are to be found where the lime particles had been, surrounded by a zone containing only alite.

Again, as in the case of belite formation, the rate of reaction is influenced by

- the distance the CaO has to travel (depending on raw mix fineness and homogeneity), and in addition to the quantity and viscosity of the liquid phase (controls the gas with which the movement of CaO can occur).

As the viscosity of the raw mix is reduced by

$$MnO > Fe_2O_3 > MgO > CaO > Na_2O$$

it can thus be appreciated that the presence of Fe_2O_3 and MgO in the liquid phase assists in the alite formation process.

4. KINETICS OF CLINKER BURNING

4.1 General Aspects

The burning of cement consists of a series of reactions taking place between finely divided solids, and it is only at temperatures above 1250 °C that a liquid is formed and becomes the medium through which reaction occurs. When liquids or solutions react together, the actions usually occur rapidly and the products depend only on the composition of the reaction mixture and the temperature. With reactions between solids the conditions are very different. Reaction takes place at the surface of grains, and diffusion of fresh solid to the surface proceeds but slowly. If a particle of clay is imagined surrounded by particles of lime, the reaction will commence at the surface of the clay particle. This in turn will react with further clay inside. If complete combination is not attained, the stage to which the reaction has progressed depends not only on the temperature, but also on the time, or rate of heating and the general physical and chemical condition of the reacting materials.

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This phenomenon of the rate of reaction can be schematically represented as follows:

	k_T	
raw mix	⟶	clinker
(1 - α)		α

where **k_T is the rate of reaction** at temperature T and α the extent of clinker formation after a certain time. The rate of reaction k_T is dependent on temperature and material characteristics (mainly mineralogy and fineness as expressed in the **fundamental Arrhenius** equation

$$k_T = Ae^{-Ea/RT}$$

k_T = rate constant at temperature T

T = reaction temperature

R = gas constant

E_a = activation energy

A = frequency factor

From the above equation it can be seen that the rate of reaction

- increases with them T and contact surface between raw mix components (frequency factor A)
- decreases with higher activation energy E_a for raw mix components.

This can be illustrated as follows. Experience demonstrates that a raw mix composed of non-reactive minerals (high activation energy E_a) such as quartz, forms clinker more slowly than a raw mix incorporating more reactive components such as clay minerals or amorphous silica. The lower activation energy required for the reaction with lime results in a high rate of clinker formation. To compensate for the **slow reactivity** of the less reactive minerals, a **higher burning temperature and / or longer burning period** (longer clinkering zone) is required.

The quantitative characterization of the kinetics of clinker formation, however, is extremely complicated, due to:

- Portland cement having a range of chemical compositions and being produced from a wide spectrum of different minerals
- the transformation takes place via a series of sequential and simultaneous reaction steps of various types
- the burning process is not isothermal but occurs in a heating process
- the non-availability of a simple, precise method for determining the extent of reaction (α)

Various attempts have, in spite of the complexity of the system, been undertaken to evaluate the kinetics of clinker formation. An example is given in the following table.

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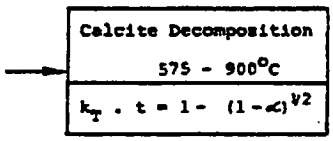
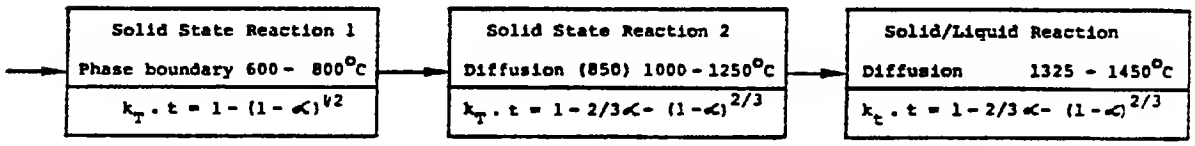
Table 8 Various models for describing the kinetics of clinker formation

Parallel Decomposition Stage



Consecutive Reaction Stages



The main feature is that the process has been subdivided into four stages having different reaction mechanisms and rates.

For further details Report No. MA 75/2408/E may be consulted.

4.2 Practical Considerations

When comparing the overall reaction velocity with the kinetics of the individual stages (Table 8) , it is known that the calcite decomposition, i.e. decarbonation of the raw meal, is not critical. **The combination reaction stages (solid/solid and solid/liquid) are the rate determining steps in the process.** The practical consequence of this is that it has been possible to replace the long rotary kiln by a suspension preheater kiln, in which the calcite decomposition may take place to a large extent within minutes. In the clinker formation stages, however, a rotary part of adequate length so as to allow sufficient reaction time, is indispensable. This is illustrated in Figure 12.

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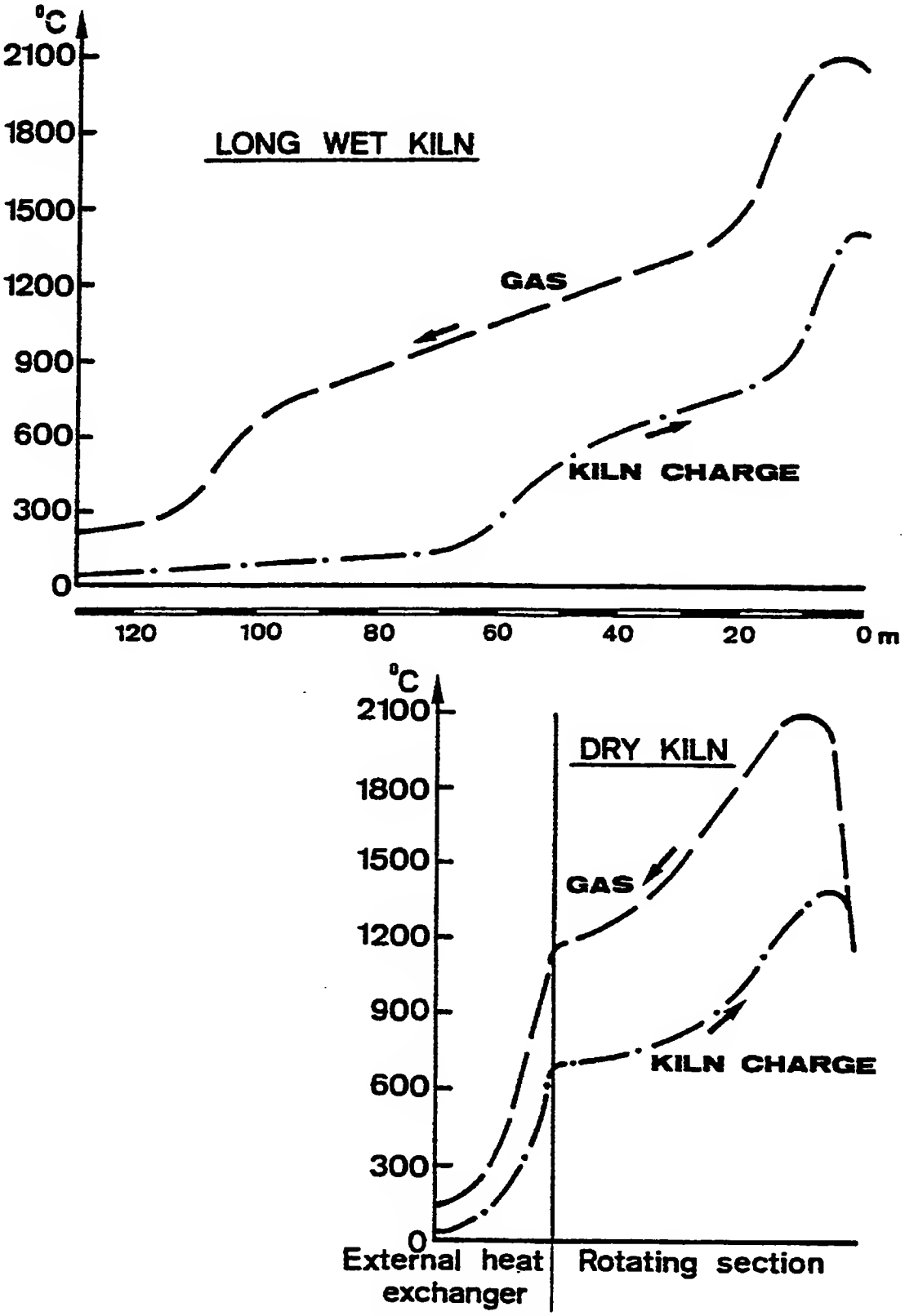
Figure 12 Temperature distribution in wet and dry process kilns



Consequently, to assess the progress of clinker formation, most consideration is given to events occurring after decarbonation.

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In practice, the most convenient method of following the reaction is by measuring the rate of decrease of uncombined lime (i.e. free lime). This technique is illustrated in Figure 13 showing two raw mixes, I and II, of identical chemistry (LS = 95, SR = 3.2 and AR = 2.2) and similar fineness ($R_{200\mu} \cong 0.5$ %, $R_{90} \cong 7$ % and $R_{60} \cong 15$ %). It is evident that the difference in mineralogy and actual particle size of the individual crystals influence both the mechanism and rate of reaction, especially at start of the clinker formation.

As can be seen in Phototable 1, mix I is composed of limestone and the fine-grained shale A which itself consists of finely mixed calcite (40 %) and siliceous (60 %) components, whilst mix II is composed of the same limestone but with the much coarser grained shale S containing a high proportion of coarse quartz particles. While raw mix I is considered to have a good burnability , raw mix II is considerably more difficult to burn.

Figure 13 Temperature change of DTA curves, phases identified and free lime

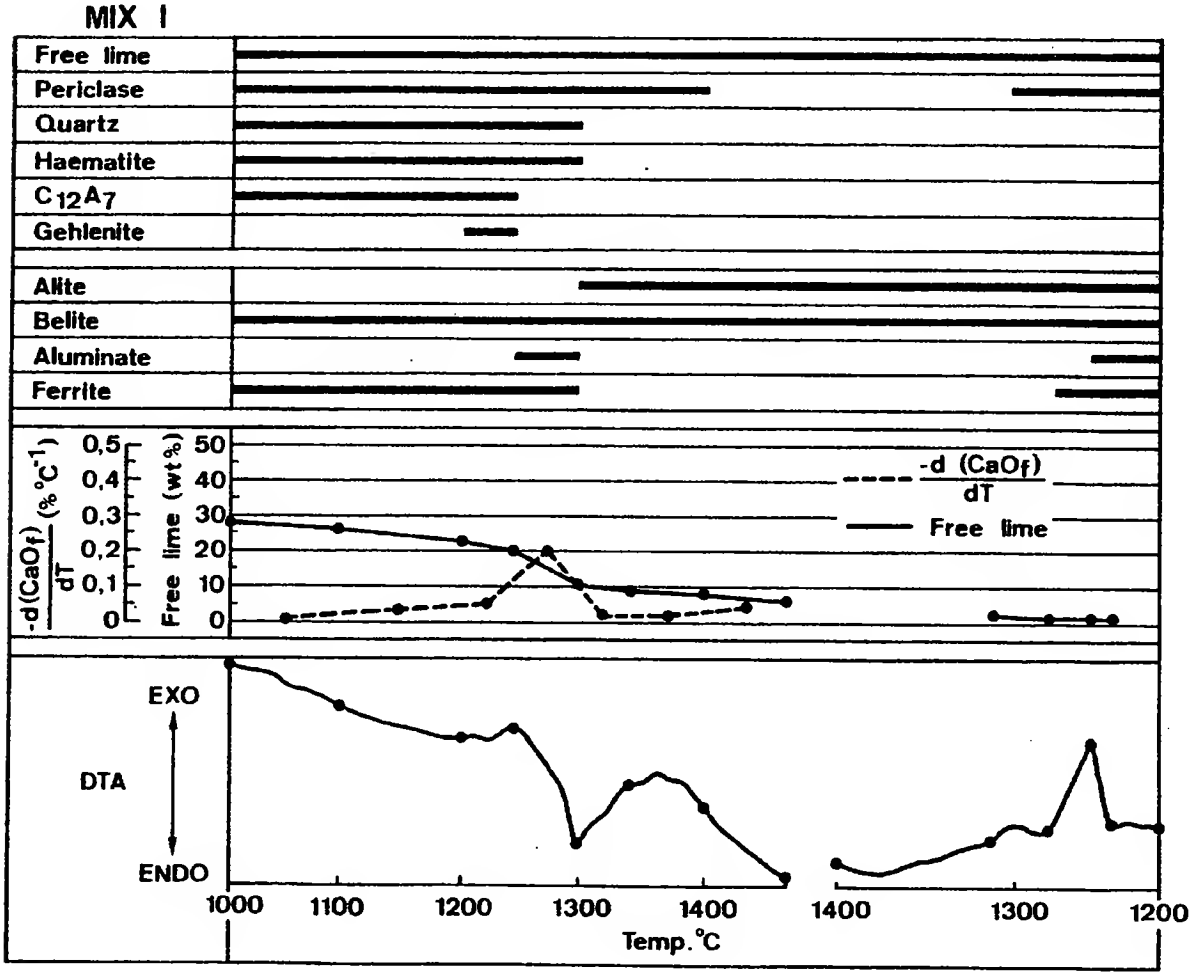
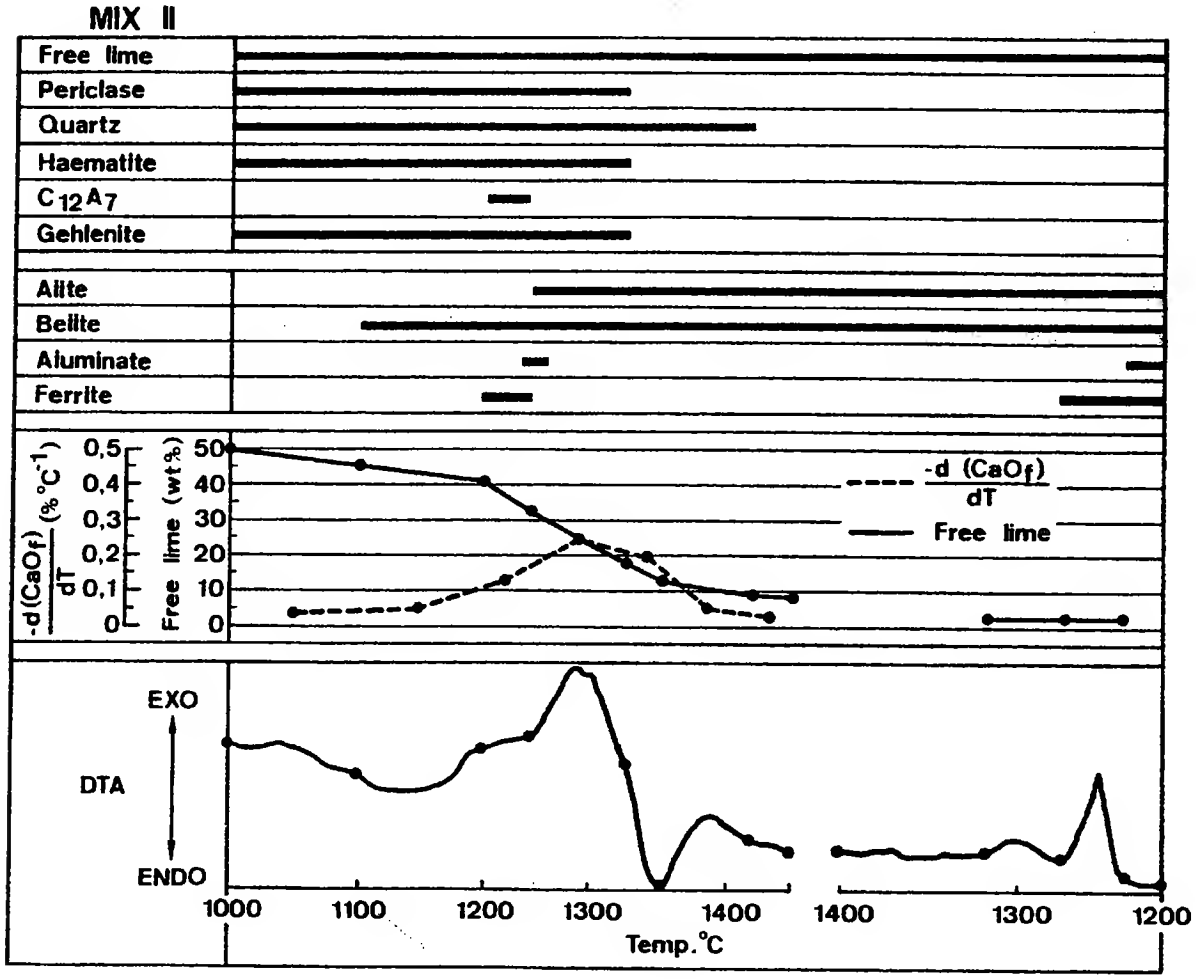


Figure 13 a Temperature change of DTA curves, phases identified and free lime



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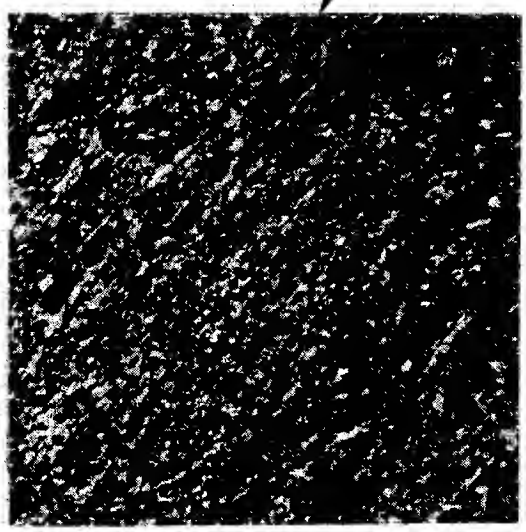
Phototable I Texture and Mineralogy of Mix Components

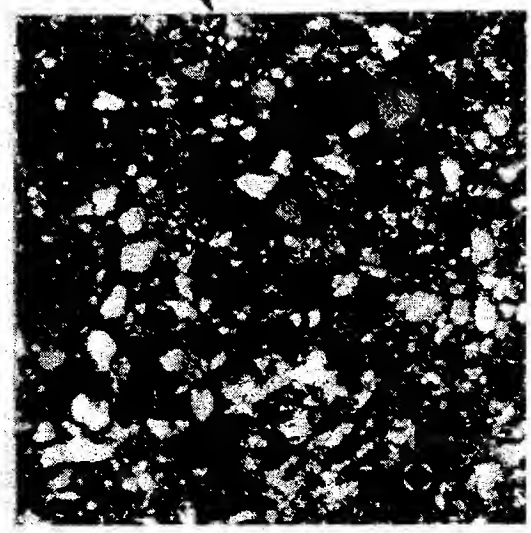
Limestone. 40 x

Raw Mix 1

Raw Mix 2



Shale A. 40 x



Shale S. 40 x

Designation	Limestone	Shale A	Shale S
Calcite	97 %	≈ 40 %	≈ 10 %
Dolomite	≈ 2 %	-	-
Quartz	traces	≈ 25 %	≈ 55 %
Chlorite	-	≈ 20 %	≈ 10 %
Illite and Micas	-	≈ 10 %	≈ 20 %
Pyrite	traces	≈ 2 %	traces
Feldspars	-	≈ 2 %	traces

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4.3 Assessment of Raw Meal Burnability

From the foregoing it will be realized that a true determination of reaction kinetics is at best feasible in fundamental research. In practice, simple methods are mostly applied to asses the **"burnability" of a mix, i.e. the ease of formation of the clinker minerals.** Three distinct methods are practiced at the "Holderbank" Technical Center:

- "Holderbank" burnability test -
  involves the heating of a raw mix under well defined laboratory conditions, and subsequent laboratory determination of the non-combined lime.
- Statistical burning model -
  in which **ten material parameters**, proven to be of significance, influence the rate of clinker formation. The uncombined CaO value, of any raw mix, relative to that of a standard raw mix prepared from identical components, is calculated.
- Physicochemical burning model -
  requires no standard raw mix. Only 4 parameters need to be considered.

4.3.1 "Holderbank" Burnability Test

The "Holderbank" burnability test is based on the isothermal burning at 1400 °C of raw meal nodules, details of which can be obtained in paper 5, page 5/18 ff.

By being carried out under standard laboratory conditions, this test allows the determination of the **relative influence of the various material parameters to be ascertained, free from the influence of process technological disturbances.**

Typical results obtained by this test for the raw mixes of all Group plants are listed in Figure 14, arranged in order of decreasing free CaO, i.e. increasingly better burnability.

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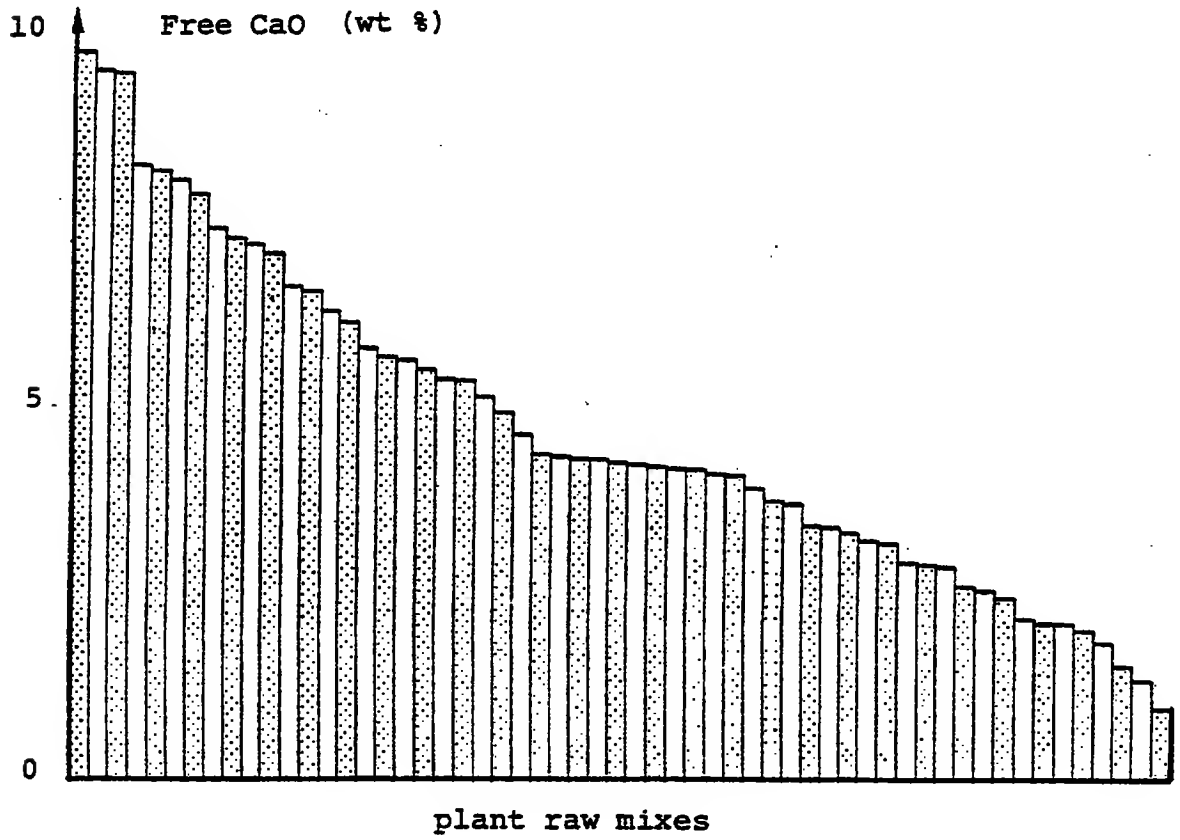
Figure 14 **Range of CaO values determined by the "Holderbank" burnability test on plant raw mixes**



The free lime value of the **laboratory** clinker extended from a minimum value of 0.94 % to a maximum value of 9.5 %. The values are distributed rather regularly around the mean value of 4.7 %. Based on this distribution and the corresponding plant experience, the burnability of raw mixes is ranked as follows:

Evaluation	Free Lime %
very good	0 - 2
good	2 - 4
moderate	4 - 6
poor	6 - 8
very poor	> 8

This classification has been shown to be applicable over many years of practical experience.

With the same raw meals the free lime values of the industrial clinker vary within a much smaller range, from a minimum value of 0.6 % CaO_{free} to a maximum value of 2.8 % CaO_{free}. The mean free lime value of the clinker examined from the "Holderbank" Group is about 1.3 % CaO with a standard deviation of s = 0.4 %.

The difference between the distribution of free lime for the laboratory produced clinks of the burnability test and plant clinkers can be seen in Figure 15.

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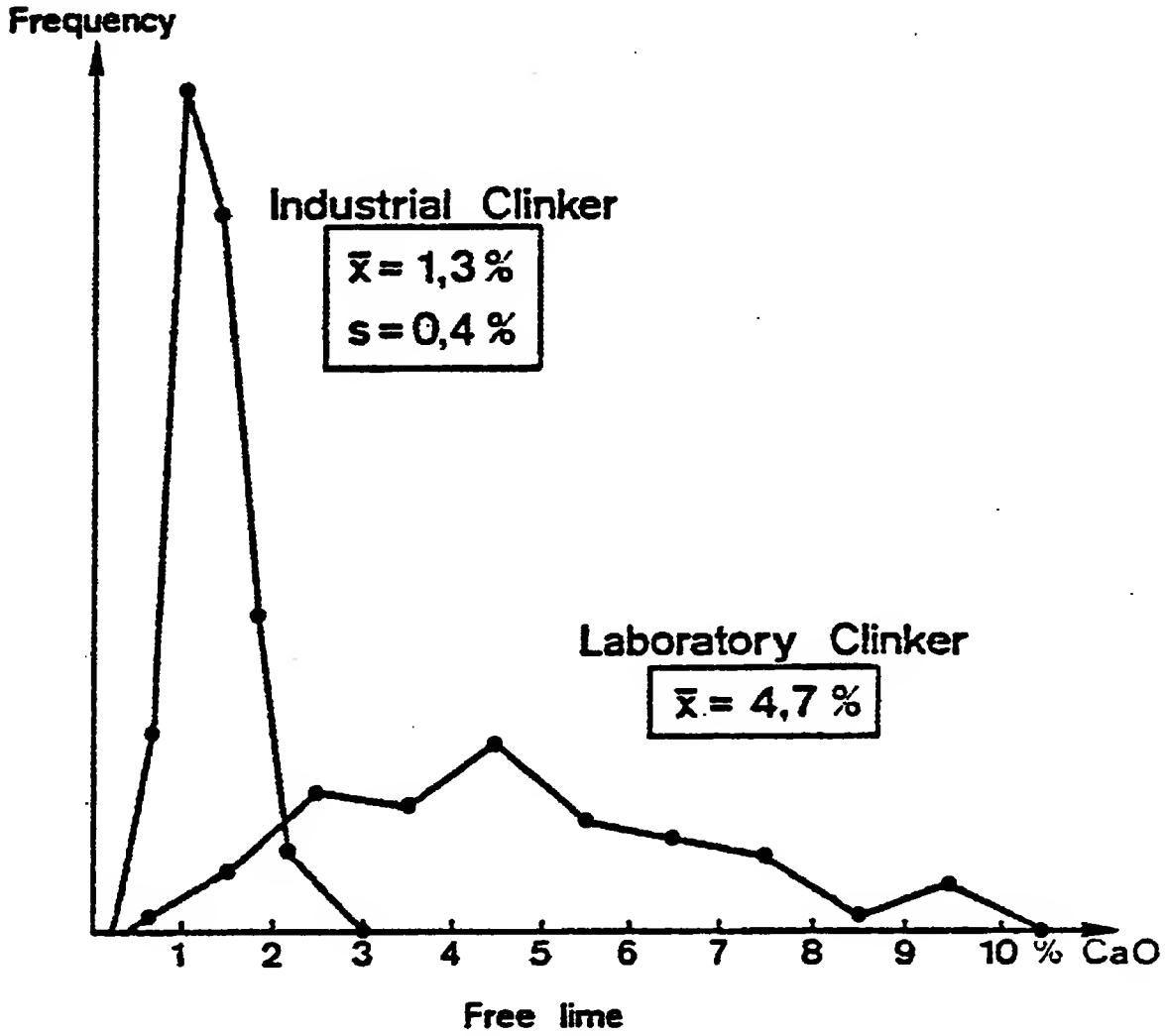
Figure 15 **Distribution of free CaO values for plant clinkers and for clinkers produced by the "Holderbank" burnability test**



This narrow range of free lime for plant clinkers can be explained by the quality requirements in the standards and of the market. The free lime values of industrial clinker, therefore, do not offer an explanation for the actual burning behavior of the raw mixes. The variability of raw mix laboratory burnability, as shown in Figure 14, is compensated by technological measures in the plant.

4.3.2 Statistical Burnability Model

In a quantitative evaluation of the data obtained by the "Holderbank" burnability test, a model has been developed - Statistical Burnability Model. The initial data for development of the model were the free lime values from burnability test at 1400 °C on 168 different raw mixes, the chemical and mineralogical composition, and the overall fineness of these mixes.

With this model, the 1350, 1400 and 1450 free lime values of other raw mixes from the same raw material components can be determined based on one single burnability test of one mix. In this way, the burnability model can be used as an instrument for **optimization of raw mixes** by allowing for the effect due to changes of the following factors:

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Chemical Parameters	lime saturation, silica ratio, alumina ratio, K_2O + Na_2O, MgO
Physical Parameters	residue on 200 and 90 µm sieves, quantity of mica, quartz and iron minerals

An impression of the use of the statistical burnability model may be gained from the data listed in Table 9 and presented graphically in Figures 16 and 17. Using a standard raw mix (no. 2), of overall fineness $R_{90\mu}$= 11 %, and determined free lime value of 4.9 % that would be observed by varying the temperature, overall fineness and chemical composition have been **calculated.** For a series of mixes from any one set of components it is cheaper, quicker and equally accurate to use the model rather than burning each mix in the laboratory.

Table 9 Influence of fineness and chemistry on laboratory burnability of a raw mix

Mix No.	1	2 (Reference)	3	4	5	6	7
Chemical Characteristics							
LS		93,0		96,0	93,0	93,0	96,0
SR		2,37		2,37	2,60	2,37	2,60
AR		1.72		1,72	1,72	1,90	1,72
Fineness %R							
90 um	8	11	17	11			
200 um	0,2	0,4	0,6	0,4			
Free lime							
1350°C	5,6	6,3	7,1	7,5	7,1	6,2	8,4
1400°C	4,3	4,9	5,6	6,0	5,6	4,8	6,8
1450°C	3,0	3,5	4,1	4,4	4,1	3,5	5,1

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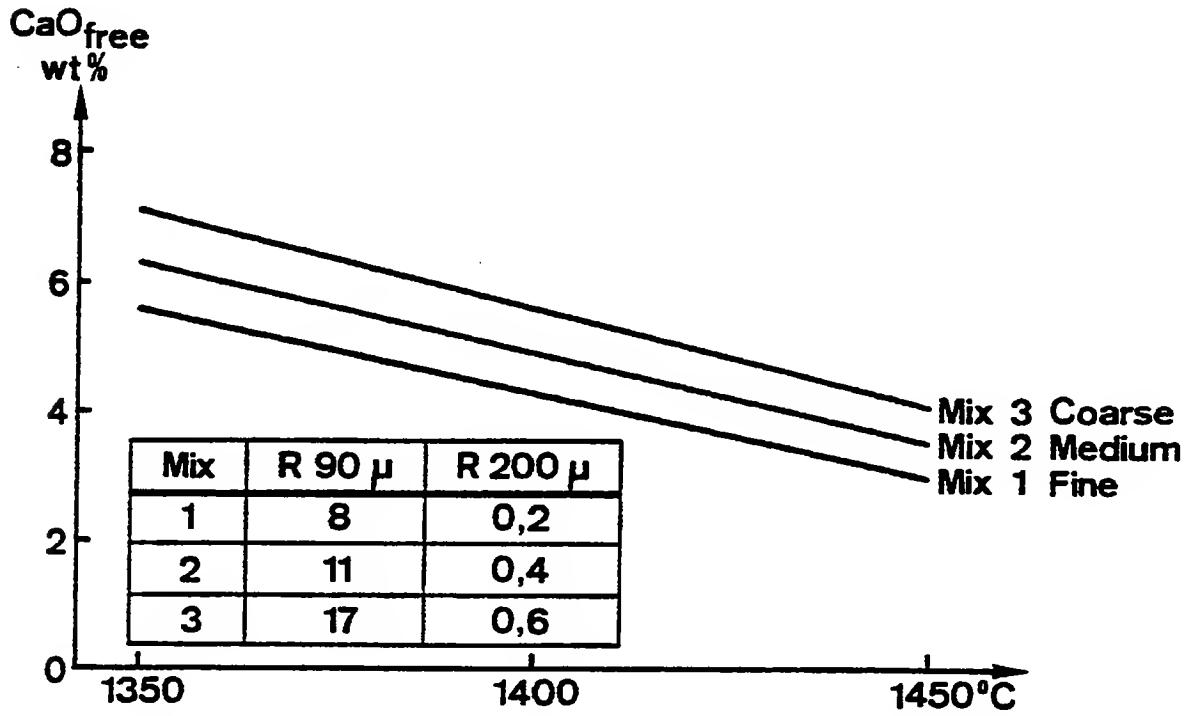
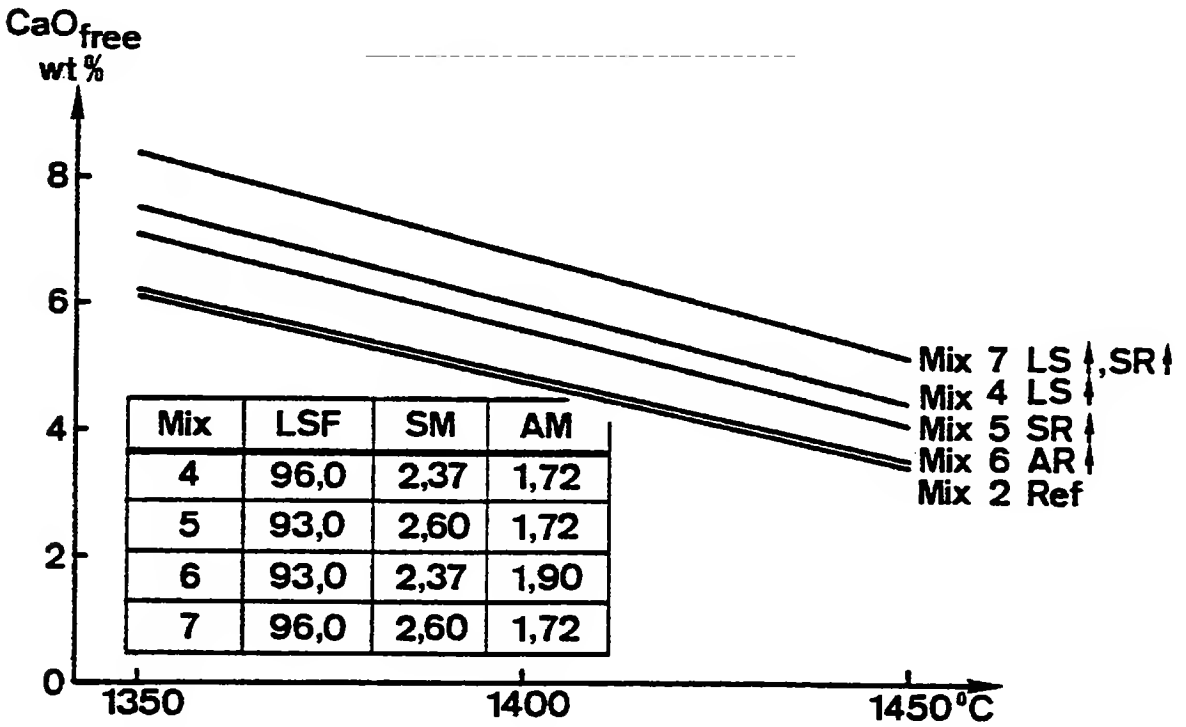
Figure 16 Change of free lime with raw meal fineness



Mix	R 90 µ	R 200 µ
1	8	0,2
2	11	0,4
3	17	0,6

Figure 17 Change of free lime with raw meal composition



Mix	LSF	SM	AM
4	96,0	2,37	1,72
5	93,0	2,60	1,72
6	93,0	2,37	1,90
7	96,0	2,60	1,72

Influence of burning temperature

As can be seen in Table 9 and Figures 16 and 17, increasing the burning temperature results - as would be expected - in lower free lime values. Thus, for example, mix 2 (4.9 % free lime at 1400 °C) would contain 6.3 % at 1350 °C and 3.5 % at 1450 °C. The influence

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of reducing the temperature 50 °C to 1350 °C is thus roughly the same as that resulting by increasing the lime saturation from 93 to 96%, as in mix 4.

Influence of raw meal fineness

Increasing the mix 2 residue on 90 μ from 11 to 17 %, results in the free lime at 1400 °C increasing to 5.6 %, while reducing the residue to 8 % decreases the free lime to 4.3 %. Increasing the residue to 17 %, thus has the same influence as that caused by raising the silica ratio from 2.37 to 2.6. Most practical use of being able to calculate the effects due to fineness is the assistance it gives when evaluating the optimum of raw meal mills.

Influence of LS, SR, and AR

The influence of increasing the LS, SR, and AR of mix 2 (LS 93, SR 2.7, and AR 1.72) can be seen in Figure 17. Increasing the AR to 1.9 has practically no effect, increasing the SR to 2.60 does, however, increase the free lime values (mix 5), but to a lesser extent than that caused by increasing the LS to 96. Increasing both the LS to 96 and the SR to 2.6, increases the free lime the most.

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4.3.3 Physicochemical Burnability Model

Further investigations during recent years have contributed additional knowledge with regard to burnability. It has long been known that the amount of uncombined lime depends on:

- Specific reaction area, i.e. the area of contact between the grains
- local oversaturation, i.e. grain size of individual minerals
- ambient conditions, i.e. pressure, temperature, and burning time
- diffusion coefficient of CaO through the liquid phase, i.e. composition of the liquid phase
- amount of liquid phase formed during burning
- supply and demand of CaO

It has recently been found that all these influencing factors may be incorporated in four parameters, namely silica ratio, lime saturation, amount of oversized quartz grains and amount of oversized calcite grains. Pressure, temperature and burning time are considered to be constant.

Silica ratio (SR) and lime saturation (LS)

The formation of C_3S from C_2S and CaO is governed by the diffusion of CaO through the melt. The silica modules and lime saturation are sufficient to describe this chemical reaction quantitatively.

The amount of CaO which can be accommodated within the liquid phase and in which it can diffuse and thus react, is inversely proportional to the silica ratio. A linear relationship exists between max. lime saturation and silica ratio values at which no free lime can be observed.

Quartz and calcite grains

Whether a grain of material reacts fully under given burning conditions depends on its diameter, structure and chemical composition. Monomineral particles above a certain critical size do not react fully, even after a long burning period. They are equivalent to local oversaturation of one chemical component. The overall raw mix residue is therefore not always a wholly reliable criterion for assessment of the burnability, unless the character of the grains is known. Quartz grains which exceed certain dimensions react with adjoining lime grains to from belite clusters. This prevents further reaction and gives rise to free lime.

Too large calcite grains result in CaO not being completely combined as also results from grains whose lime saturation is over 100 %.

For the "Holderbank" burnability test conditions the following grain diameters, quartz $\geq$ 32 µm and calcite $\geq$ 90 µm, were found to be critical limits.

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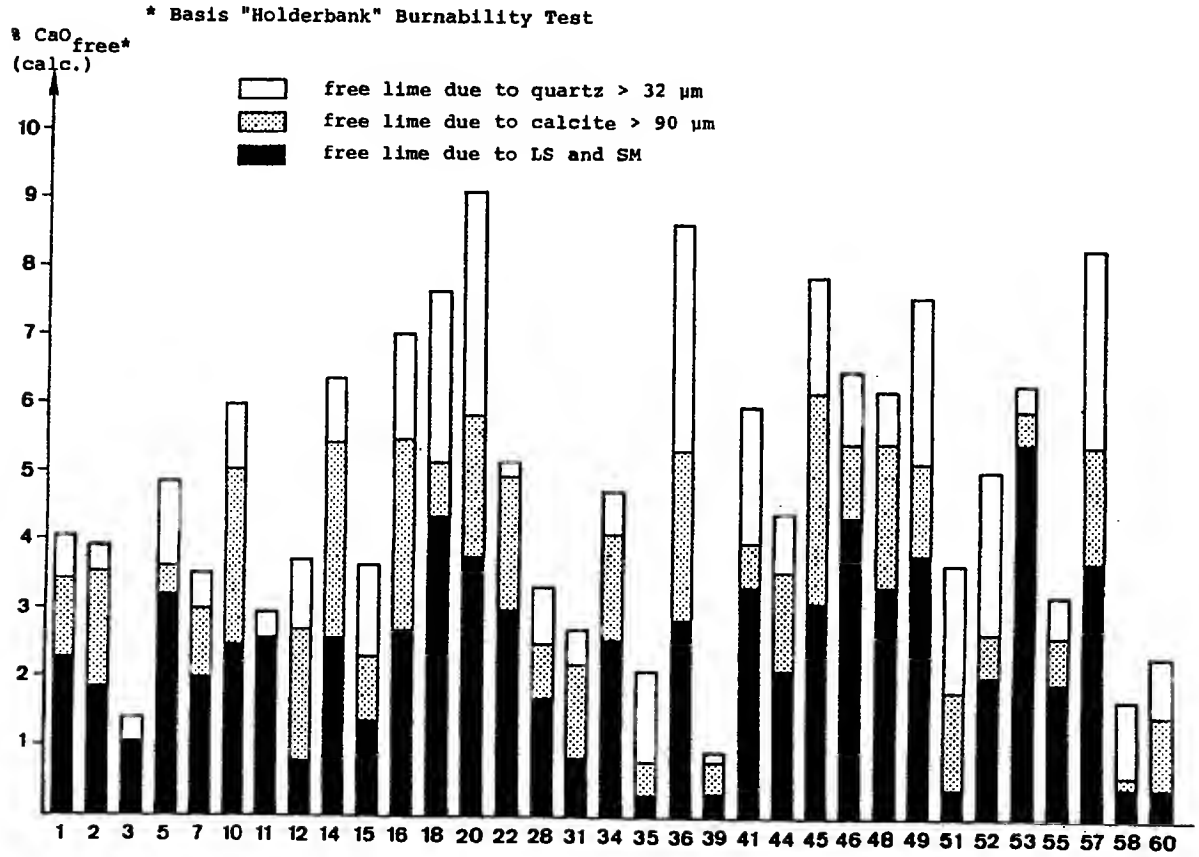
Figure 18 Influence of chemistry and coarse grains on burnability of raw mixes



An impression of the advantages offered by the physiochemical model may be drawn from the data presented in Figure 18 showing the influence of oversize quartz and calcite grains and of lime saturation (LS) and silica ratio (SR) on the uncombined CaO content of various industrial raw mixes.

The total free lime values calculated from the four mentioned parameters correlate well with the values determined experimentally by the "Holderbank" burnability test (r = 0.97), thus confirming that the four chosen parameters are capable of characterizing to a considerable extent raw meal burnability.

As can be easily observed (Figure 18), the quantity of free lime attributed to the individual parameters varies considerably. Whereas in some mixes the free lime is primarily associated with the chemical parameters (LS/SR), in others the influence of the quartz and calcite grains is predominant.

For example in raw meal no. 3, despite a high overall residue of 24 % on a 90 µm screen, no free lime results from the oversize calcite grains. The reason for this lies in the cement stone composition of this component, in which the individual calcite crystals are fine-grained and homogeneously distributed. The benefit of distinguishing between the overall fineness and the actual dimensions of the individual crystals can thus be appreciated.

In contrast to raw meal no. 3, about half the free lime in raw meal no. 10 is due to oversize calcite, originating from a high-grade limestone component of a titration value of 95 %.

In raw meals no. 46 and 53 the uncombined CaO can be mainly attributed to chemical rather than physical parameters. In the former it is mainly the high silica ratio (3.5), in the latter the very high lime saturation (103 %) which is responsible for the resulting content of free lime. Finer grinding of these raw meals would thus not appreciably lower the

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uncombined lime values of clinker burnt in the burnability test. Only a change in chemical composition would improve the burnability of these raw meals.

In raw meal no. 48, despite the large overall residue (43 % > 90 µm) and the given 6.5 % quartz content, the influence of quartz is not unduly great. The reason for this is that the individual crystals are small and well distributed, even within the individual grains.

It is now obvious that by individually examining the 4 parameters which principally influence the burnability, it is easier to characterize, then was previously the case, the critical features of any raw mixture.

It is thus, for instance, easier to decide whether components should be combined or separately ground, or to decide the optimal chemical and physical properties of any raw mix.

5. THERMOCHEMISTRY OF CLINKER FORMATION

A detailed discussion of the heat balance of rotary kiln lies outside the scope of this paper and will, in any case, be considered later under the heading of process technology. What will, however, be mainly discussed are the basic thermochemical data relevant of the material aspects of clinker production.

During clinker production, heat is both absorbed (endothermic heat changes) and produced (exothermic heat changes in certain characteristic temperature rings).

On the basis of the reaction sequence already presented in Table 2, the heat changes pertinent to the individual reactions are known to be as follows:

Table 10 Nature of heat changes occurring during clinkering

Temperature range (°C)	Type of Reaction	Heat Change
20 - 100	Evaporation of free H_2O	Endothermic
100 - 300	Loss of physically adsorbed H_2O	Endothermic
400 - 900	Removal of structural H_2O (H_2O, OH groups from clay minerals)	Endothermic
600 - 900	Dissociation of CO_2 from carbonate	Endothermic
> 800	Formation of intermediate products, belite, aluminate and ferrite	Exothermic
> 1250	Formation of liquid phase (aluminate and ferrite melt)	Endothermic
	Formation of alite	Exothermic
1300 - 1240 (cooling cycle)	Crystallization of liquid phase into mainly aluminate and ferrite	Exothermic

A most effective way of visualizing the temperature at which the heat changes occur, their nature and magnitude, can be obtained from differential thermal analysis (DTA) curves of cement raw mixes, an example of which is given in Figure 19.

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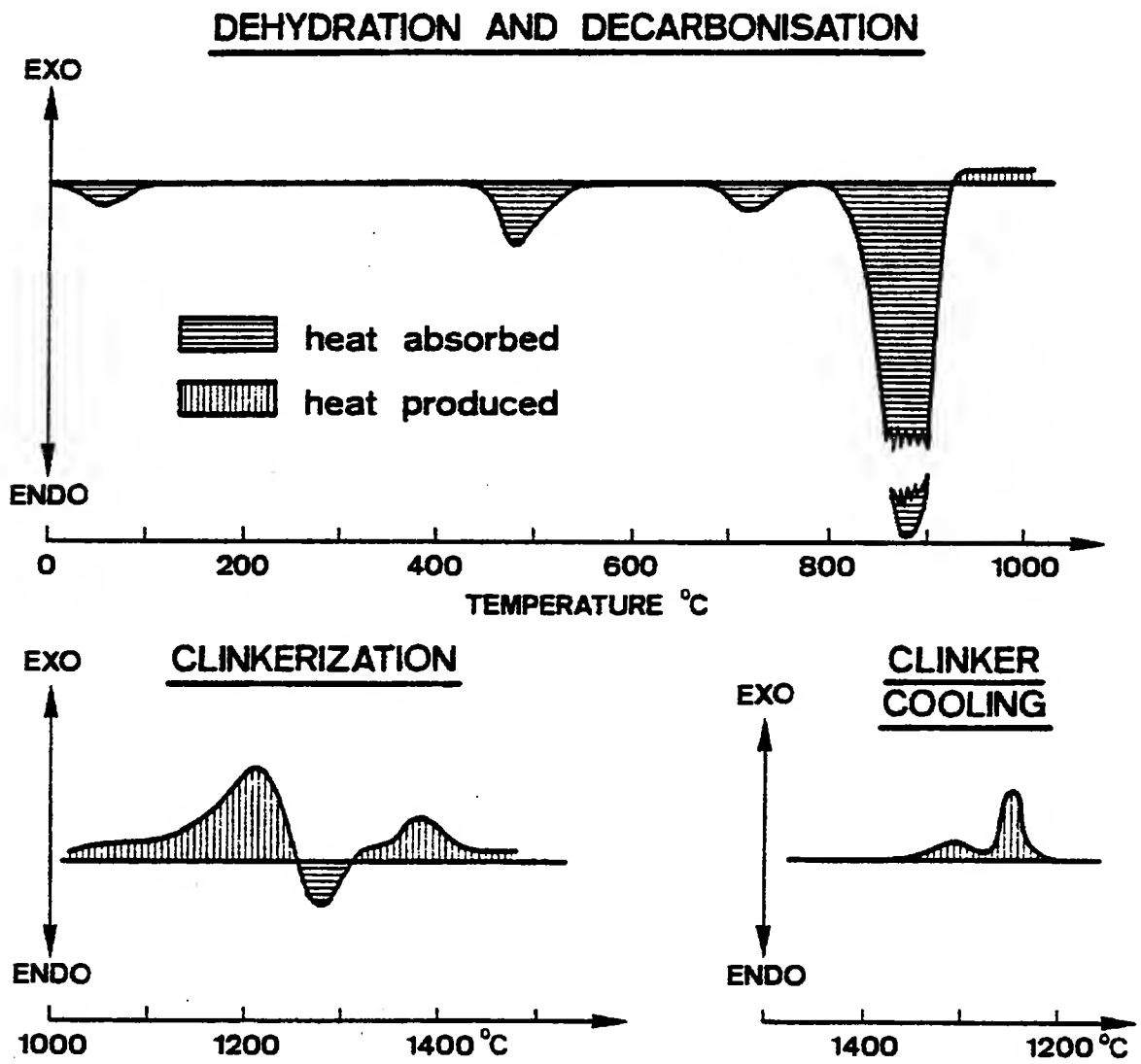
Figure 19 **DTA curves of typical cement raw meals**



It is obvious from these curves that by far the greatest heat requirement occurs between 850 - 900 °C, i.e. for the decomposition of the carbonate minerals. The total heat requirements for dehydration, decarbonisation and melting exceed the heat liberated by the formation of belite and the intermediate and final products.

Not included in the DTA representation, however is the heat involved in the actual heating of the raw meal and the cooling of the clinker and of the CO_2 and H_2O gases obtained from the calcite and the clay minerals.

The heat of absorption and liberation, however, varies according to the nature of the individual raw mix, as is clearly demonstrated in Figure 13. **The DTA curve**, therefore, allows a **differentiation of the reactivities** of different raw mixes to be made.

When the **quantitative** values for the various reactions are considered, the heat requirements for the material only with respect to clinker formation may be expressed as follows in approximate terms.

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Table 11 Thermal balance of heat changes occurring during clinker formation

Endothermic processes:	**dehydration, decreasing, heating**	
	kJ/kg clinker	**(kcals/kg)**
dehydration of clays	≈ 170	≈ 40
decarbonisation of calcite	≈ 1990	≈ 475
heat of melting	≈ 105	≈ 25
heating of raw materials 0 - 1450 °C	≈ 2050	≈ 490
	≈ 4315	≈ 1030
Exothermic processes:	**heat of formation, cooling of clinker, CO_2, H_2O**	
	kJ/kg clinker	**(kcals/kg)**
crystallization of dehydrated clay	≈ 40	≈ 10
heat of formation of clinker minerals	≈ 420	≈ 100
crystallization of melt	≈ 105	≈ 25
cooling of clinker	≈ 1400	≈ 335
cooling of CO_2 (ex calcite)	≈ 500	≈ 120
cooling of H_2O (ex clays)	≈ 85	≈ 20
	≈ 2550	≈ 610
Net theoretical heat of clinker formation (endothermic - exothermic)	≈ + 1765	≈ + 420

Depending on the carbonate content, lime saturation and mineralogical composition, the net heat requirement varies between approx. 1630 - 1800 kJ/kg clinker (i.e. 390 - 430 kcals /kg).

These values, of course, only apply to a dry raw meal. For a slurry, the heat requirements will certainly be higher by that necessary of the removal of water. This value will be dependent on the actual slurry water content, and for the average slurry of 35 % H_2O is ≈ 2100 kJ/kg clinker. The **net theoretical heat of formation of 1 kg clinker from slurry is, therefore ≈1765 + 2100, i.e. 3865 kJ/kg** (920 kcals/kg).

When considering the heat required for actual clinker production, involving heat losses, the values of approx. 1700 and 3865 kJ/kg clinker are too low. A more realistic heat balance taking process inefficiency into consideration, is approximately as follows:

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Table 12 Heat balance of wet and dry kiln

	Dry kiln		Wet kiln	
	kJ/kg clinker			
Evaporation of H_2O (0.4 % dry kin and 35 % wet kiln)	≈ 20	(≈ 0.6 %)	2100	(38 %)
Heat of reaction	≈ 1765	(≈ 54 %)	≈ 1765	(≈ 32%)
Heat losses through gas, clinker, dust, etc.	≈ 840	(≈ 26 %)	≈ 1250	(≈ 23 %)
Heat losses by radiation and convection	≈ 650	(≈ 20 %)	≈ 360	(≈ 7 %)
	≈ 3275 kJ/kg		≈ 5475 kJ/kg	

The heat losses due to radiation, exhaust gas, etc. will be explained in more detail under the heading of process technology.

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Chapter 2

Portland Cement Clinker

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Portland Cement Clinker

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1. CLINKER MINERALOGY

1.1 Theoretical Clinker Phases

The chemical composition (wt.%) of ordinary Portland cement clinker in the 'Holderbank' Group is on the average as follows:

Loss on ignition	0.48
SiO_2	22.2
Al_2O_3	5.0
Fe_2O_3	2.8
CaO	65.4
MgO	1.8
SO_3	0.54
K_2O	0.63
Na_2O	0.25
TiO_2	0.27
Mn_2O_3	0.09
P_2O_5	0.14
Cl^-	0.01
F^-	0.08
	99.70
CaO_{free}	1.3

With this knowledge it is possible to predict both:

- the phases which should theoretically be present in a cooled clinker of this composition
- the quantity of each phase which should be present.

For the given average chemical composition the potential mineralogy is calculated to be as follows:

C_3S	=	54.3 %	C_3A	=	8.5 %
C_2S	=	22.8 %	C_4AF	=	8.7 %

How was this calculation performed?

To predict the minerals which should potentially be present, the **phase equilibrium diagram** for the appropriate oxide system was consulted. Such diagrams are most easily located in the standard reference series 'Phase Diagrams for Ceramists' (Ref. 1).

To a first approximation the average clinker may be considered as consisting of only the three major oxides CaO, SiO_2 and Al_2O_3 and hence its composition may be plotted using the well-known ternary phase diagram, CaO - Al_2O_3 - SiO_2. The given composition, i.e. 65.4 % CaO, 22.2 % SiO_2, and 5.0 % Al_2O_3, when recalculated in terms of these three oxides alone, yields 70.6 % CaO, 24.0 % SiO_2, and 5.4 % Al_2O_3, totaling altogether 100 %.

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Consulting the CaO - Al_2O_3 - SiO_2 phase equilibrium diagram reproduced in a simplified formation in Figure 1, exhibiting only the **sub-solidous equilibrium relationships**, it can be demonstrated that the average composition ✶ lies within the C_3S - C_2S - C_3A compatibility triangle.

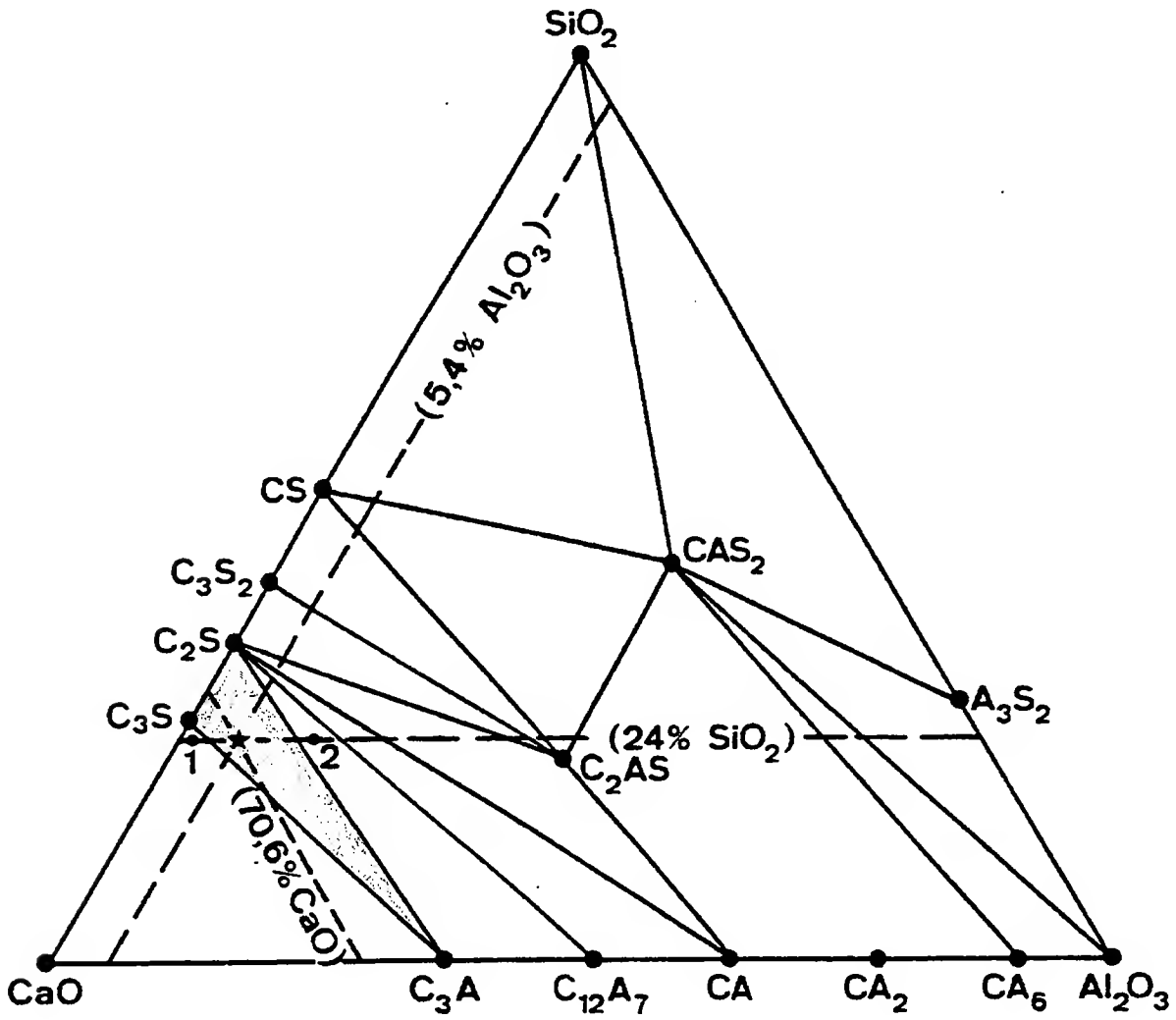
Because the chosen clinker composition lies within this triangle, these three phases will be formed on heating, provided that equilibrium is attained.

Small changes in the chemical composition of the mix are accompanied by changes in the **quantity** of the clinker phase C_3S - C_2S - C_3A. As soon as the changes exceed a certain limit, different minerals are then formed.

Thus, if the SiO_2 composition is held constant at 24 % and the Al_2O_3 reduced to 2 % (i.e. point 1 in Figure 1), the phases formed would be C_3S - C_3A- CaO, whereas increasing the Al_2O_3 to 12.5 % (point 2 in Figure 1) would result in the formation of C_2S - C_3A - $C_{12}A_7$.

It can thus be realized why **maintaining the correct composition is important** in the production of Portland cement, and **why it cannot be varied over too wide a range.** Because only the three phases C_3S - C_2S - C_3A possess the desired hydraulic properties, cement clinker composition should lie within the C_3S - C_2S - C_3A triangle.

Fig. 1: Sub-solidous phase equilibrium relationships in the system CaO - Al_2O_3 - SiO_2

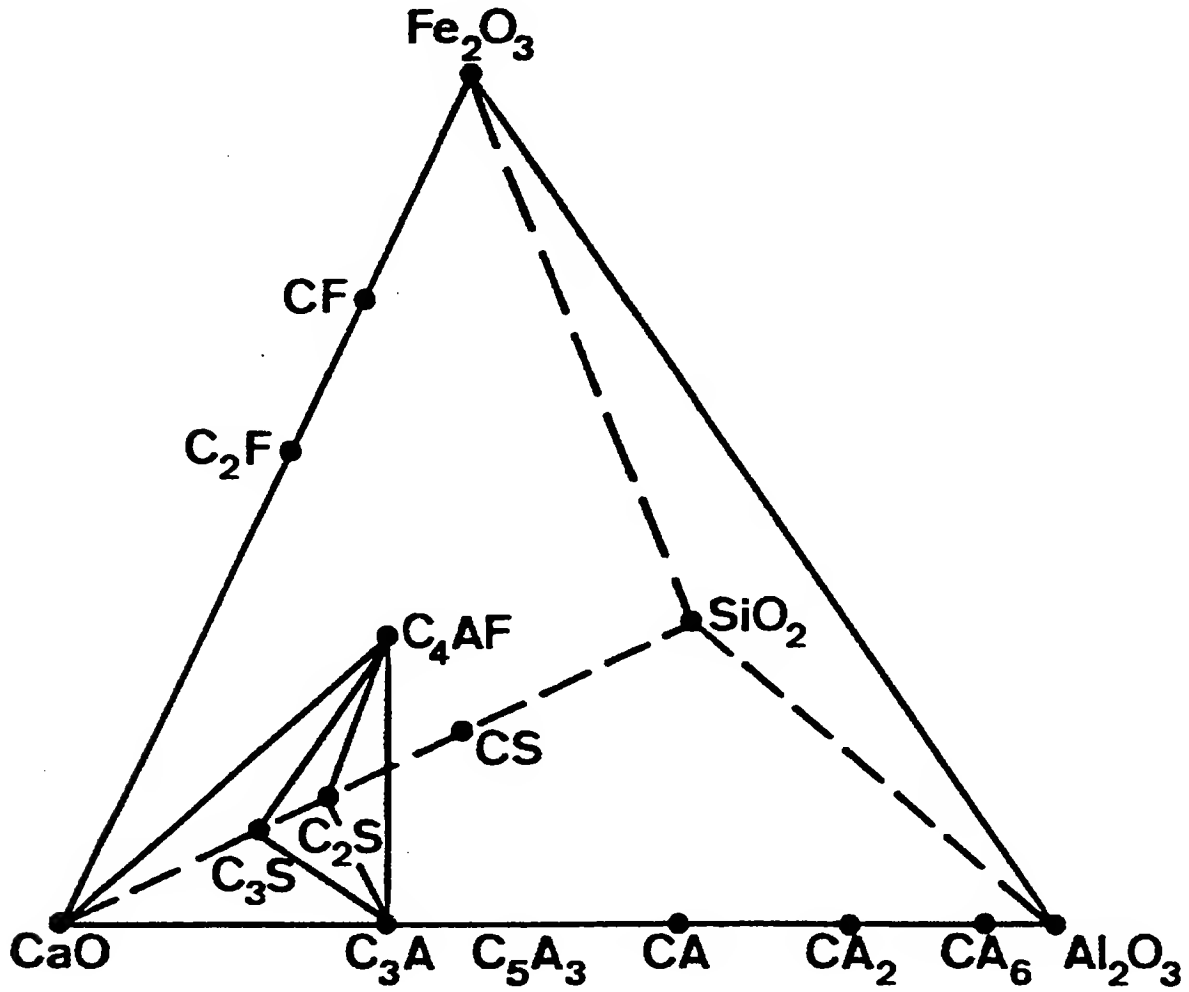


Clinker containing CaO - C_3S - C_3A would contain undesired free lime (responsible for expansion on hydration) while clinker containing C_2S - C_3A - $C_{12}A_7$, would contain no C_3S, and would therefore exhibit unfavorable hydration and hardening characteristics.

A more accurate prediction concerning the theoretical phases present is obtained when the presence of Fe_2O_3 is taken into consideration. On recalculation in terms of the 4 principal oxides, the composition of the so far considered average clinker becomes 68.5 % CaO, 23.2. % SiO_2, 5.3 % Al_2O_3 and 3.0 % Fe_2O_3, again totaling 100 %.

As in the case for CaO - Al_2O_3 - SiO_2, the above composition when plotted in the CaO - Al_2O_3 - Fe_2O_3 - SiO_2 phase equilibrium diagram (Figure 2), shows the potential composition as lying within the C_3S - C_2S - C_3A - C_4AF compatibility volume and hence consists of these phases.

Fig. 2: Sub-solidus compatibility relations in the system CaO - Al_2O_3 - Fe_2O_3- SiO_2



A point of special interest here is that three of the phases C_3S - C_2S - C_3A are binary compounds, i.e. composed of only two oxides. The other phase C_4AF is a ternary oxide, composed of CaO, Al_2O_3 and Fe_2O_3. Although C_4AF is often considered as being an individual compound of fixed composition, it is in reality a member of the solid solution series 2 CaO. Fe_2O_3 - 'CaO . Al_2O_3' in which 50 % of the Fe^{3+} atoms are replaced by Al^{3+} atoms, hence giving the composition 2 CaO · 1/2 Al_2O_3 · 1/2 Fe_2O_3, i.e. 4 CaO · Al_2O_3 · Fe_2O_3.

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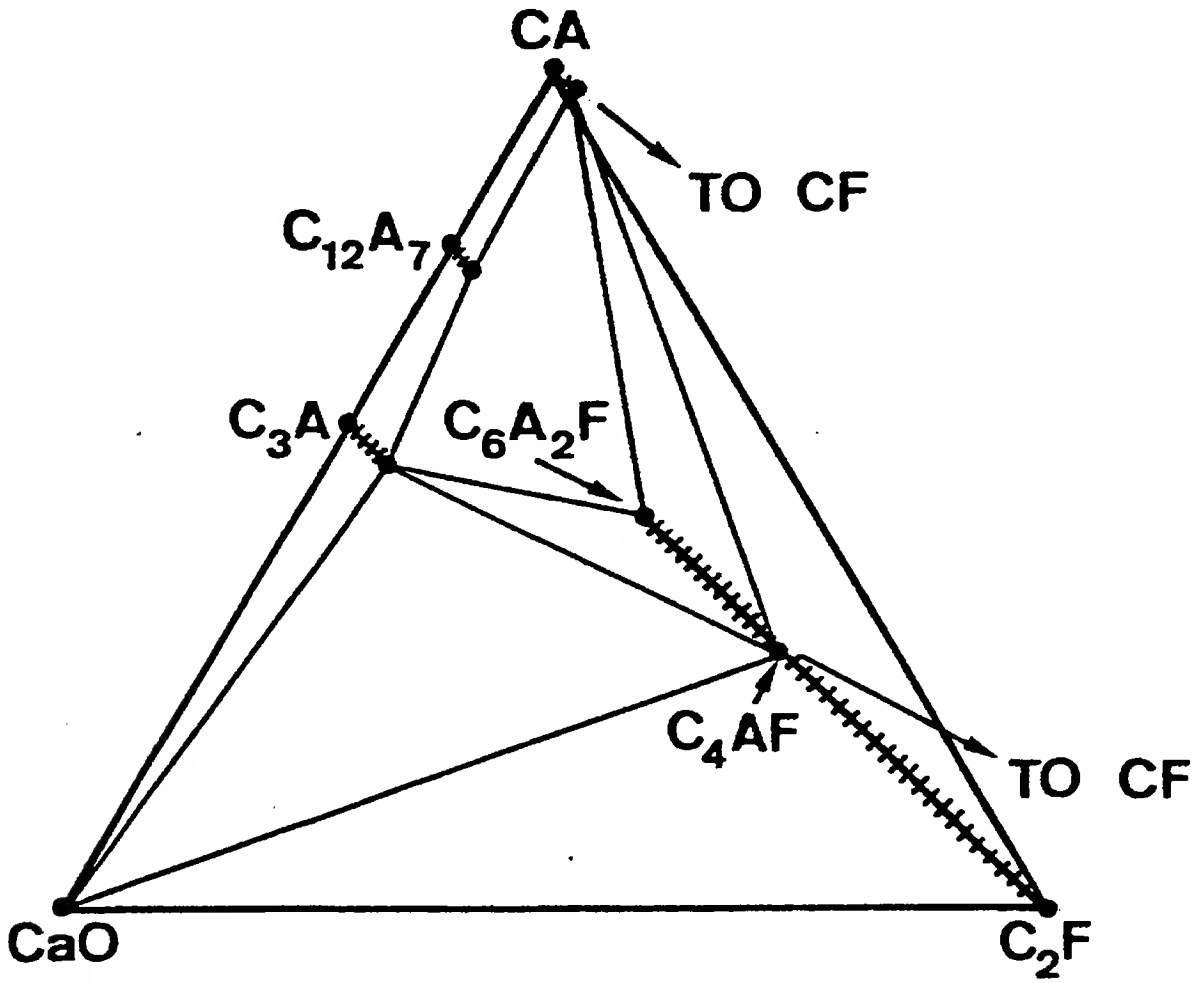
This solid solution series can be clearly seen in Figure 3, running from composition 2 CaO · 1/2 Al_2O_3 · 1/2 Fe_2O_3 to the end member of approximate composition 6 CaO · 2 Al_2O_3 · Fe_2O_3, i.e. C_6A_2F, in which 2 out of 3 Fe^{3+} atoms have been replaced by Al^{3+}. The theoretical end member with 100 % replacement of Fe^{3+} by Al^{3+}, i.e. 'C_2A', is not stable at normal atmospheric pressure, but only at pressures in the order of 15000 bars.

Figure 3 **Phase equilibrium diagram of the portion CaO - 2CaO . Fe_2O_3 - CaO · Al_2O_3 of the CaO · Al_2O_3 · Fe_2O_3 system**



In the already mentioned average clinker the oxides CaO, Al_2O_3, Fe_2O_3 and SiO_2, total 95.6 %, the remaining 4.4 % being principally composed of MgO (1.8 %), the alkalis K_2O and Na_2O (0.9 %) and SO_3 (0.5 %). These **minor oxides** not only are located within the lattices of the C_3S, C_2S, C_3A and C_4AF, (as will be demonstrated later) but also exist in the clinker as free MgO (periclase) and as alkali and alkali/calcium sulfates.

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1.2 The Real Clinker Minerals

We are now acquainted with the clinker phase from the theoretical point of view, but what are the real chemical compositions of the clinker minerals?

Except for periclase (MgO) and free lime (CaO), all clinker minerals are **solid solutions**, i.e. they occur as single crystalline phases which vary in composition within finite limits without the appearance of an additional phase. The 'pure' compounds C_3S, C_2S, C_3A, thus, do not occur in this simple form, but contain a whole series of oxides such as **MgO, Na_2O, K_2O, Fe_2O_3, and Al_2O_3.**

Whereas for the pure phases we use the abbreviations C_3S, C_2S, etc., for the real minerals only the names **alite, belite, aluminate** and **ferrite** should be used.

Until the development of extremely sophisticated analytical techniques (electron microprobe analyzer and energy disperse analyzer), the determination of the maximum quantities of these minor oxides that could be incorporated into the 4 principal clinker minerals was based on laboratory experiments using the pure theoretical phases C_3S, C_2S, C_3A and C_4AF. The quantities actually present, i.e. the true distribution of the oxides, remained unknown. By using the microprobe technique, however, it is now possible to determine the rage of composition of even the individual alite, belite, aluminate and ferrite crystals in any factory produced clinker.

In a recent study (2), based on 28 clinkers from 'Holderbank' Group plants, it was found by electron microprobe analysis that the range of MgO, Fe_2O_3, Al_2O_3, K_2O and Na_2O present in the alite and belite crystals varies from plant to plant (see Table 1 and Figure 4). Moreover, even within one particular clinker grain, the individual alite and belite crystals show varying minor element incorporation.

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Figure 4 **Wt % chemical variation ($\square \pm s$) of MgO, Fe_2O_3, Al_2O_3, K_2O, and Na_2O in alites and belites in various cement plants**

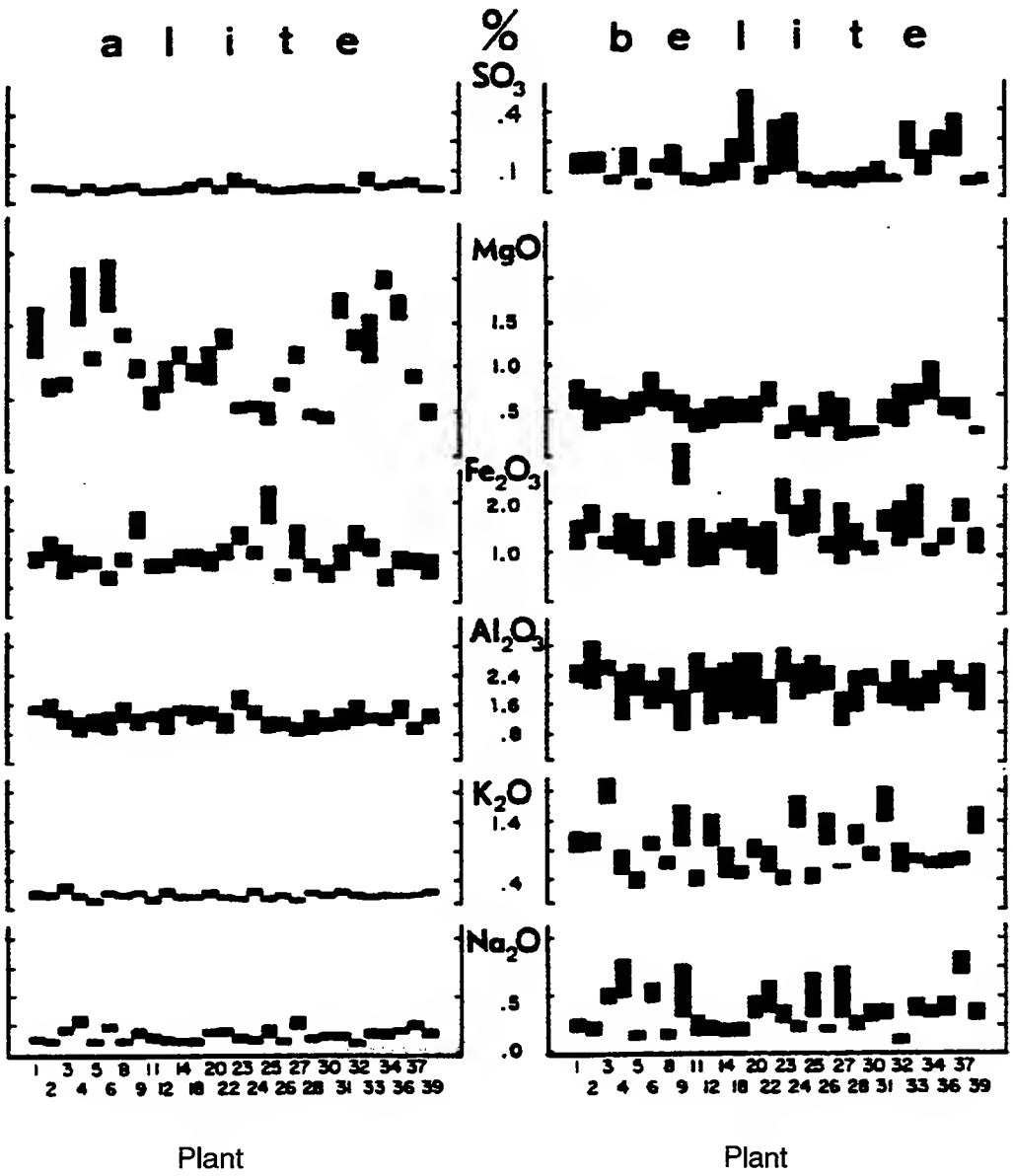

Table 1 Real clinker minerals: chemical variation between different plant clinkers

	Alite			Pure	Belite			Pure	Aluminate			Pure	Ferrite			Pure
	min.	max.	$\bar{x}$	C_3S	min.	max.	$\bar{x}$	C_2S	min.	max.	$\bar{x}$	C_3A	min.	max.	$\bar{x}$	C_4AF
SiO_2	24.4	25.9	24.5	26.31	29.8	32.9	31.3	34.88	3.1	4.3	3.9		1.8	3.1	2.5	20.98
Al_2O_3	1.0	1.8	1.3	-	1.5	2.8	2.2	-	28.7	31.8	30.8	37.74	20.2	24.4	22.5	32.86
Fe_2O_3	0.4	1.7	0.8	-	0.7	2.3	1.2	-	5.2	8.3	6.9	-	20.8	26.7	23.7	46.16
CaO	69.3	72.4	70.6	73.69	62.2	66.4	63.8	65.12	51.5	58.3	54.6	62.26	47.4	49.9	49.1	
MgO	0.5	2.1	1.1	-	0.2	0.8	0.5	-	0.8	1.5	1.1	-	1.9	3.2	2.6	
K_2O	<0.1	0.3	0.1	-	0.3	1.9	0.	-	0.3	4.6	2.0	-	0.1	0.3	0.2	
Na_2O	<0.1	0.3	0.1	-	0.1	0.8	0.3	-	0.2	0.8	0.8	-	0.1	0.2	0.1	

1.3 Polymorphic Modifications of the Clinker Minerals

If the clinker minerals are studied in detail, we find out that not only does the chemical composition vary, but the same clinker phase also exists in different polymorphic forms, which can exhibit different physical properties. Determining the various polymorphic forms present in clinker has not been an easy task so that it was first necessary to study the polymorphism of the pure phases C_3S, C_2S, C_3A and C_4AF in order to obtain sufficient information and experience to be able to advance to alite, belite, aluminate and ferrite.

1.3.1 C_3S and Alite

Between room temperature and 1100 °C **pure C_3S** displays 6 polymorphic modifications, namely:

- 3 triclinic T_I, T_{II}, T_{III}
- 2 monoclinic M_I and M_{II}
- 1 rhombohedral R_I

From DTA analysis it is known that the transitions take place as follows:

$$T_I \overset{600°}{\leftrightarrow} T_{II} \overset{920°}{\leftrightarrow} T_{III} \overset{980°}{\leftrightarrow} M_I \overset{990°}{\leftrightarrow} M_{II} \overset{1050°}{\leftrightarrow} R$$

and because the changes involve only a light displacement within the lattice structure (i.e. no breakage of the strong chemical bonds), they are therefore easily reversible. These transformations consequently cannot be prevented, so that in pure C_3S, the high temperature modifications can only be stabilized by quenching.

When passing to **alite**, the real mineral in industrial clinker, incorporating Al_2O_3,, Fe_2O_3, MgO, K_2O, Na_2O, and TiO_2 within the crystal lattice, it has been found to exist in the following modifications:

- trigonal R
- monoclinic M_I
- triclinic T_{II}

The incorporation of foreign atoms thus ensures the stabilization of the high temperature monoclinic and trigonal forms, and in some clinker even both of these modifications can be simultaneously present. Because the various modifications are crystallographically all very similar, the various polymorphic forms of alite in cement clinker are considered as having essentially the same hydraulic activity. Observed differences in activity seem to be related to structural faults in the crystal as a result of incorporation of elements such as Al, Fe, Mg, etc.

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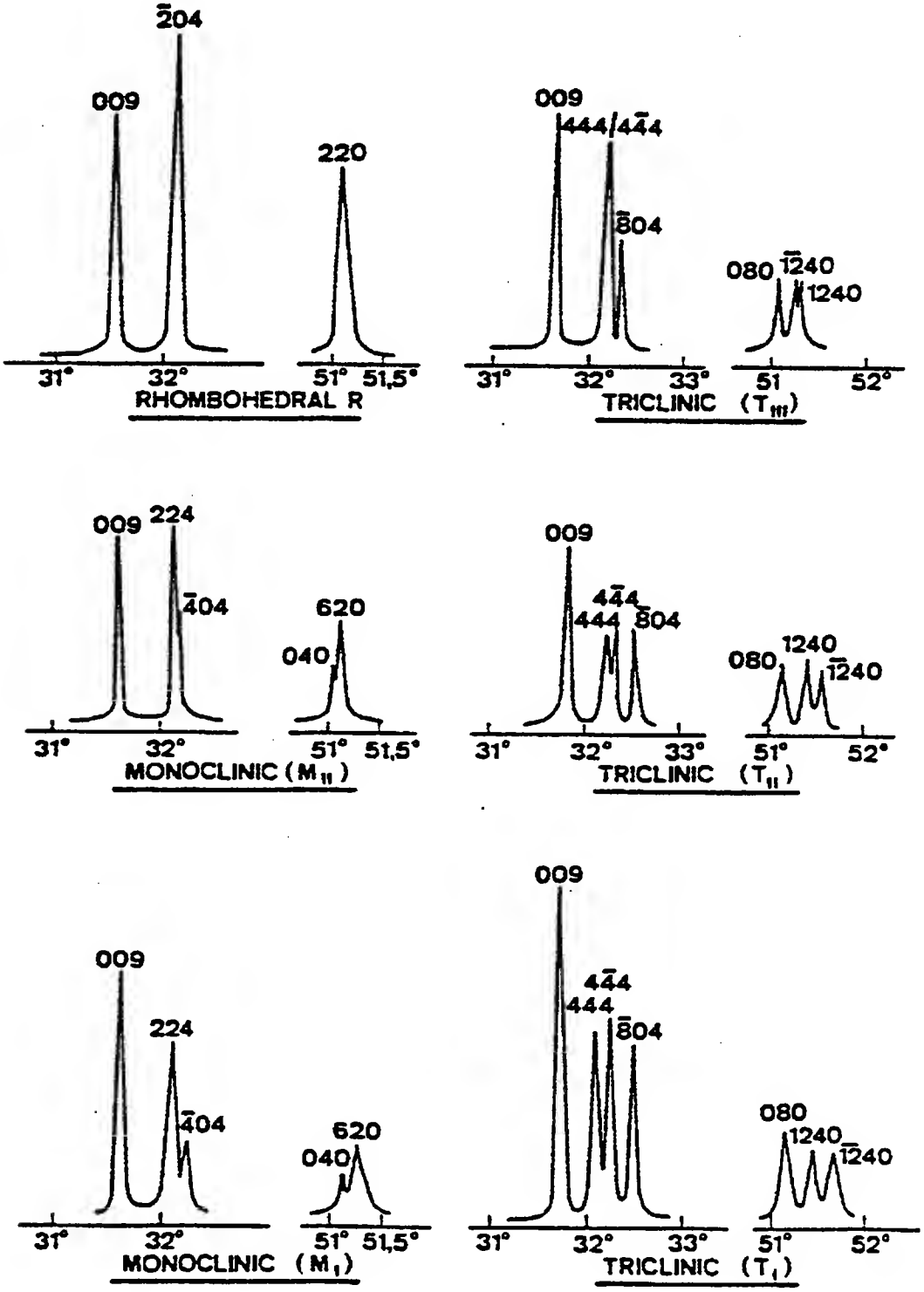
A differentiation between the various polymorphic forms is partially possible by both microscopy and by x-ray diffraction as shown in Figure 5 using sophisticated apparatus and analyzing equipment.

Figure 5 X-ray diffraction patterns of the various C_3S polymorphs



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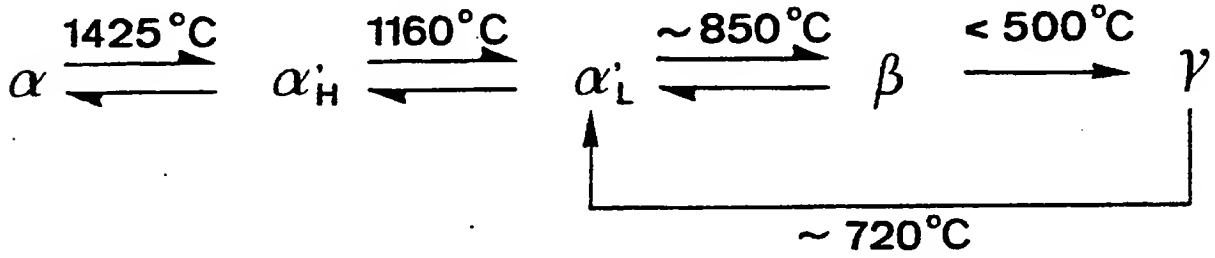
1.3.2 C_2S and Belite

Pure C_2S exhibits 5 crystalline modifications between room temperature and 1500°C, namely the form α, β, α'_L, α'_H and α. The stability range of these is displayed in a simplified form in Figure 6.

Figure 6 Stability ranges and polymorphic sequences of C_2S



The changes $\beta \rightarrow \alpha$ and $\alpha \rightarrow \alpha'_L$ are irreversible and slow as they involve breaking all the primary chemical bonds. These structural changes are accompanied by a considerable volume change (13 %). The β-belite is considered to have good hydraulicity, while the α-form is hydraulically only little active. Alkalis play an important role: in the presence of K_2O and Na_2O and α' and an α-form develop, which have a higher activity than the β-form.

1.3.3 C_3A and Aluminate

Pure C_3A is a definite compound with a cubic lattice, showing no polymorphic transformations. When increasing quantities of Na_2O, they are incorporated within the lattice, forming a solid solution series incorporating up to 5.7 % Na_2O, orthorhombic (O), monoclinic (M) and tetragonal (Tg) modifications occur; their stability relations are summarized in Figure 7.

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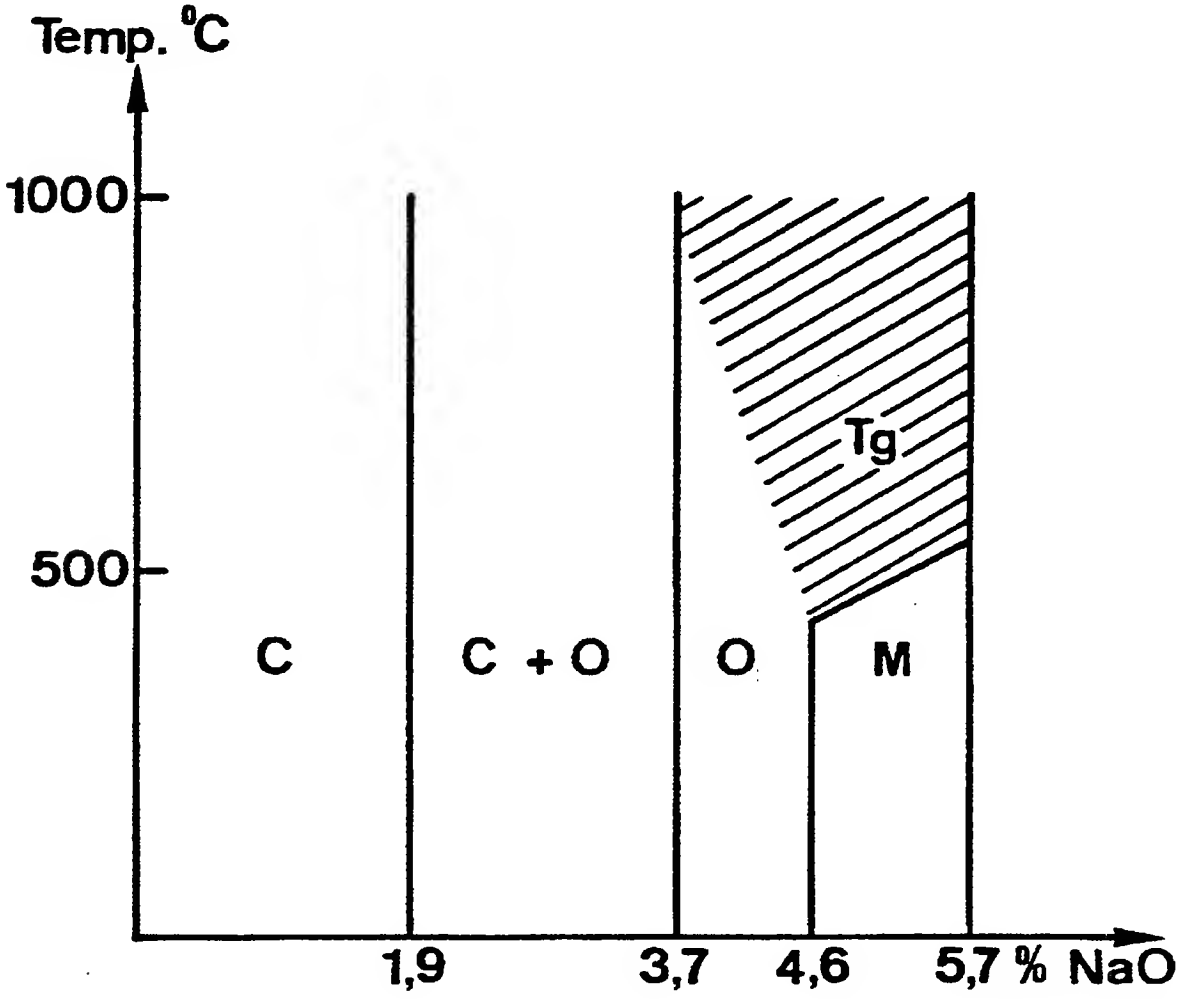
Figure 7 Phase equilibrium diagram for Na_2O - C_3A solid solutions



The **aluminate** phase in clinker exists in the cubic (C), orthorhombic (O) and tetragonal (Tg) forms. Two forms, e.g. O + C or O + Tg, can coexist. Up to date, the monoclinic (M) formation has not been observed. There is some evidence to suggest that the various forms differ in their reactivities.

1.3.4 C_4AF and Ferrite

Pure C_4AF is not a definite compound but a member of the continuous solid solution series extending between 2 CaO · Fe_2O_3 and 2 CaO · Al_2O_3 , the latter end-member being stable only at high pressure, up to approx. 66 % Fe_2O_3 in 2CaO · Fe_2O_3 can be replaced by Al_2O_3. The member of 4 CaO · Al_2O_3 · Fe_2O_3 (C_4AF) corresponds to the 50 % replacement.

The **ferrite** phase in Portland cement varies chemically between C_6A_2F and C_6AF_2 (solid solution series) and is orthorhombic like the pure C_2F. As yet no other modification has been reported. The ferrite phase is considered to be only slightly hydraulically active.

For a detailed x-ray diffraction investigation of aluminate and ferrite in clinker, the phases can be concentrated dissolution of alite and belite using salicylic acid.

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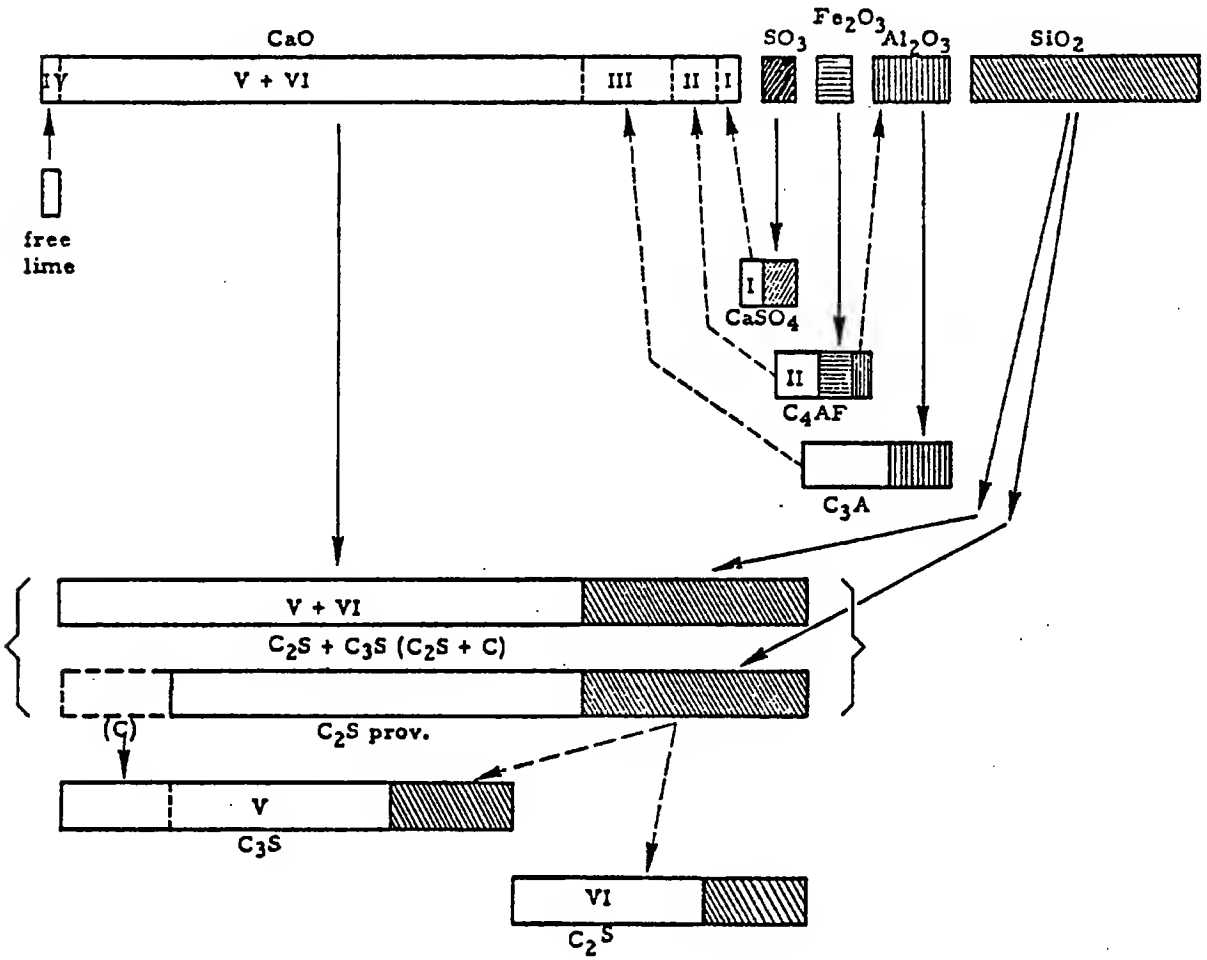
1.4 Quantitative Clinker Mineralogy - Potential and Reality

1.4.1 Calculation of Potential Clinker Composition

For almost 50 years the **potential** clinker mineralogy has been calculated by the procedure first proposed by Bogue. This method assumes that:

- chemical equilibrium has been attained
- Fe_2O_3 reacts with Al_2O_3 and CaO to form **C_4AF**
- Al_2O_3 remaining from combination as C_4AF reacts with CaO to form **C_3A**
- SO_3 combines with CaO to formation $CaSO_4$
- CaO remaining from the above combinations reacts with the silica. **C_3S** is the first formed
- with the SiO_2, and any CaO left over reacts with the C_2S to form **C_3S**. If CaO remains after converting all of the C_2S to C_3S it remains as uncombined CaO.
- MgO remains essentially uncombined.

Figure 7A Mineralogical composition derived from chemical analysis (Graphical representation of the steps in calculation)



For the purposes of calculation, the 'CaO' and 'SiO_2' values used are given.as

'CaO' = CaO_{total} - CaO_{free}

'SiO_2' = $SiO_{2\ total}$ - insoluble residue

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The calculation does not take into account the influence of the minor elements: alkalis, Mg, Ti, Mn, Cr, P, etc., and the solid solution of the principal oxides: CaO, SiO_2, Al_2O_3 and Fe_2O_3 within the alite, belite, aluminate and ferrite phases. As we have already seen, the composition of the 4 principal phases is not C_3S, C_2S, C_3A and C_4AF. **It is therefore not surprising that the potential and real clinker compositions never agree.** This situation is further aggravated by the non-attainment of equilibrium during clinker production. In addition it is now known that SO_3 in clinker is mainly combined with the alkalis and only to a slight extent with CaO. The currently used Bogue formula for clinker does, therefore, not consider the influence of SO_3.

1.4.2 Differences between Potential and Real Phase Composition

The most practiced method at present to determine the **real** mineralogical composition is by means of microscopic point counting, complemented by the use of crystallized aluminate and ferrite. The results obtained by these methods, as is to be expected, differ from those obtained during the Bogue calculation procedure.

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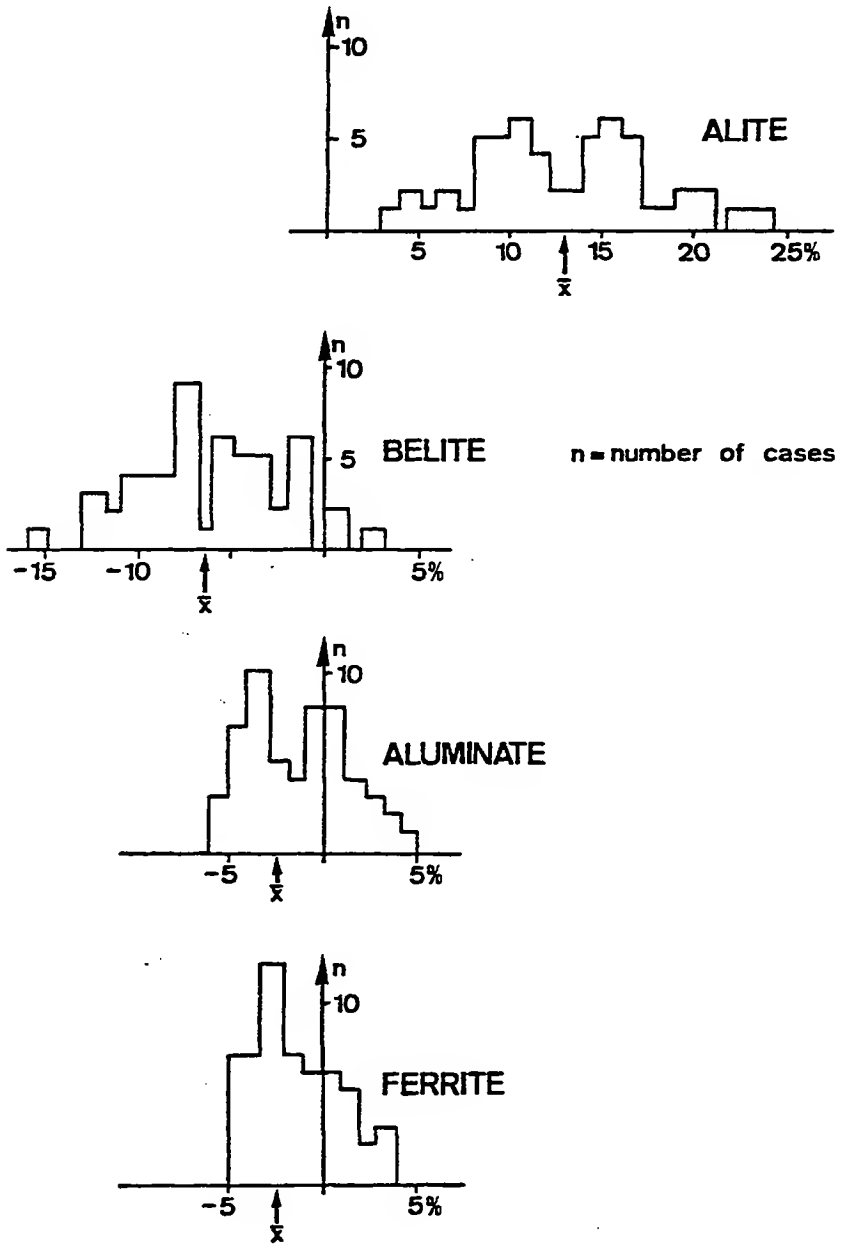
How large are the differences between the potential and real mineralogical composition of the clinker?

This can best be seen in Figure 8 based on the results of measurements on over 50 different plant clinkers (3).

It can be seen that:

- **Alites** as calculated are always **too low**, ranging from 3 - 24 %, on average 13 %
- **Belites** are mostly calculated **too high**, on average 6 %
- **Aluminate and ferrite** as calculated can either be too high or too low, but in general both phases are calculated approx. 2 % too high.

Figure 8 **Difference between Actual and Potential Phase Composition (experimental minus Bogue)**



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1.4.3 Modifications of Bogue's Formula

A useful improvement in the correlation between potential and real mineralogical composition can be obtained if slight corrections are made to the Bogue calculated values. On the basis of data obtained (Figure 8), the following formulae have been proposed:

$C_3S_{corr.}$	=	$C_3S_{Bogue} + 4.0\ MgO_{clinker} + 5.5\ K_2O_{clinker}$
$C_2S_{corr.}$	=	$C_2S_{Bogue} - 1.5\ MgO_{clinker} - 2.2\ K_2O_{clinker}$
$C_3A_{corr.}$	=	$C_3A_{Bogue} + 7.8\ Na_2O + 1.5\ AR - 2.1\ SO_3 - 5$
$C_4AF_{corr.}$	=	$C_4AF_{Bog} - 6.5\ Na_2O - 1\text{-}7\ AR + 5.0\ Mn_2O_3 + 3$

These corrected formulae are valid under the following conditions:

MgO	< 4 %	Na_2O	< 0.5 %	Mn_2O_3	< 0.3 %
K_2O	< 1 %	SO_3	< 1.0 %	AR	> 1.0

In the case of AR < 1.0 uncorrected Bogue's formulae are more precise to calculate C_3A and C_4AF.

The improvement achieved by using these modified formulae can be seen in Figures 9 and 10. These corrections improve Bogue's original concept in that the clinker phases can be more accurately calculated, taking into consideration the incorporation of minor elements without having to undertake time consuming mineralogical analyses.

For pure routine quality control purposes in any individual plant, the use of the original Bogue equations is still acceptable as long as no significant change in raw materials occurs. For comparison of calculated values between different plants or with directly measured clinker mineralogy (microscopy, x-ray diffraction) the use of the modified formulae is recommended.

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Figure 9 Alite and Belite

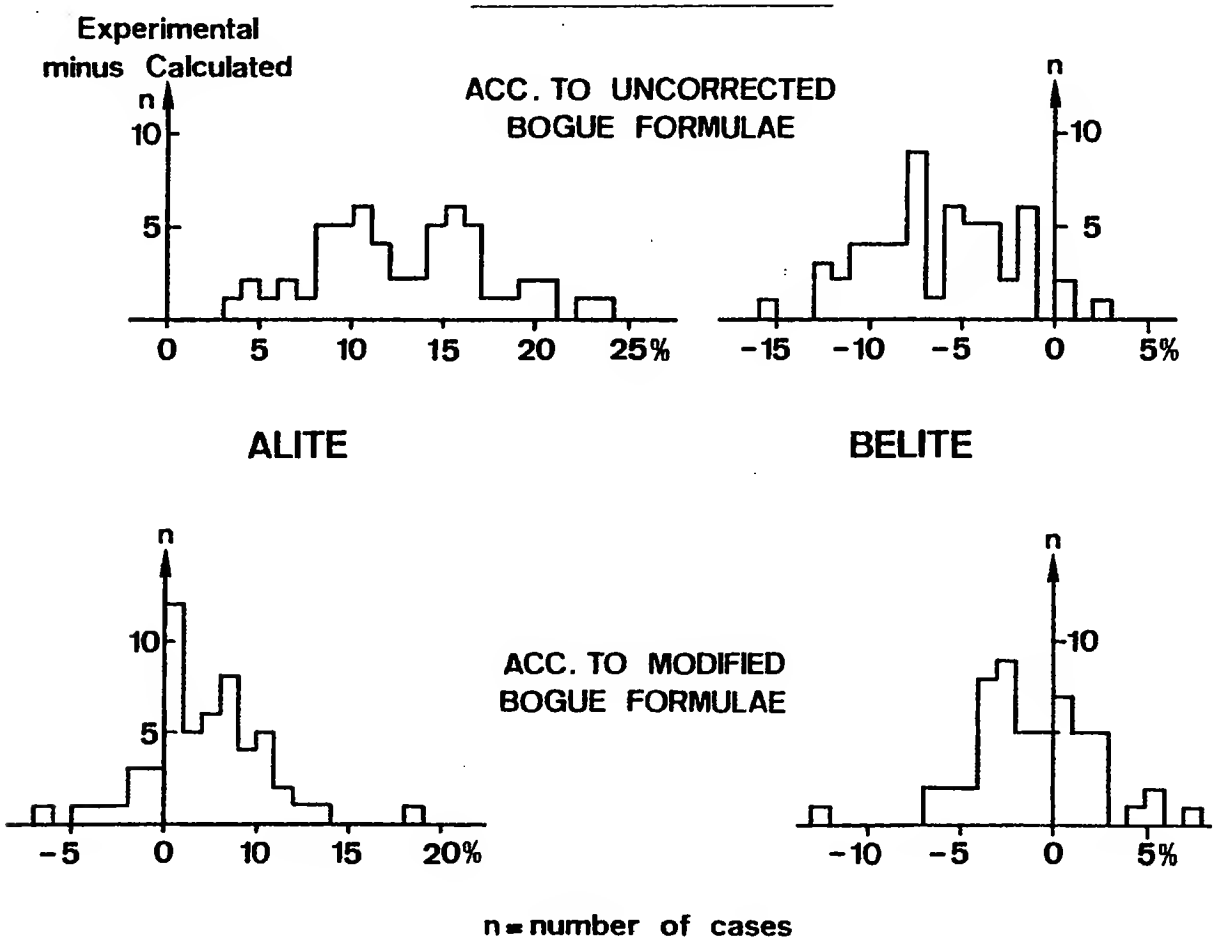

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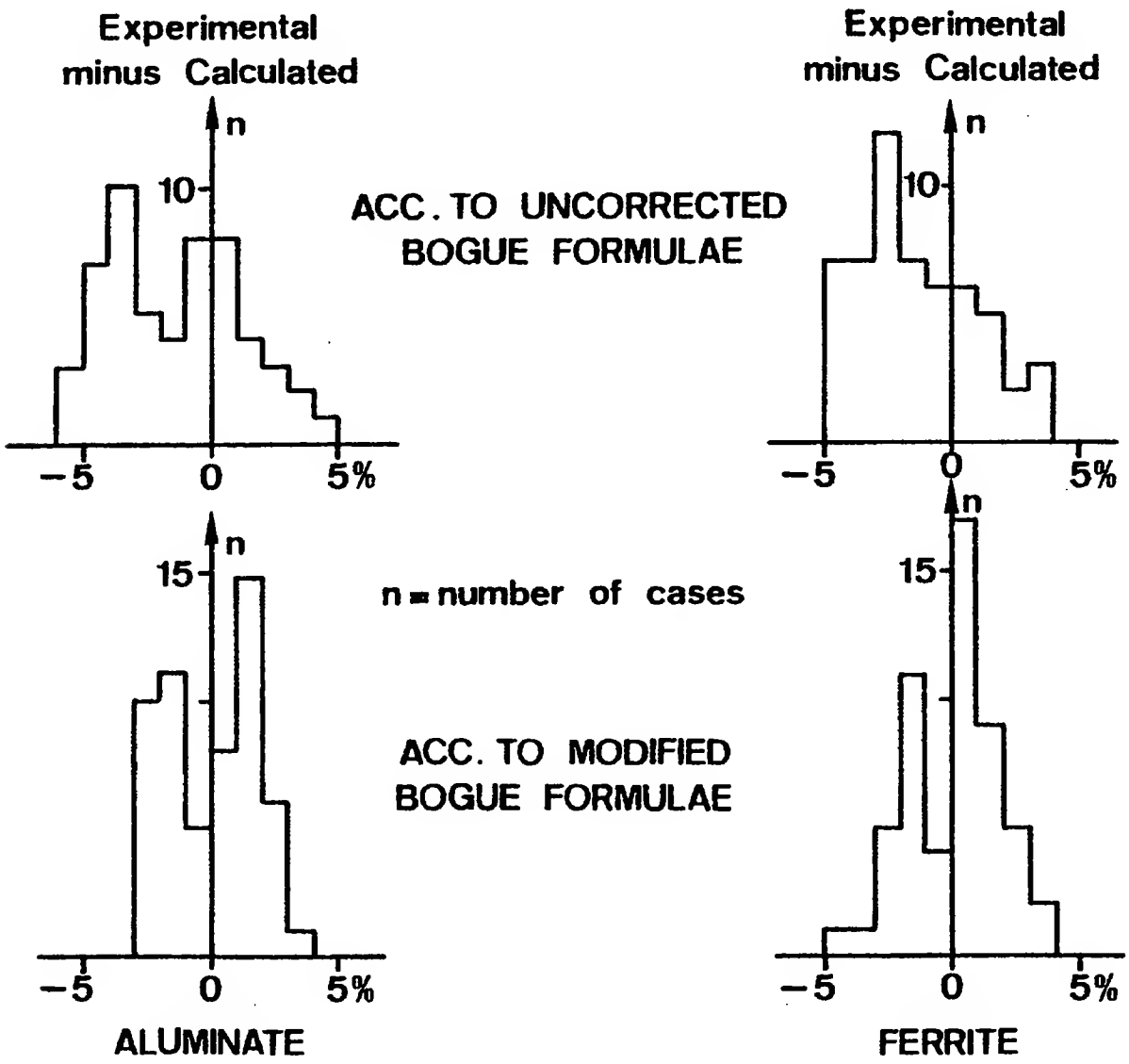
Figure 10 Aluminate and Ferrite



2. CLINKER MICROSTRUCTURE

2.1 Microscopical Appearance of the clinker minerals

2.1.1 Introduction

In the foregoing chapter we have discussed the principles of clinker formation, its chemistry and mineralogy. These parameters can only account for part of the clinker and of the resulting cement properties. The microstructure or fabric of a clinker is also decisive when considering phenomena such as clinker grinding, cement hydration and hardening characteristics.

In this chapter only an introduction to the microscopical appearance of industrial clinker will be given. As this is a complex topic, only the basic principles will be discussed. For further information reference should be made to any of the reference works on this subject (4, 5).

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2.1.2 General

The appearance of a crystalline aggregate is generally referred to as fabric, which is defined as the whole of all geometric features or properties of an aggregate. Three groups of geometric features can be discerned: **phase association, texture and structure.**

Phase association is the information group which defines what phases constitute the fabric and in what quantities they exist. **Texture** includes the information concerning size, shape and arrangement of the crystals and also the intergrowth relationship of the individual phases. Finally, **structure** indicates to what extent a fabric is anisotropic, whether some crystal phases are oriented in a certain direction and what type of space-filling, i.e. porosity, occurs.

With these geometric features, we can describe the fabric of a material. This description explains **how** and out of **what** a piece of clinker is build up. When completely differently manufactured clinkers are investigated and compared, one realizes to what degree the 'what' and 'how' differ, thereby resulting in the most varied clinker fabrics.

The fact that clinker does not always have the same appearance leads to the important question of why one particular fabric and not another is formed. It is overall accepted that the genesis of a crystalline material influences to a great extent its appearance or fabric. The genetic factors of the clinker formation process are identical to those conditions on which the industrial manufacture of clinker is based, this being the interaction of parameters beginning with the raw material, passing through the sintering process and ending with clinker storage.

2.1.3 Microscopical Appearance of Clinker Phases

2.1.3.1 Alite

External Form

Alite, the principal constituent of cement clinker, occurs most frequently as idiomorphic crystals, i.e. as well-defined, sharp-edged crystals. The characteristic form, as can be seen in Plate 1, are compact tabular crystals, often exhibiting a hexagonal section. Alite crystals can also, however, be found in a form whose outlines are dissected by indentations and protrusions, and whose hexagonal shape is barely discernible. Such crystals are termed xenomorphic alite and can be seen in Plate 2. Lath-shaped alite crystals have also been reported.

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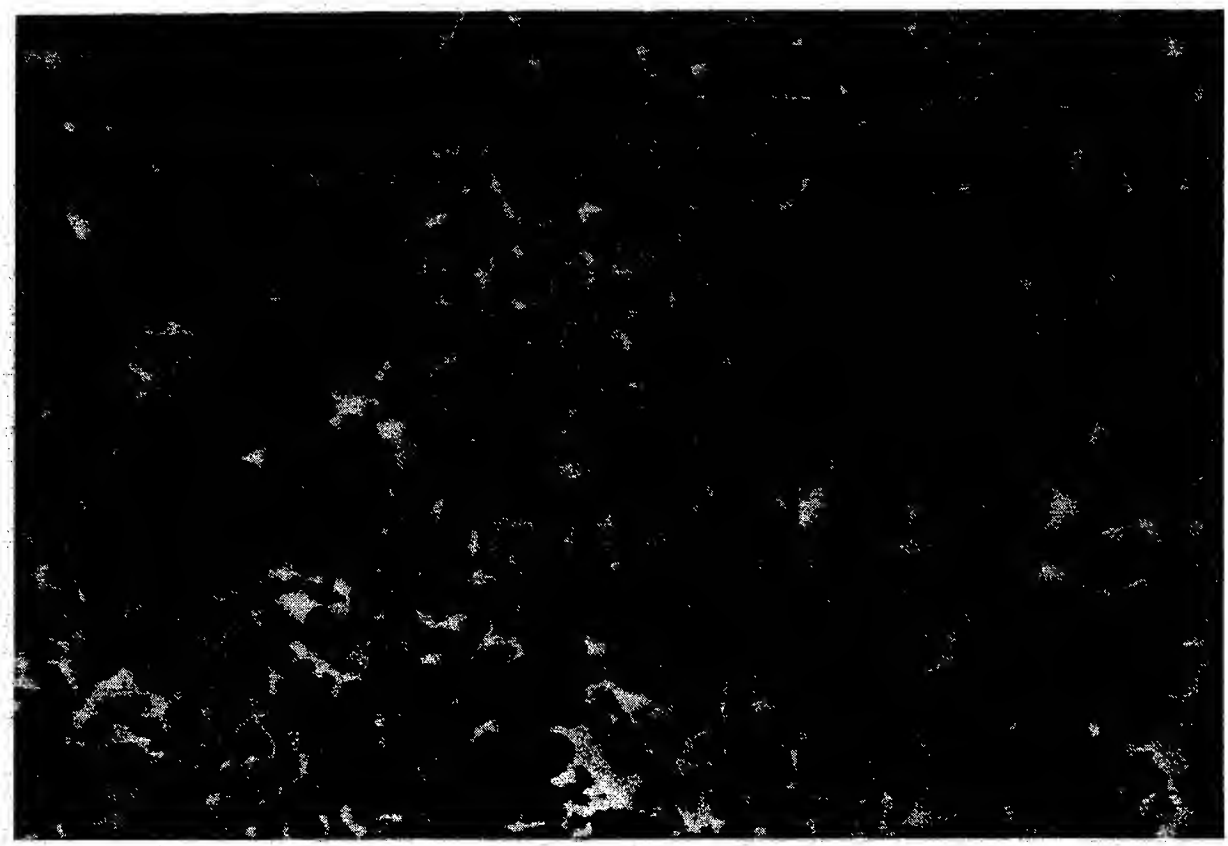
Plate 1 **Idiomorphic alite (brown). 250 x**

Plate 2 **Xenomorphic alite. 250 x**



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Internal Features

A feature commonly observed within alite crystals is the inclusion of small rounded belite crystals, these having been formed before the alite, as has previously been indicated in Paper 10. An example of this phenomenon can be seen in Plate 3.

Plate 3 Alite exhibiting belite (blue) inclusions. 250 x

Other features can also be observed within alite crystals and include the presence of single and polysynthetic twinning, as can be seen in the excellent plates in Ref. 4 (p. 11, Fig. 4).

At the highest sintering temperature, alite always exhibits its highest symmetry, i.e. trigonal (R) form, but on cooling to normal temperature, passes through several polymorphic stages, and so in industrial clinker appears mostly in the monoclinic (M) an triclinic (T) forms.**The presence of simple twinning is indicative of crystals having passed through the trigonal (R) → monoclinic (M) polymorphic transition. The presence of polysynthetic twinning is associated with the change from the monoclinic to the lowest symmetry, i.e. triclinic (T) form.** Differentiation between the individual monoclinic or triclinic forms is not possible by microscopy but requires the use of x-ray diffraction (see Figure 5) or DTA techniques. The optical distinction of the various polymorphic changes is made possible by the different etching behavior of the various crystal modifications.

Another internal feature of alite crystals is the presence of zones having different etching properties, i.e. a zonar structure. These zones are associated with the selective inclusion of different ions during the formation process. Examples of the various forms of twinning and zonar structure are to be found in (4).

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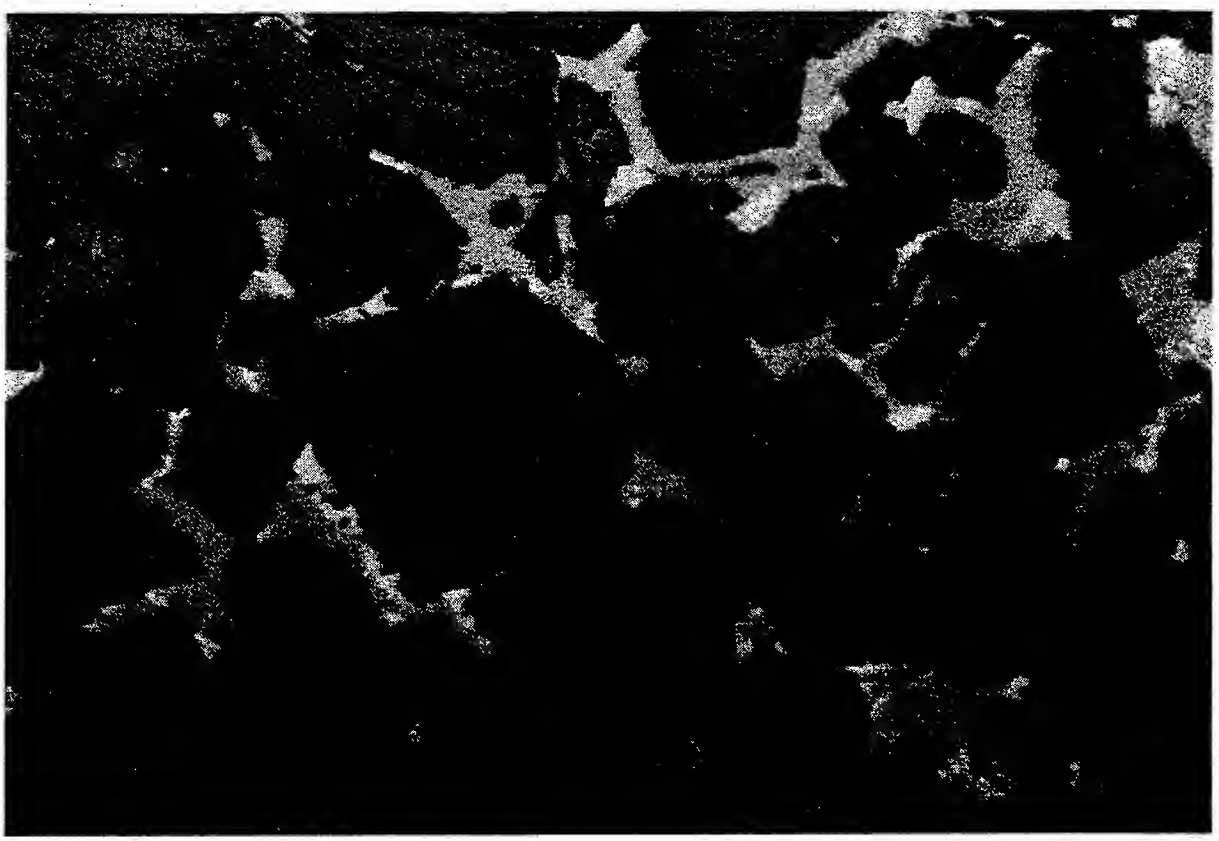
Alite decomposition

As has already been mentioned, pure C_3S is unstable below 1250 °C and tends do decompose into belite and sub-microscopic free lime. Although **this decomposition is strongly suppressed in alite due to the incorporation of foreign ions**, the presence of bivalent iron i.e. Fe^{2+} and CaF_2 opposes this stabilization effect. An example of partially decomposed alite crystals can be seen in Plate 4.

Plate 4 Alite decomposition into belite and free lime. 250x



Scanning electron microscope (SEM) photographs of alite, (see Plates 21 - 23) also indicate the idiomorphic formation shown in Plate 1.

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2.1.3.2 Belite

External Form

The round crystal form (Plate 5) is the most characteristic shape of belite, although in rare cases crystals showing a tendency to a hexagonal form are also observed (Plate 6).

Plate 5 Typical rounded belite 250x

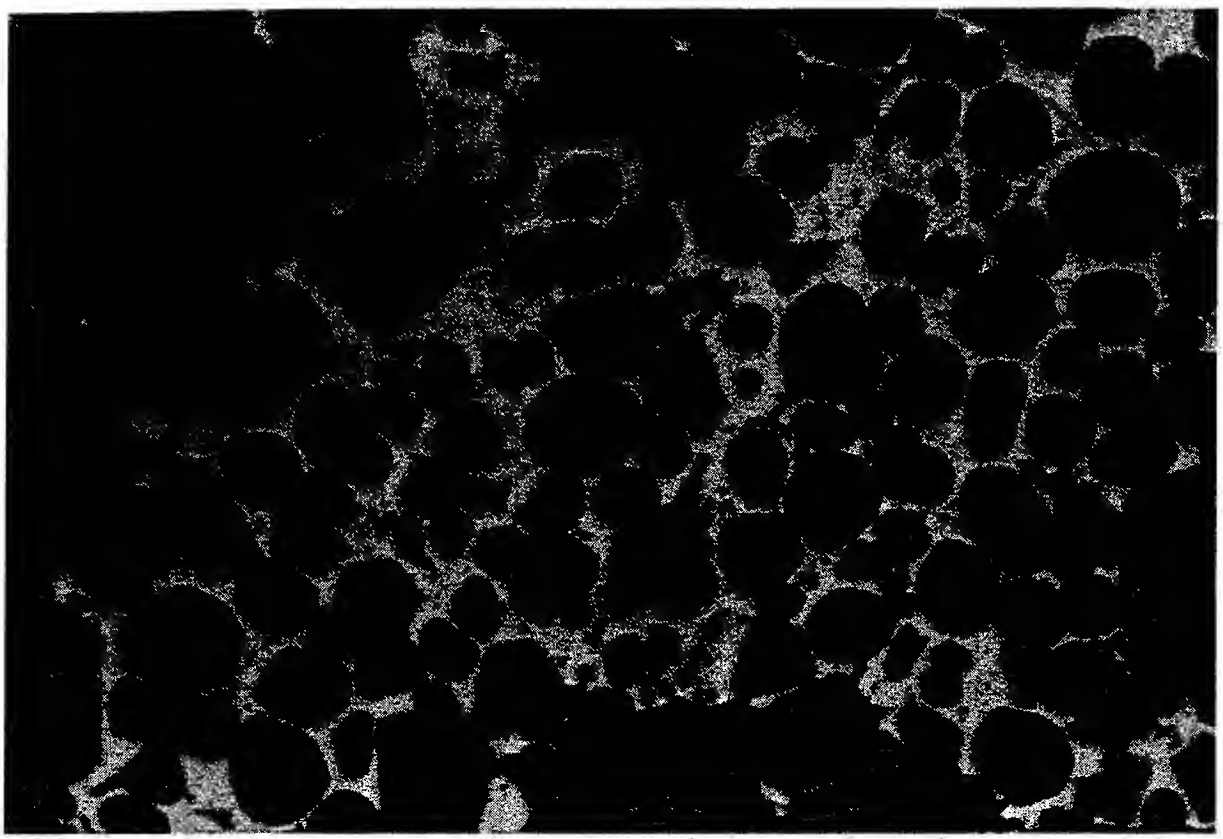

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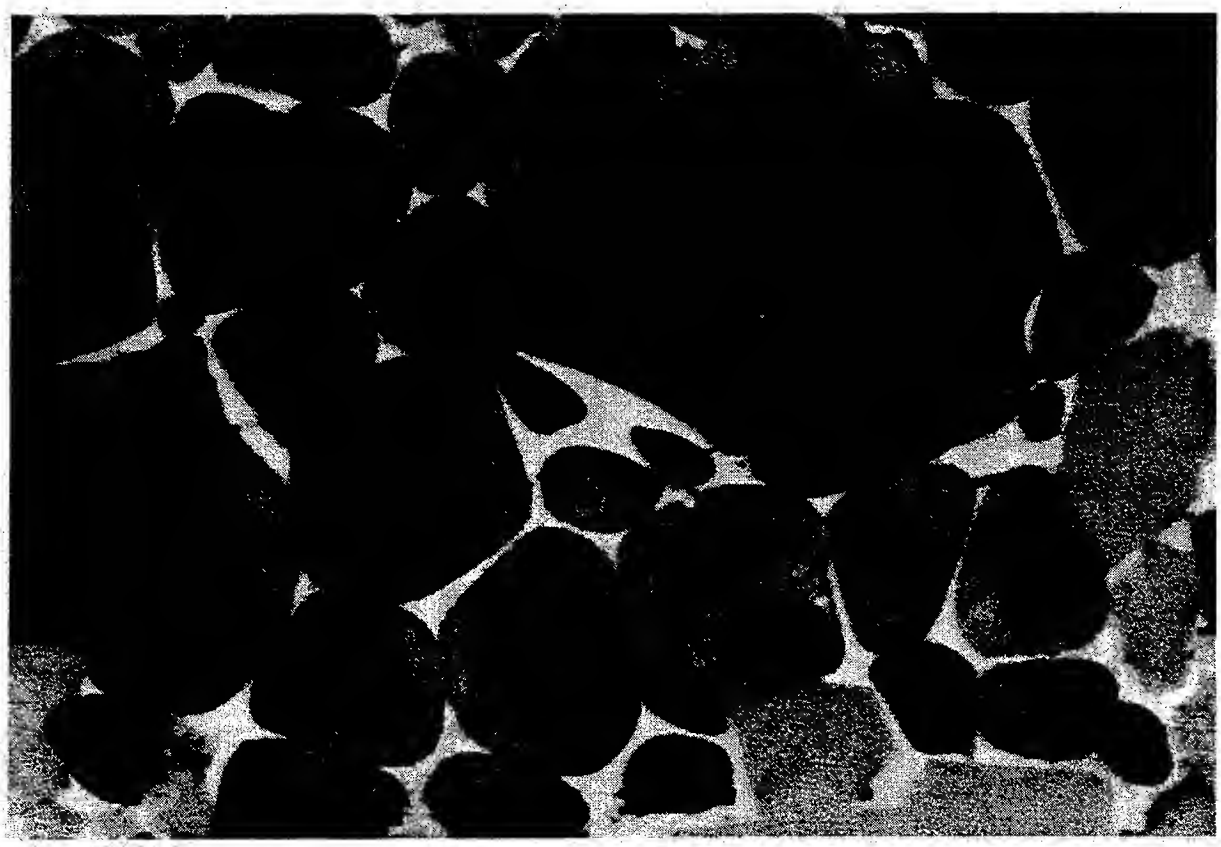
Plate 6 **Belite with complex lamellar structure due to $\alpha \rightarrow \alpha'$ transition. 640x**



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Internal Features

Just like in the case of alite, the passing through the $\alpha \rightarrow \alpha'$, $\alpha' \rightarrow \beta$ and $\beta \rightarrow \alpha$ polymorphic transformations can also be microscopically observed. The first mentioned transition realties in belite crystals exhibiting a **cross lamella structure** as shown in Plate 6, while after undergoing the $\alpha \rightarrow \beta$ transition, **polysynthetic twinning** occurs.

Cross lamella and polysynthetic twinning can sometimes be observed in the same crystal.

The transition from $\beta \rightarrow \alpha$ belt, accompanied by an appreciable change of volume can be observed microscopically.

α - belite is undesirable in clinker because the strong volume expansion breaks up the clinker granules, producing much dust, and also because it is not hydraulically active.

Excellent plates showing polymorphic transformation phenomena are to be found in (4).

Secondary Belite

The liquid in contact with alite and belite at clinking temperature contains approximately 6 % SiO_2 as well as CaO, Al_2O_3 and Fe_2O_3. During cooling, immediately after the maximum temperature, this SiO_2 comes out of solution in the form of small secondary belite crystals as sown in Plate 7. Such belite can exhibit a drop-like or dendritic structure.

Plate 7 Secondary crystallization of belite 250 x

Scanning electron microscope photographs of belite seen in Plate 22 also confirm the rounded shape of belite observed in polished sections.

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2.1.3.3 Uncombined Lime

This is more commonly known as **free lime**, and always appears as rounded spherical crystals (Plate 8), even when it is not constricted by other crystalline phases. When viewed at higher magnifications, e.g. 10,000 through scanning electron microscopy as shown in Plate 24, it can be seen that the rounded free lime grains do have planar surfaces. It is invariably found along with alite, but not in contact with belite, as in the latter case the two would react to form alite.

Plate 8 Spherical free lime (off-white) 250 x

Uncombined lime should not be confused with the micro-crystalline secondary free lime, resulting form the decomposition of alite.

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2.1.3.4 Matrix

The principal clinker minerals alite and belite are surrounded at clinking temperature by a liquid phase (matrix). Upon cooling the latter crystallizes to yield principally aluminate and ferrite but, depending upon its composition, can also yield periclase and sulfate containing minerals.

2.1.3.5 Aluminate

The form of the aluminate phase depends both upon the cooling rate and the alkali content of the liquid phase. When the cooling rate is slow, well-separated idiomorphic aluminate crystals can be observed (and consequently also ferrite crystals), as can be seen in Plate 9. This type of aluminate is the normal or cubic form, containing insufficient incorporated foreign ions (alkalis) to change its symmetry to one of the lower symmetry types.

Plate 9 **Differentiated aluminate (gray) and ferrite (white) caused by slow cooling. 640 x**

When the liquid phase is rich on alkalis, an aluminate solid solution containing both K_2O and Na_2O crystallizes in the elongated lath from shown in Plate 10, often termed 'alkali aluminate'.

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Plate 10 Alkali aluminate with lamellar habit. 570 x

This form, as has been reported earlier, may have orthorhombic (O) or tetragonal (Tg) crystal symmetry, requiring x-ray diffraction to differentiate between the two modifications.

However, when the clinker has been rapidly cooled, the aluminate and ferrite are finely grained and closely intergrown as shown in Plate 11.

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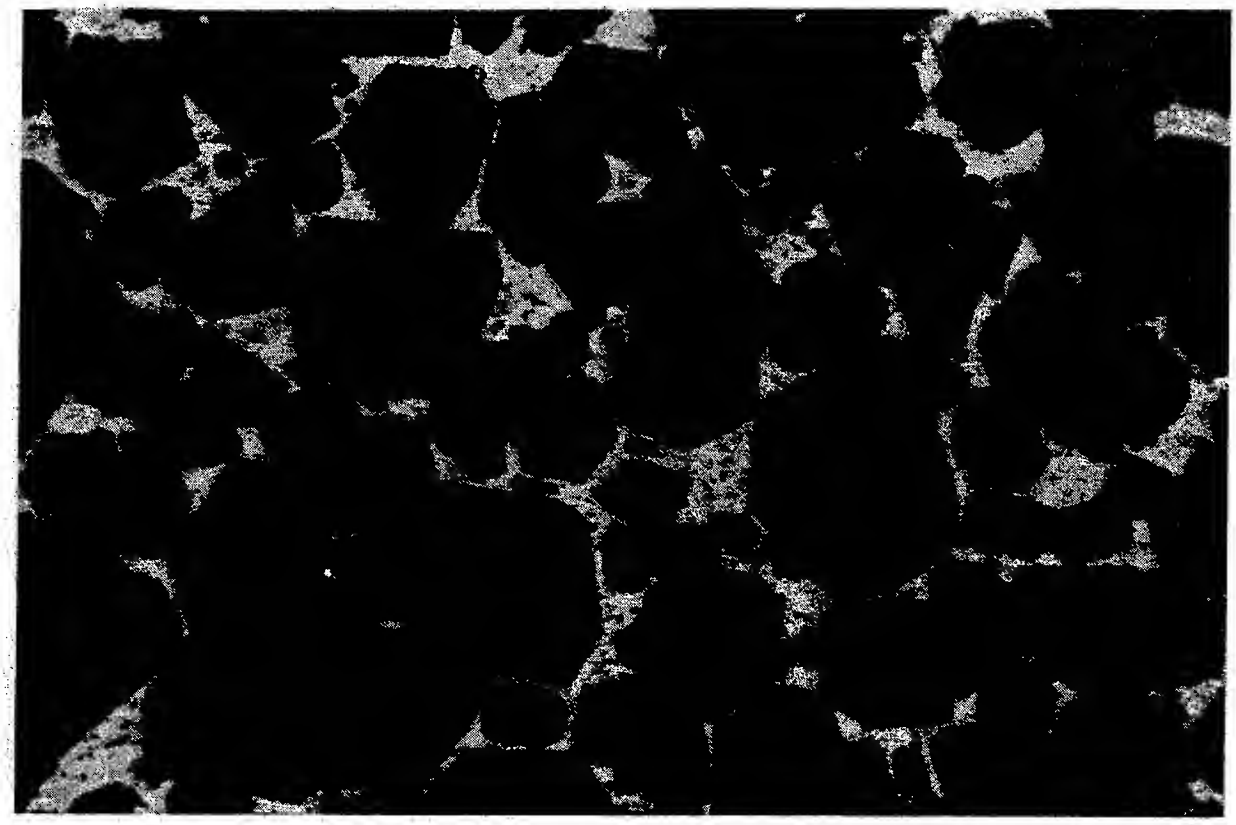
Plate 11 Finely grained aluminate and ferrite due to rapid cooling. 250 x



Although the individual crystals are often so fine that they cannot be optically distinguished, x-ray diffraction confirms that they are present in a crystalline and not amorphous form.

2.1.3.6 Ferrite

Just like the aluminate, the external form of the ferrite, as can be seen in Plate 9 and 11, depends upon the cooling rate. The composition of this phase varies, due to solid solution, around the ratio C_4AF. The actual value cannot be determined by microscopy but x-ray diffraction gives an indication of their composition.

2.1.3.7 Periclase

Periclase occurs only in clinker rich in MgO as an independent phase. In general two types can be distinguished, one idiomorphic with polygonal sections, and one dentritic. Periclase is very easily identified under the microscope due to its resistance to grinding and etching.

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Plate 12 Idiomorphic periclase (gray). 640 x

2.1.3.8 Sulfate containing phases

The last phases to crystallize, long after the aluminate and ferrite have solidified, are the alkali sulfates. These have been identified as being of two separate types, namely a simple alkali sulfate of composition K_2SO_4, and a calcium potassium sulfate of composition 2 $CaSO_4$, both being able to incorporate small amounts of sodium. They are often found around the edges of pores. Micrographs showing these two types of sulfates can be seen in Plates 13 and 14.

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Plate 13 Alkali sulfate (black, angular) 250 x

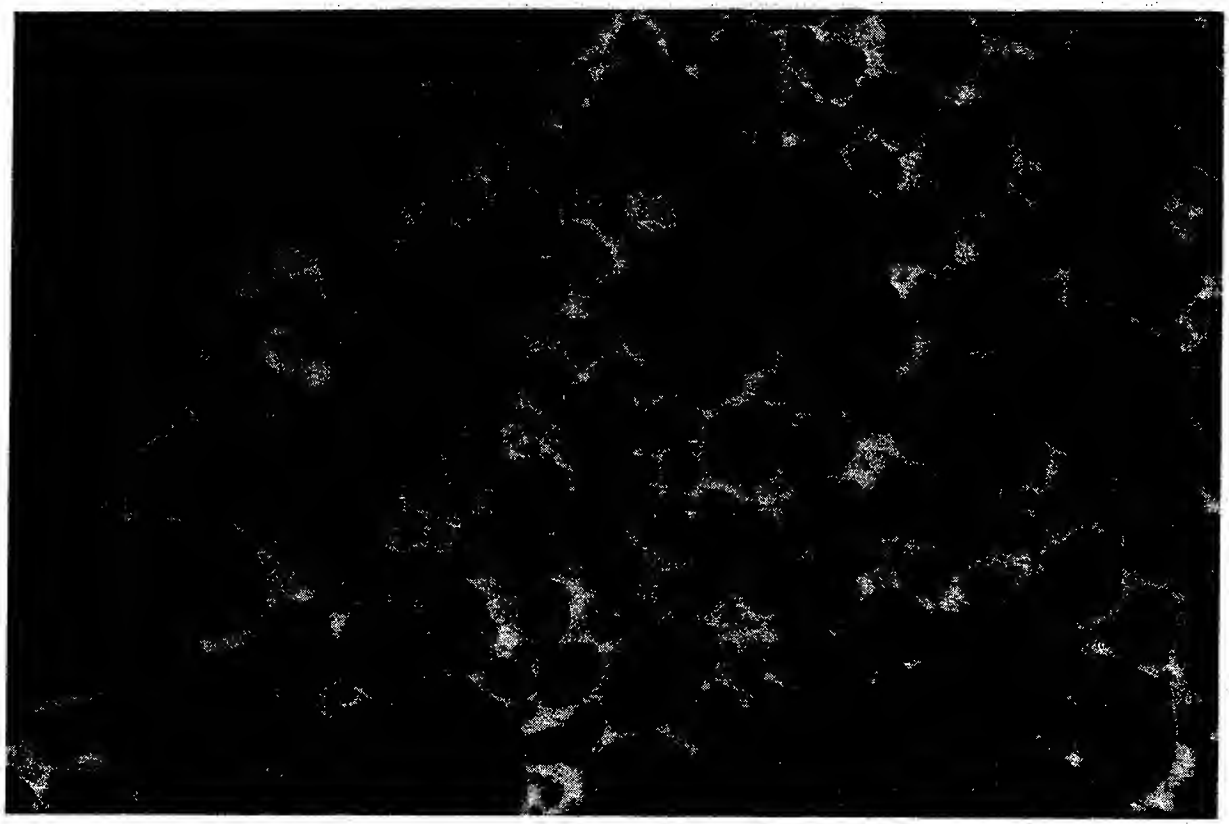
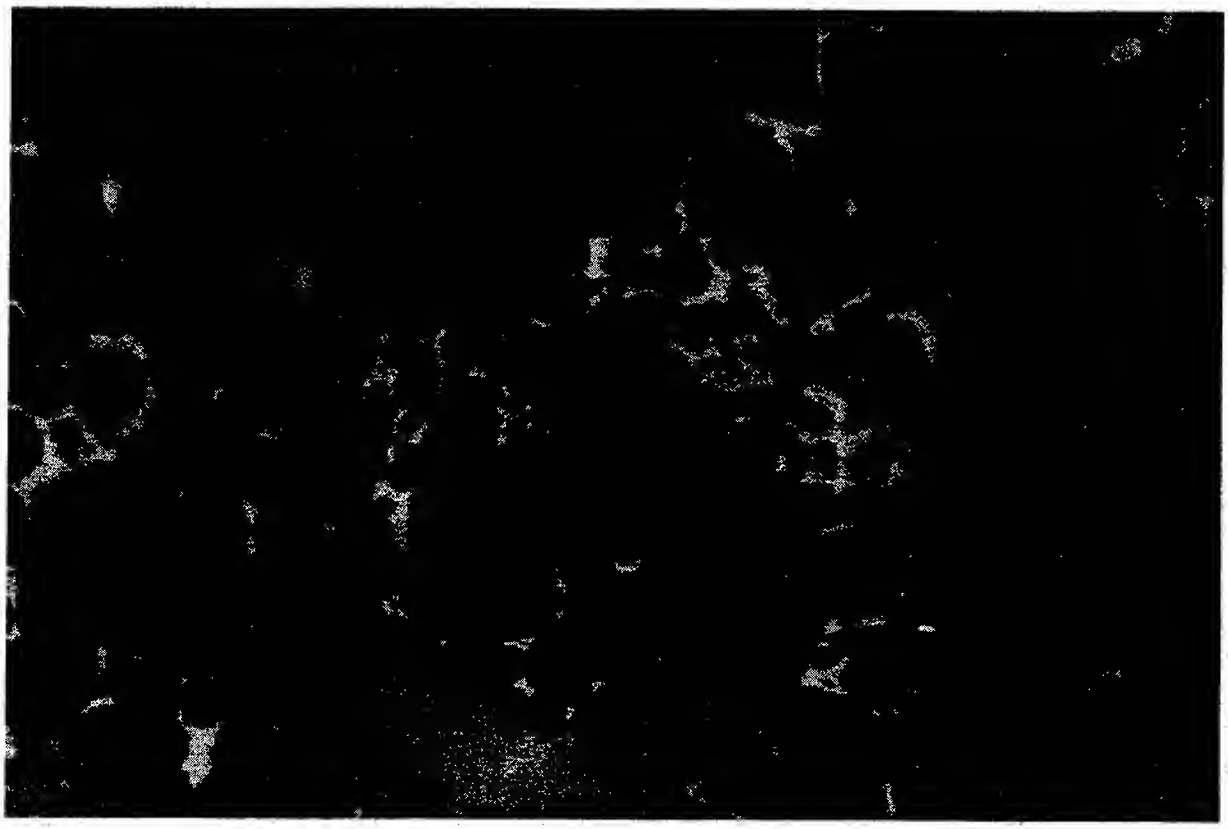


Plate 14 Calcium potassium sulfate (needle shaped). 250 x



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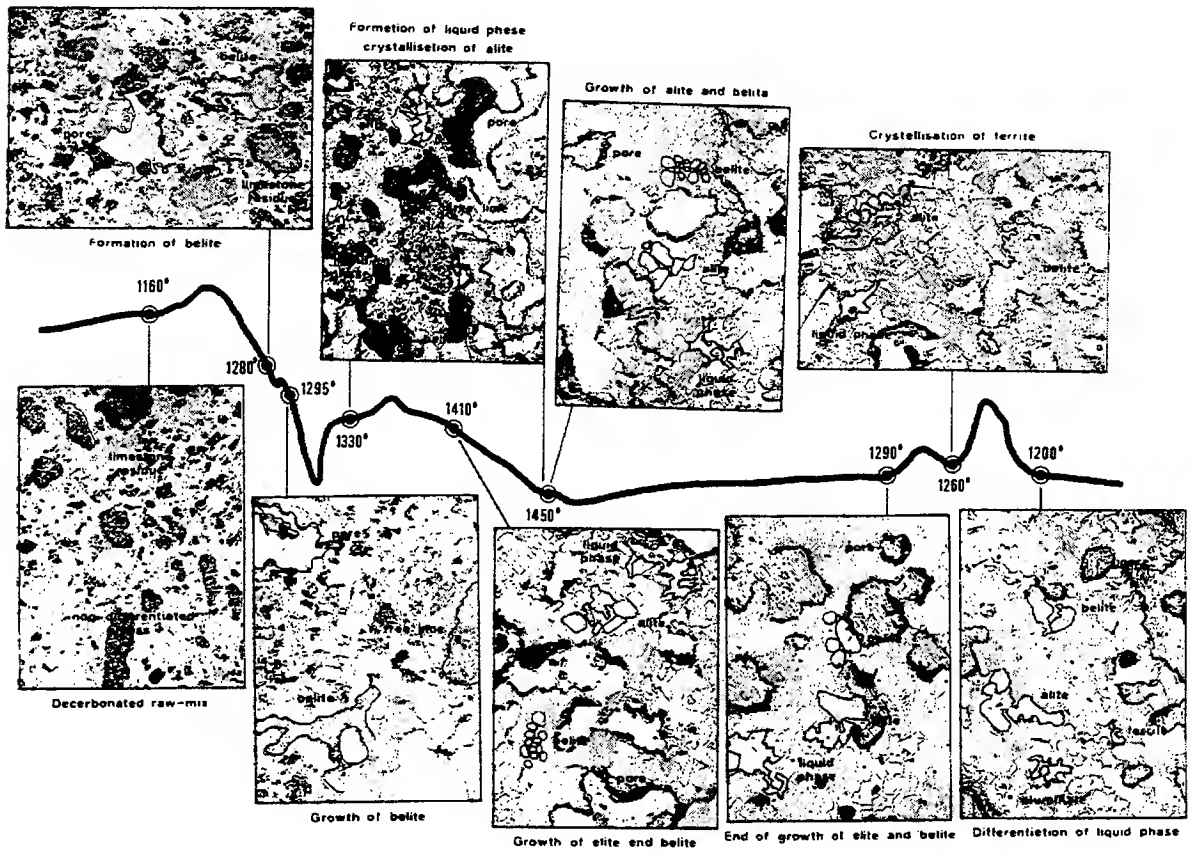
2.2 Formation of Clinker Microstructure During Burning

As clinker formation takes place over a range of temperature, it is not surprising that a gradual change of fabric occurs between the initial raw meal and the finally cooled clinker.

The most common way of ascertaining such changes is to microscopically examine polished sections prepared from raw materials which have been heated to selected temperatures and then immediately quenched, to retain their high-temperature microstructure. Suitable samples may be either obtained via sampling parts located at various positions along a kiln shell, or by heating raw mixes in the laboratory. Now following, the change of microstructure of samples obtained at various temperatures during clinkering in a laboratory DTA investigation are summarized.

As can be deduced from Figure 11 and from other sources (6), the clinker on attaining the various indicated temperatures, shows the following changes in microstructure:

Figure 11 Changes in Microstructure at various temperatures



Upheating cycle:

1160°C

Individual particles of decarbonized limestone (i.e. free lime) and unreacted hematite can be seen against the sub-microscopic, non-differentiated mass. Quartz particles are surrounded by reaction rims, several microns thick. The presence of free lime and belite can be demonstrated by x-ray diffraction. No large pores can be observed.

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1280°C

The reaction rim around quartz particles has now become thicker and developed into rounded belite crystals, large enough to be optically identified. The outline of large decarbonized limestone grains can still be recognized. The presence of large pores is now observed.

1295°C

Quartz grains have mostly disappeared and in their place are now grains of belite. While new belite crystals are formed, the already existing ones at the same time increase in size. Small pockets of liquid phase can already be observed. The structure is now decidedly porous.

1330°C

The presence of liquid phase is very evident at this temperature, as is the presence of optically identifiable alite and free lime crystals.

1410°C

Alite and belite crystals are still growing and showing their typical morphological forms in liquid rich zones. Large pores can still be observed.

1450°C

Alite and belite have further increased in size but have still not reached their final dimensions. Free lime can still be observed in many areas of the clinker in its typical rounded form.

Cooling Cycle:

1290°C

Alite and belite have reached their final size while porosity has generally decreased. The liquid phase is still undifferentiated.

1260°C

Crystallization of the ferrite phase from the melt is detectable (see DTA curve).

1200°C

The clinker has its final microstructure. Alite and belite can now be observed in a matrix consisting of crystallized aluminate and ferrite of sufficient size so as to allow their identification. Alite often appears to have been resorbed (xenomorphic crystals), while on account of slow cooling belite often shows finger-like growths. Small areas of exsolved material can sometimes be seen within aluminate areas, i.e. secondary crystallization of belite has occurred.

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2.3 Influence of Production Conditions on Clinker Microstructure

2.3.1 General

The microstructure of industrially produced clinker is influenced by several process parameters, the most important of which are given in Table 2.

Table 2 Industrial parameters influencing clinker texture

Influence Parameter	Specific factors
Raw mix properties	chemical composition fineness and Mineralogy homogeneity
Fuel type	coal ash
Burning conditions	burning time / temperature kiln atmosphere
Cooling conditions	rate of cooling

2.3.2 Chemical Composition

If burning is properly performed, the quantitative mineralogical composition of a clinker depends only on the chemical characteristics of the raw feed. From theoretical considerations (phase diagrams and Bogue calculation) and microscopical observations, the change of clinker mineralogy with varying clinker compositions can be predicted.

Although it may seem trivial, it is considered useful to give the clinker mineralogy associated with clinker chemical composition (Table 3).

Table 3 Effect of chemical composition on clinker mineralogy

Chemical composition	Clinker mineralogy	
lime saturation too low	alite belite free lime	little abundant traces
lime saturation too high	alite belite free lime	abundant little excessive
alumina ratio high	aluminate > ferrite	
alumina ratio low	ferrite > aluminate	
silica ratio high	little liquid phase	
silica ratio low	liquid phase abundant	
alkalis	alkali - aluminate present	
alkalis together with SO_3	alkali - sulfates present	
more Mg than can be incorporated in the lattice of principal phases	periclase present	

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2.3.3 Fineness and Mineralogy

During the clinkering of a raw mix, a migration and exchange of material takes place. The smaller the grain size, the shorter the transport distance required to achieve complete reaction.

Once a certain grain size is exceeded, the lime and quartz components can no longer react completely within the allowed time. Critical particle size for minerals such as calcite and quartz lie in the range of approx. 50 - 100 µm. The maximum grain size is, therefore, that at which the raw material particles are still just able to react. When this limit is exceeded, 'undigested' relicts remain. If the clusters of free lime and belite are examined, particularly those with sharp boundaries (Plates 15 and 16), it is evident that they owe their formation to the presence of oversize raw material grains. They still show the original appearance of a raw meal grain, the siliceous particles being present as belite clusters and the carbonate particles as free lime clusters as a result of incomplete reaction.

Plate 15 Belite cluster due to large, SiO_2 - rich grain. 220 x

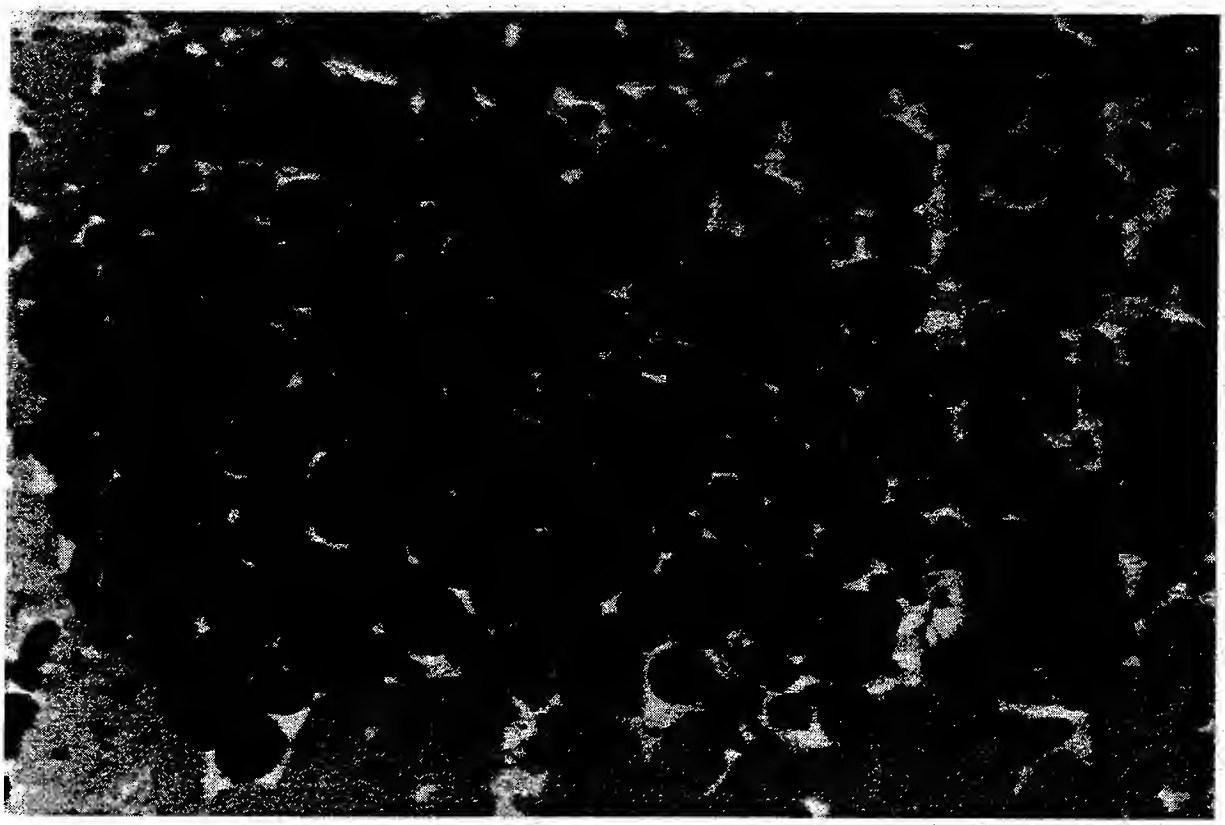

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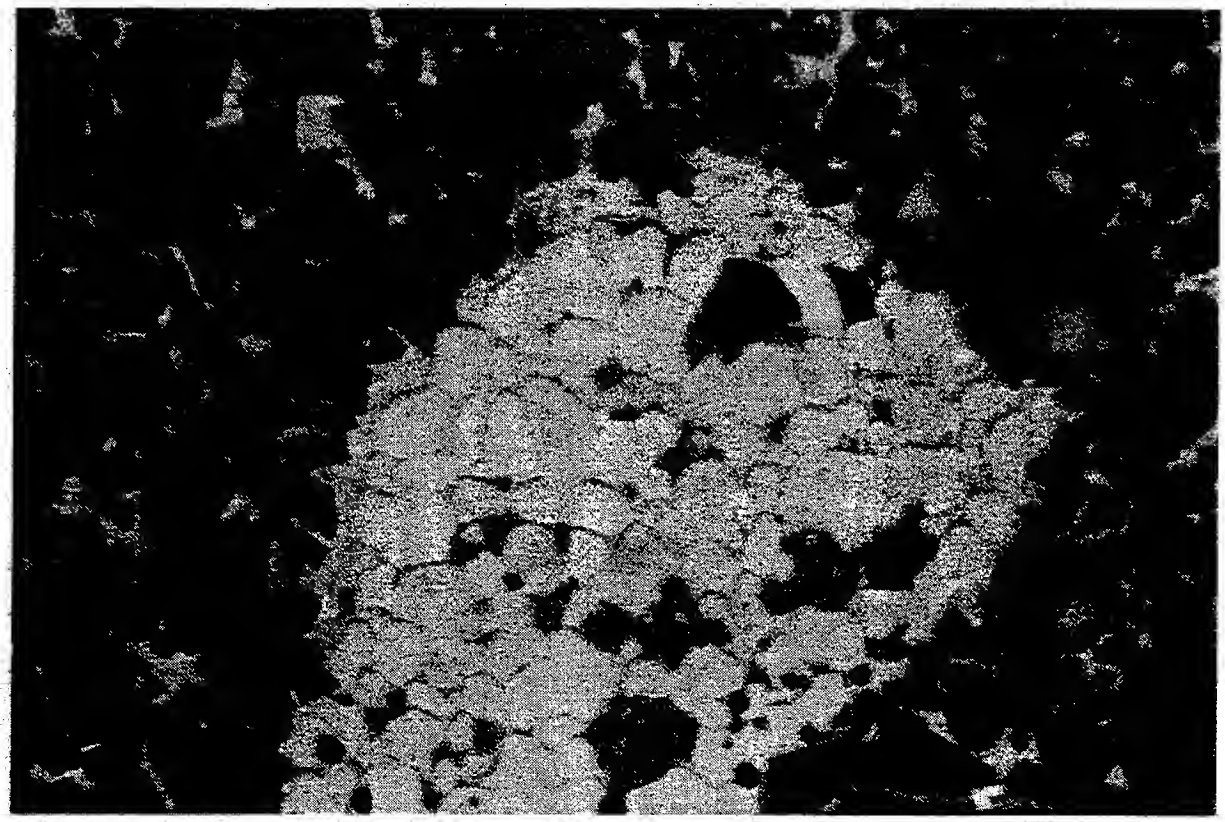
Plate 16 Free lime cluster due to large, lime-rich grain. 64 x



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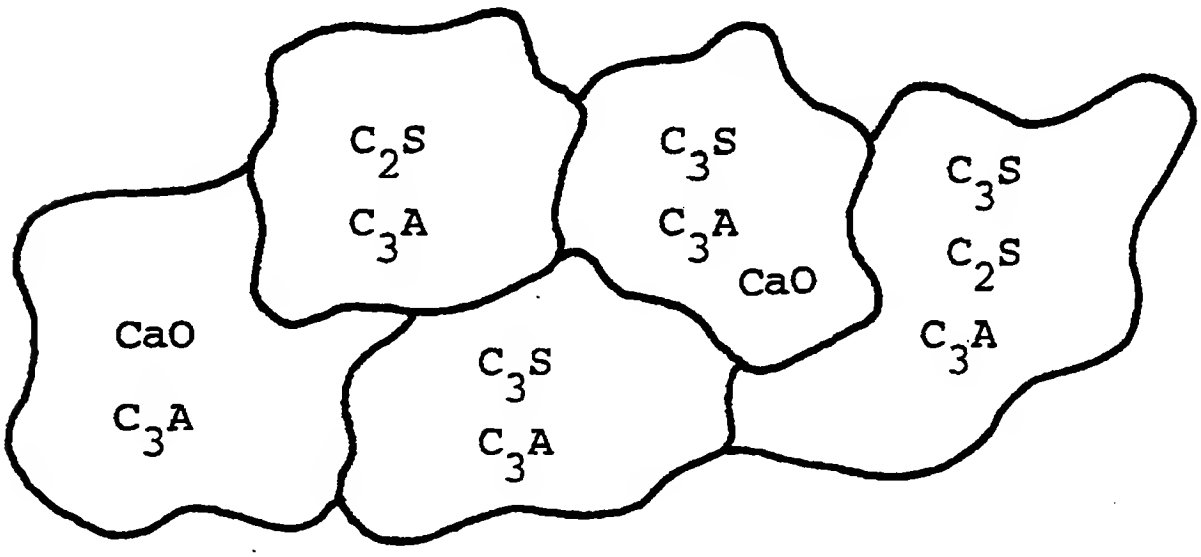
2.3.4 Homogeneity

If the raw mix is insufficiently homogenous, volume units of varying size will exist which do not have the overall bulk chemical composition. It is easy to deduce form the phase diagram for the system CaO - Al_2O_3 - Fe_2O_3 - SiO_2 the phase compositions which can coexist assuming different volumes to have difference chemical composition. In Figure 12 the different phase assemblages in the system CaO - Al_2O_3 - SiO_2 can be seen.

Figure 12 Possible mineral assemblages in the system CaO - Al_2O_3 - SiO_2



With insufficiently homogenous raw feeds we should therefore expect a clinker microstructure of varying mineralogy.

In reality this model exists also in the clinker. Plate 17 shows a clinker with a very inhomogenous distribution of alite, belite and free lime, reflecting directly raw meal inhomogeneity.

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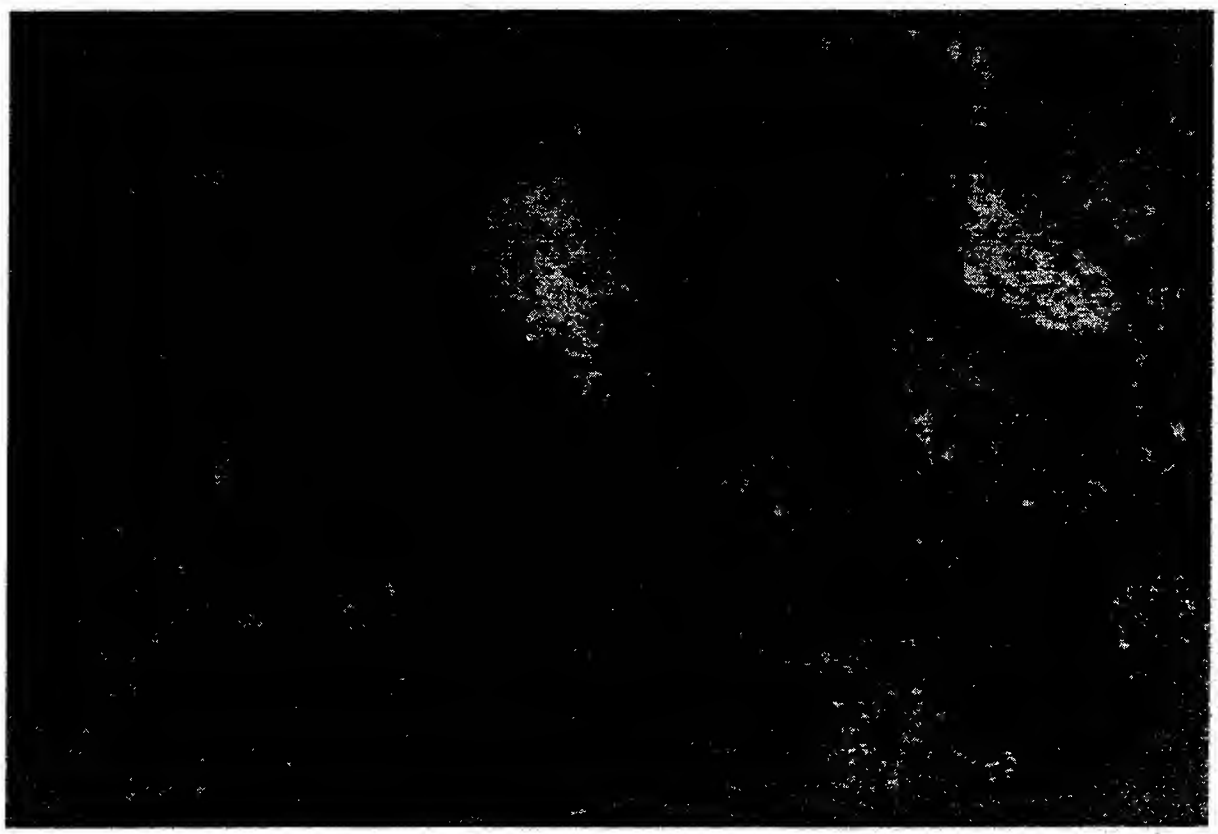
Plate 17 Inhomogeneous distribution of clinker minerals. 64x



In comparison with the inhomogenous microstructure produced from an inhomogenous raw mix, Plate 18 sows the clinker fabric when the raw mix is correctly homogenized.

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Plate 18 Homogenous distribution of minerals. 250 x

In comparison with the effects caused by oversize particles, the concentrations areas due to inhomogeneity contain normally more than one clinker component and usually extend over lager areas. Experience shows, however, that a certain clustering, particularly of belite, is almost always observed in plate clinker. The even distribution (Plate 18) is rather an exception.

2.3.5 Coal Ash

Rotary kilns are in ever increasing numbers again being fired using coal.

Unlike gas and oil, this type of fuel imposes traces of its presence on clinker textures. During the burning of coal, siliceous ash is produced as molten ash drops and these fall as a sediment of the clinker being formed. There is thus a SiO_2 concentration on the surface of the kiln charge. Depending on how far the ash drops are carried into the kiln, they will be more or less assimilated with the clinker granules. Due to the deposition of ash drops rich in SiO_2 on the kiln charge, there is a local lack of CaO. At these places only belite can be formed and thus the deposition leads to typical belite streaks in the clinker as can be seen in Plate 19. Generally, these streaks are found in the peripheral areas of the pieces of clinker.

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Plate 19 Belite layer due to coal ash. 64 x

2.3.6 Burning Time / Temperature

Large-scale industrial process conditions do not allow thermodynamic equilibrium to be reached. If the kiln charge passes through the kiln too quickly (flushing) or the sintering temperature is too low (underburning), the same characteristic fabric results as shown in Plate 20.

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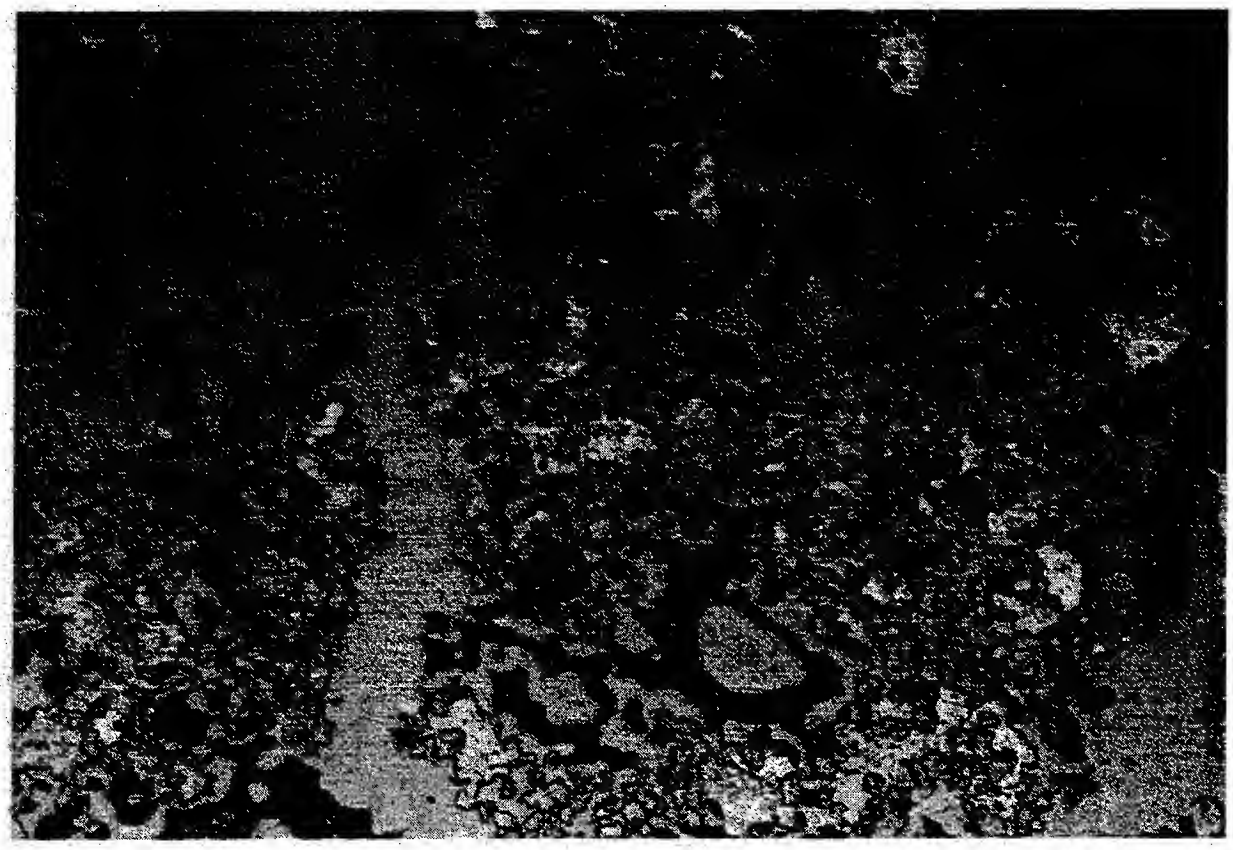
Plate 20 Underburnt clinker. 64 x



Whether the temperature, the heating time of both parameters were too low cannot be distinguished. As can be seen in Plate 20, the alite an belite crystals are unusually small. Alite is surrounded by large amounts of liquid phase, while belite is concentrated in clusters. In the areas where alite and liquid phase coexist, accompanying free lime appears as a result of uncompleted reaction between lime and silica. High porosity with vesicular cavities in places is also typical.

If all these characteristics are considered, it may be concluded that here the formation of the clinker phases stopped at an early stage. The time and / or temperature was not sufficient to form normal clinker from the raw materials.

Another time - temperature anomaly occurs if the clinker remains in the highest temperature range for too long or if the sintering temperature is too high. Coarsening of the alite and belite grains takes place. It can be shown that the crystallization of calcium silicate also continues during cooling as long as the cooling rate is sufficiently slow. Alite and belite can continue to grow under such conditions - provided that the quantity of the reacting material is large enough - until the liquid phase crystallizes. Therefore, with an extreme crystal growth of calcium silicate, it is not clearly distinguishable if the clinker was over burnt or whether the cooling rate was too slow. **In all these events, hard burning results in clinker of reduced hydraulic activity.**

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2.3.7 Cooling Rate

Clinker cooling takes place in two stages, the first cooling stage occurring within the kiln, the second in the clinker cooler.

The rate of cooling within the kiln depends upon the flame length, on its position in the kiln and on the throughput and speed of the kiln charge. The clinker temperature at the kiln outlet is probably between 1350 and 1200 °C. If the flame is long, this part of the cooling process is very slow and alite and belite can grow to an excessive crystal size. In some cases (when the cooler efficiency is low), alite partially decomposes into belite and free lime (see Plate 4).

The texture of the solidified liquid phase is also very dependent on the cooling rate. During slow cooling, the crystals have time to grow. Ferrite and aluminate form a coarsely grained matrix (Plate 9). Alternatively, if the cooling process proceeds quickly, the opposite takes place, i.e. the crystals are finely grained (Plate 11).

Cooling can also proceed so quickly that the crystals can only form in the sub-microscopic range. Distinction between aluminate and ferrite is no longer possible by microscopy but can be effected by x-ray methods.

3. METHODS OF CLINKER INVESTIGATION

3.1 Introduction

Investigations on clinker are usually restricted to determining the following properties:

- identification of the minerals
- estimation of the quantity of each species present
- determination of the composition of individual crystals of any single species and hence the composition range of that species
- determination of the habit and spatial distribution (texture) of the minerals.

Several groups of methods are available for the determination of these properties:

- light microscopy, scanning electron and electron microprobe method
- x-ray diffraction
- thermo-analytical methods
- wet-chemical extraction techniques

Often it is necessary to use the combination of these methods if the maximum amount of information or if the highest degree of accuracy is required. No single technique is capable of determining all the desired properties. In Table 4 the normal range of application of each method is listed.

As can be seen, wet chemical methods are certainly capable of being performed at all plants, whereas only a limited number of plants are equipped to perform x-ray diffraction and reflected light microscopy. These latter methods are, however, certainly available in the central laboratory of a group of plants. Investigations using scanning electron microscopy, and an electron microprobe analyzer can only be undertaken by specialized laboratories possessing such advanced and expensive instruments.

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Table 4 Normal application range of the various methods of clinker investigations

Investigation Method	Clinker properties				Equipment		
	Mineral identification		Mineral composition	Habit / texture	Found in cement plant laboratories	Cost of equipment	Available in TS
	Qualitative	Quantitative					
Microscopy							
Reflected light	+	+	o	+	sometimes	moderate	yes
Scanning electron	o	o	(+)	+	definitely not	very high	no external equipment used
Electron microprobe	+	o	+	o	definitely not	very high	
Diffraction							
X-ray	+	aluminate / ferrite free CaO periclase	ferrite solid solution	o	in a few cases	high*	yes
Wet chemical extraction							
Salicylic acid extraction		alite + belite	o	o	No special equipment required	very low	yes
Glycol extraction		free CaO	o	o		very low	yes
Controlled water extraction		soluble alkali sulfates	o	o		very low	yes

* except when purchased as an addition for an existing x-ray fluorescence spectrometer

3.2 Microscopy and Microprobe Analysis

Cement clinker is, almost without exception, investigated by reflected light microscopy on etched, polished sections. The use of transmitted light microscopy either on thin sections or on powders is rarely practiced.

Great care must be exercised when selecting representative samples, the quantity required depending on the topic under investigation. When information on minerals content, porosity, etc. , is required, the chosen material should be sampled from not only one but from several clinker lumps.

The selected sample is first embedded into a suitable resin (e.g. epoxy resin) to facilitate handling and then treated as follows:

- ground to produce a flat, matt surface (e.g. using carborundum powder)
- polished to obtain a flat, shining surface (e.g. using 0.3 µm diamond paste)
- etched to differentiate between the individual clinker minerals (e.g. using hydrofluoric or acetic acid vapors, or using nitric acid solution).

By using non aqueous grinding and polishing media, the loss of the water soluble alkali sulfates and free lime during sample preparation can be prevented.

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3.2.1.1 Qualitative mineral analysis

Using the thus prepared polished section it is possible by using magnifications of up to 500 x to determine qualitatively in clinker:

- the minerals present and their habit
- their polymorphic modifications (sometimes)
- clinker fabric (porosity, homogeneity, etc.)

and from this information deduce the influence of such process conditions as:

- fineness of raw mix grinding
- homogeneity of raw feed
- time / temperature of burning
- rate of cooling
- type of fuel used

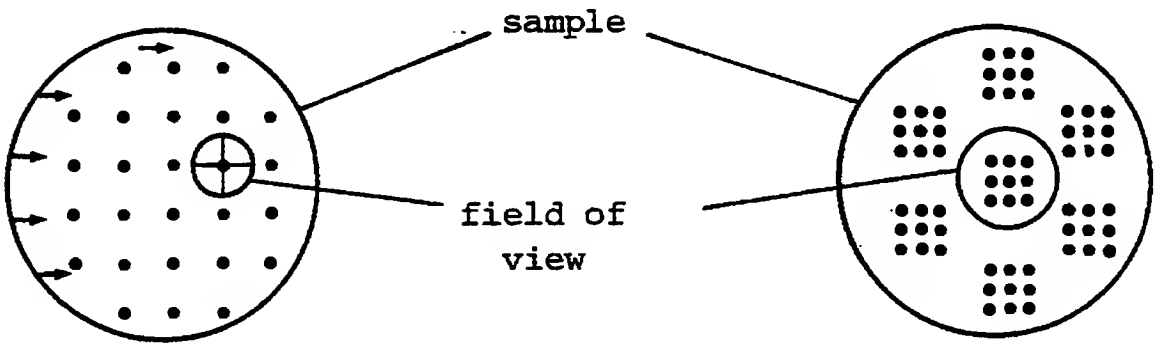
The use of this method has already been demonstrated in Chapters 2.1, 2.2, 2.3 (see also (4), (5).

3.2.1.2 Quantitative mineral analysis

Clinker properties are **quantitatively** determined by the use of one or two methods, namely:

- **Linear point counting** i.e. the sample is moved in regular steps and the mineral under the cross-wire is identified and counted.
- **Area point counting** i.e. a point grind containing 100 points, placed in the microscope ocular, lies apparently over the clinker, and the mineral at each point is counted without moving the clinker. Only after all the points have been counted is the clinker moved to another field of view.

Linear point counting **Area point counting**



Both methods determine the area percent of the minerals in the polished section, and to convert the results into weight percent, the following mineral densities (g/cm^3) must be taken into account:

alite	3.20	belite	3.28	aluminate	3.04
ferrite	3.77	periclase	3.58	free lime	3.30

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To obtain a high accuracy, the TS/MA counts 4000 points (N), giving an absolute statistical counting error (δ) at the 95 % confidence level of

$$\delta = 1.96\sqrt{\frac{P(100-P)}{N}}$$

P = % points for the counted minerals

The most usual features to be quantitatively determined include:

- clinker mineralogy
- size distribution of minerals (is related to hydration reactivity)
- homogeneity
- porosity

Whereas the Bogue formula calculates the **potential** quantities of minerals present, microscopical analysis determines the quantities **actually present**. The difference between both methods has already been demonstrated in Chapter 1.4.2 for alite and belite.

3.2.2 Scanning Electron Microscope Analysis

By using a fine electron beam in place of light in such a way that the beam scans the sample surface in a raster pattern, it is possible to obtain an optical image of the surface on a TV screen. This image can have a magnification of up to 10,000 x, combined with a **good depth of focus**. By further increasing the magnification (up to 100,000 x) the quality of the image is reduced.

Using the **energy disperse x-ray spectrometry** technique (EDAX), it is now also possible to determine semi-quantitatively the chemical composition of the observed minerals.

Examples of clinker investigated by scanning electron microscopy are displayed in Plates 21 - 24.

Plate 21 Idiomorphic alite (A) embedded in sulfate-rich matrix (S). 2000 x

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Plate 22 Alite (A) and belite (B) in sulfate-rich matrix (S). 2000 x

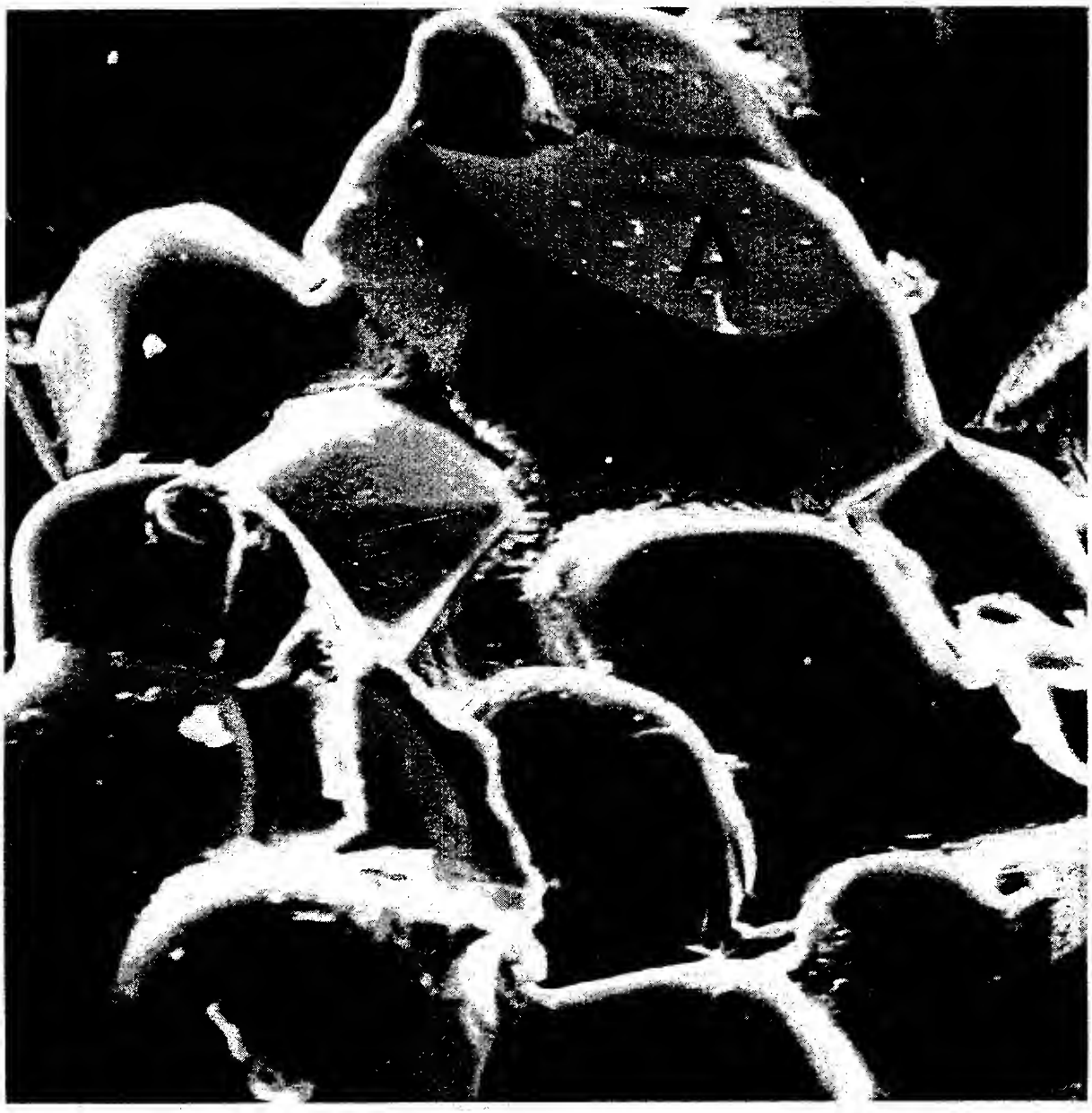

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Plate 23 **Alite (A) in aluminate / ferrite matrix (M). 2000 x**

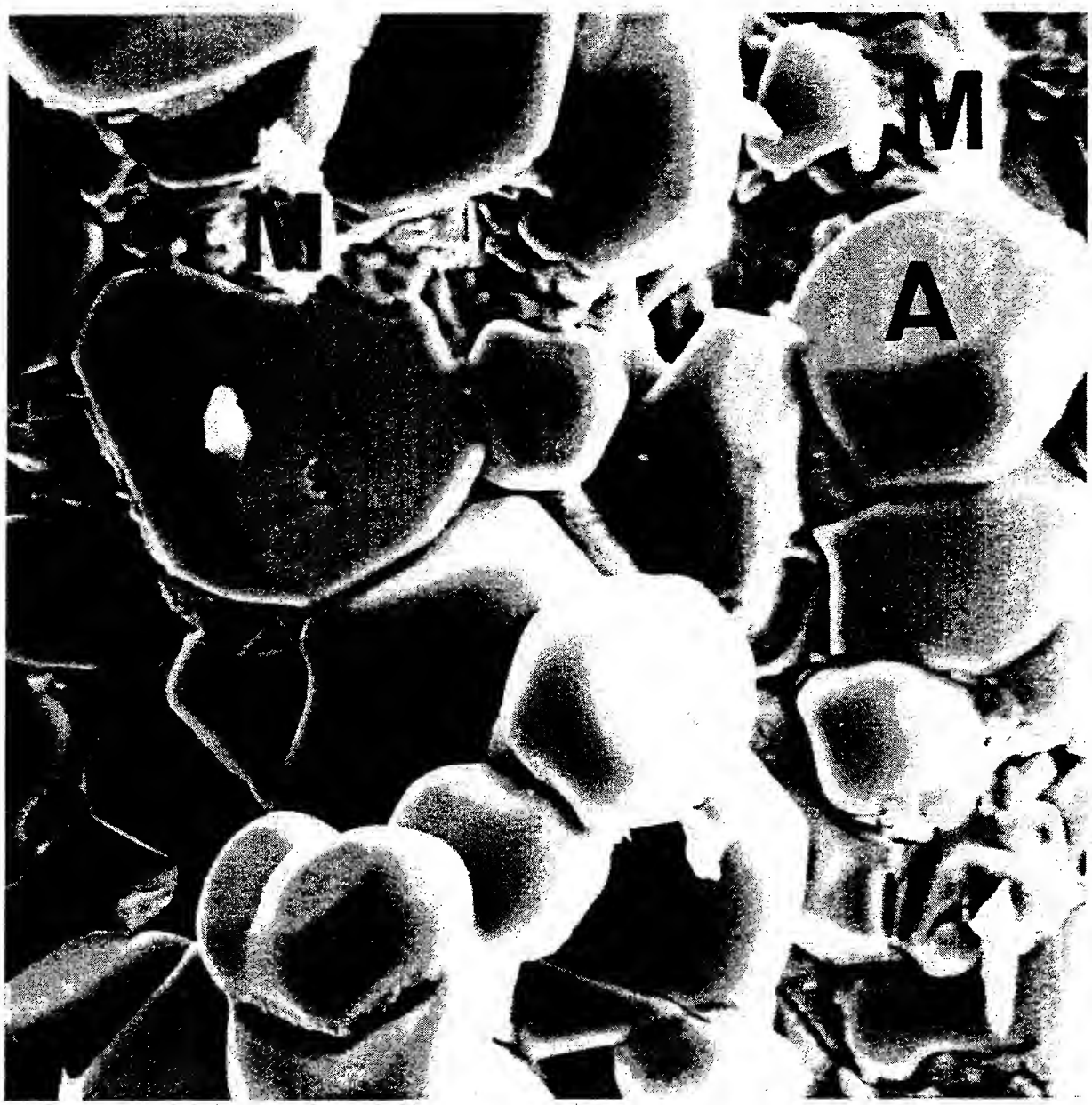

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Plate 24 **Free lime (L) in contact with alite (A). 9500 x**

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3.3 X-ray Powder Diffraction

3.3.1 Methodical

The examination of clinker by x-ray powder diffraction can with much experience be used to determine:

- the minerals present - quantitative and qualitative
- their polymorphic modification (s)
- the approximate composition of certain minerals

The full potential use of the method as a basic routine technique, however, is made difficult by the following:

- variations in the chemical composition of almost all minerals
- more than one modification of the same mineral almost always present
- the x-ray intensities of the various modifications are not equal
- overlapping of various reflections and small concentrations require pre-preparation of the sample by wet chemical extraction
- the relatively high cost of the equipment. This can be somewhat reduced when x-ray diffraction facilities are combined with existing x-ray fluorescence equipment.

3.3.2 Experimental Technique

Samples to be investigated must be finer than 20 μm when determinations are desired. The prepared sample is then irradiated in a focused x-ray beam (e.g. Cu or Cr radiation) and the characteristic peaks of each mineral recorded.

For **qualitative** analysis the 2 θ values of the peaks are compared with the characteristic peaks of the known clinker minerals and in this way the following can be determined:

- the presence and polymorphic modifications of alite, belite
- the presence and solid solutions of aluminate and ferrite
- the presence of MgO, CaO_{free}

For **quantitative analysis,** suitable strong reflections of the required minerals, free of interference from other mineral peaks, are required. As measurement criterion either the effective **peak height** or **integrated peak area** may be applied, this being obtained by subtracting the background radiation level form the **overall** peak height or peak area.

Peak area method

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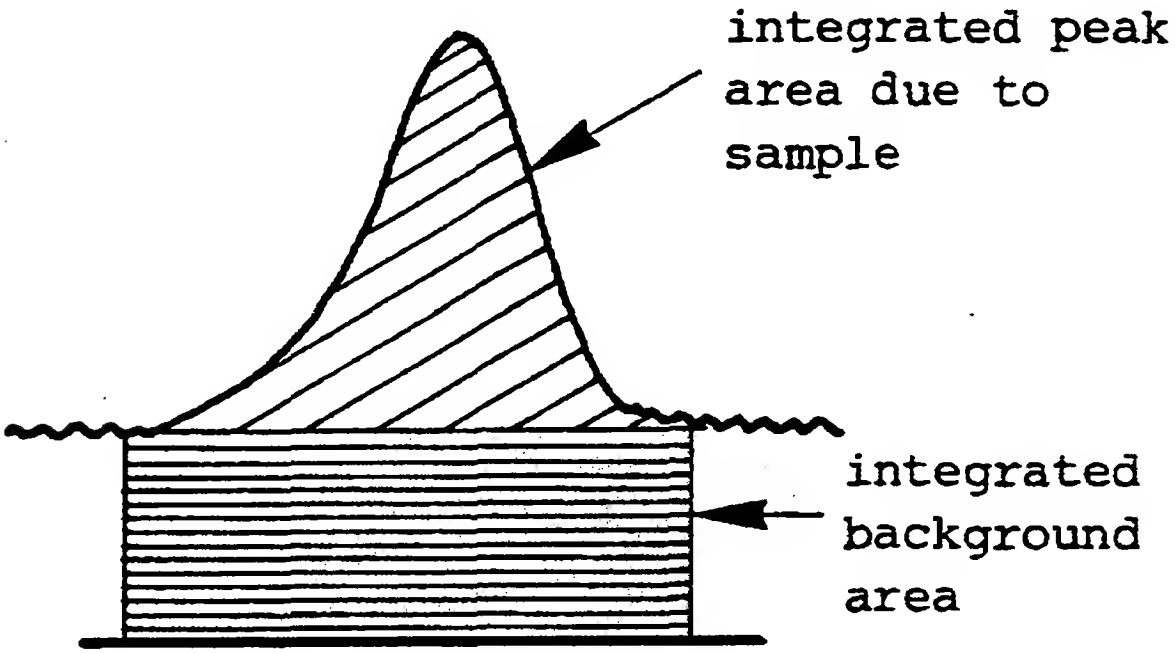
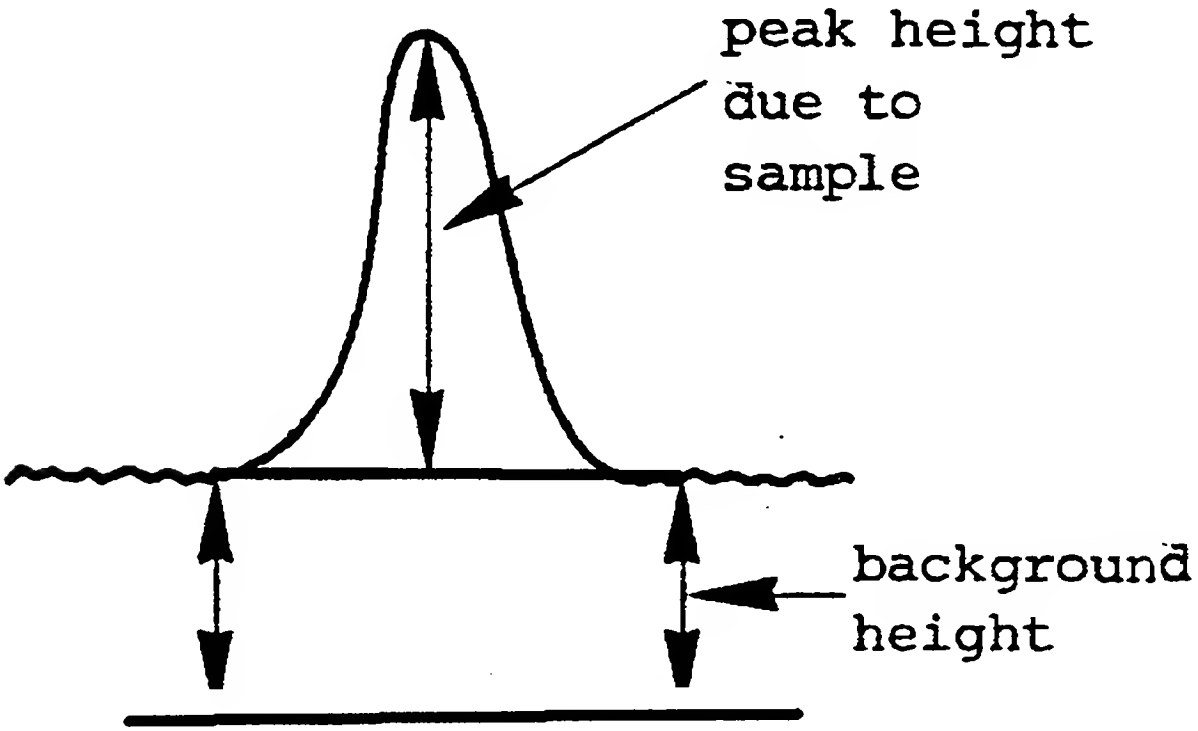


Peak height method



Two types of calibration can be applied in which the measured effective peak area of height is compared to that of either an **internal** or **external** standard (e.g. SiO_2, CaF_2, TiO_2).

The **external standard** method, i.e. standard and sample not mixed together, eliminates errors due to fluctuations in the equipment itself. (e.g. instability of x-ray generator, x-ray counter, etc.), but makes no allowance for changes in material parameters (e.g. absorption by matrix). This latter effect can be compensated for by employing an **internal standard**, i.e. by mixing the standard powder **into** the powder being investigated.

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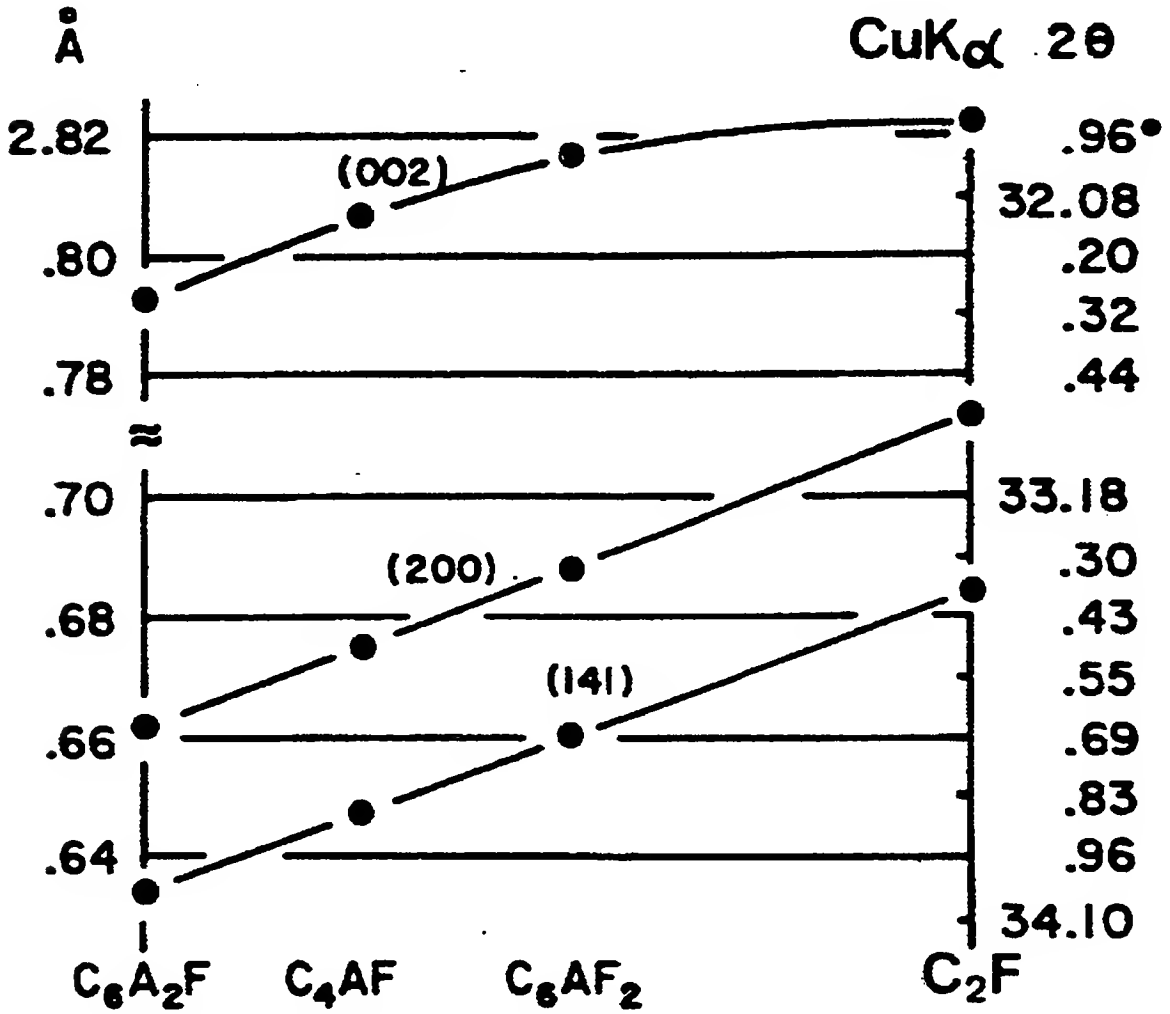
3.3.3 Applications of x-ray diffraction

Determination of ferrite solid solution

As mentioned previously, the ferrite composition is only approximately C_4AF. The true solid solution present in clinker can, however, be determined using x-ray diffraction, employing the change in the 2 θ value of characteristic peaks as evaluation criterion. The latter are shown in Figure 13.

Figure 13 Change of peak reflections (002, 200, 141) for ferrites in the C_2F, C_6A_2F series.



A useful application of knowing the **actual** ferrite composition is, that it can be used to obtain a better Bogue formula estimation of aluminate and ferrite than the one obtained by assuming the composition to be C_4AF.

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Determination of free lime

By extending x-ray fluorescence facilities to include x-ray diffraction (at a fraction of the cost for x-ray diffraction alone), it is possible to quickly and accurately measure free lime as a process control method, avoiding the use of glycol extraction. The main advantage of the x-ray method is the reduced time required (10 minutes) using the same sample prepared for x-ray fluorescence. Free lime values in the range 0.5 - 5 % can be determined at an accuracy of ± 0.3 % CaO_{free} at the 95 % confidence level.

For the determination of other, more complicated clinker analyses (e.g. quantitative determination of the clinker minerals), it is highly recommended that contact be first established with the TS/MA or other suitable qualified laboratories.

3.4 Clinker Analysis by Wet Chemical Extraction Method

When neither microscope nor x-ray facilities are available, certain minerals or groups of minerals can still be quantitatively determined by selective extraction using wet chemical methods.

Minerals which can be quantitatively determined to an acceptable degree of accuracy by such inexpensive but rather limited techniques include:

Alite + belite + free CaO: using ethanolic salicylic acid

Free lime and $Ca(OH)_2$: using ethylene glycol extraction. The proportion of free lime alone can be ascertained by subtracting the $Ca(OH)_2$ value determined via its H_2O loss on thermal decomposition

Alkali and alkali/calcium sulfates: by controlled extraction with H_2O and determination of Na_2O, K_2O and SO_3 levels in the extracted solution

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Chapter 3

Refractory Materials and Coating Formation

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Refractory Materials and Coating Formation

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1. REFRACTORY MATERIALS AND THEIR PROPERTIES

1.1 General Introduction

The refractory lining of a modern, high performance Portland cement kiln is a complex system performing different specific functions at various locations. These include:

- protection of the kiln shell and tyres from reaching excessive temperatures
- protection of the kiln shell against chemical attack by aggressive gases and liquids
- protection of the kiln shell from mechanical abrasion by kiln charge
- reduction of heat losses to acceptably low values
- increase of heat exchange between gases and kiln charge
- control of material flow

Whereas in the rotary kilns in service until the late 1930's these duties were performed by fireclay products of approximately 40% Al_2O_3 content, the high performance kilns of today necessitate the use of a wide spectrum of specially developed high MgO, Al_2O_3 and SiO_2 refractory types. Within each of these main categories, each manufacturer has designed his product differently to incorporate the required degree of resistance to the combined chemical, thermal and mechanical attack that will be encountered in service.

From the viewpoint of clinker production, the financial outlay of relining a kiln (including removal and installation of the worn and new refractories) represents approximately 2-4 % of total cement manufacturing costs. The effective loss of cement sales resulting from reduced kiln availability constitutes, however, a much higher loss should the kiln be shut down for more than 10-15 days.

Within the plants of "Holderbank" Group, an estimated 30'000 tons of refractories are consumed each year which, when the following sales costs are assumed

magnesite :	1'400 DM/ton
dolomite :	1'100 DM/ton
others :	1'050 DM/ton

results in an annual expenditure for refractory materials alone of $20'000'000. To this already substantial sum must further be added the man-hours associated with relining, and lost cement sales resulting from 44'000 hr. of kiln down time.

It is therefore obvious that the life expectancy of the kiln refractory lining is a topic of basic financial importance. Depending on kiln zone, refractory types and production conditions, service lives in the critical sintering and transition zones may range from only several weeks to well in excess of 1 year.

The principal factors determining a linings service life are schematically displayed in Fig. 1, indicating those topics which lie within the responsibility of the refractory supplier and those over which the cement producer has most influence.

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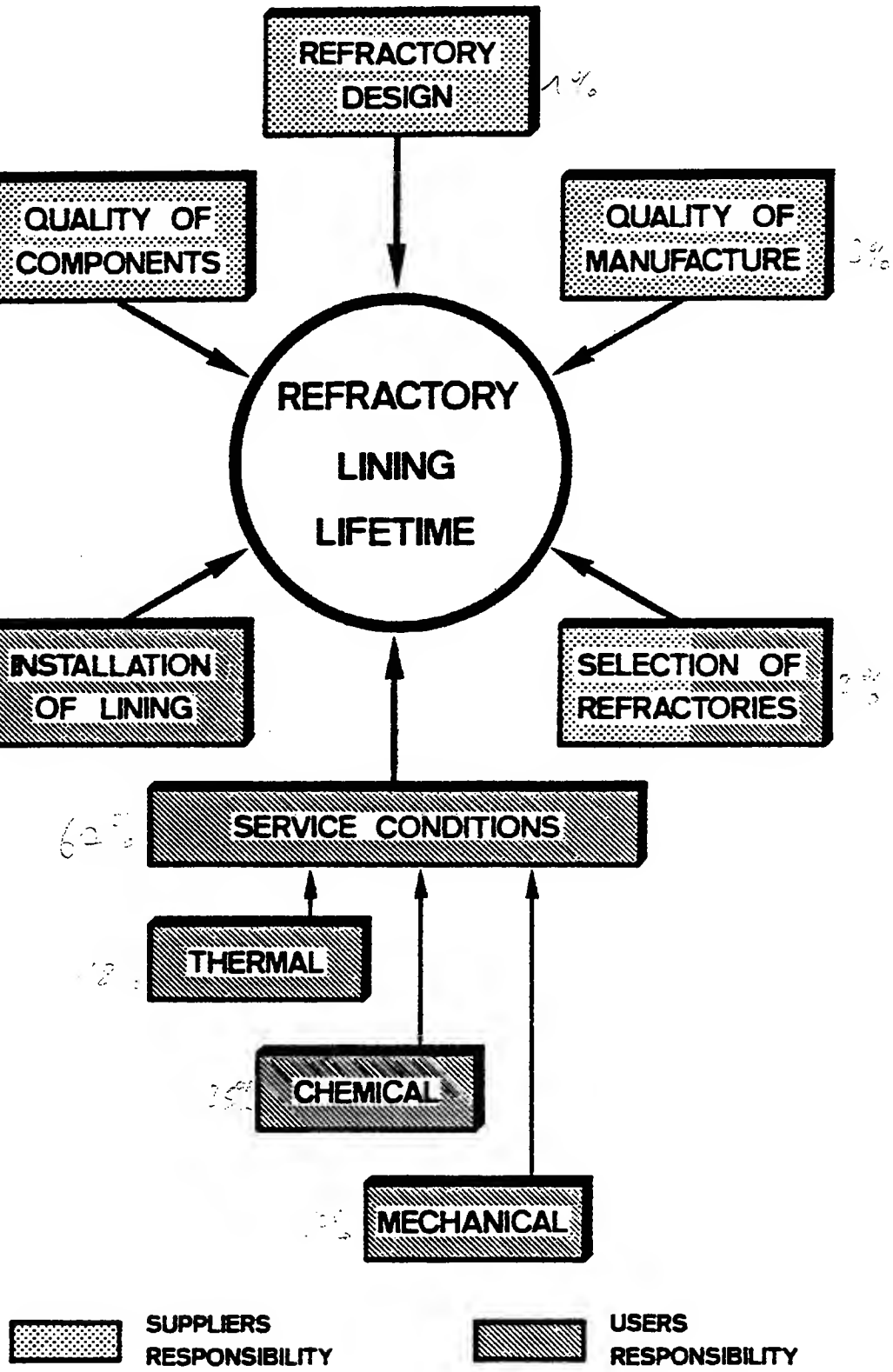
Fig.1 Factors Influencing Refractory Lining Lifetime



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The recent reintroduction of coal-firing, together with the use of dry process kilns has resulted in refractories being subjected to appreciably increased attack by fused coal ash, by local reducing atmospheres and by alkali sulfates. To combat these trends, refractory manufacturers have recently increased their range of cement kiln refractories to include iron and chrome free periclase-spinel and magnesium silicate (forsterite) types.

The technological complexity of the kiln linings has been developed to such a stage and is of such financial importance that each plant should have a refractories specialist possessing a good basic technical/theoretical knowledge and having additionally sound practical experience. To help achieve this aim, the "Holderbank" Refractories Handbook for the Cement Industry is intended as a compact source of cement industry specific information and is intended for both newcomer and refractories specialist alike.

1.2 Classification of Refractory Products

1.2.1 General

Refractory materials are mainly composed of Al_2O_3, SiO_2, MgO, CaO and Cr_2O_3 existing as the free oxide or combined together. They are defined according to ISO Standard R 836 as "non metallic materials or products having a minimum pyrometric cone equivalent of 1500°C. Internationally they are classified, according to PRE recommendations, as compromising of the following technological categories:

- dense, shaped refractory products .
- shaped insulating refractory products .
- prepared unshaped dense and insulating refractory materials

1.2.2 Chemical Classifications

The different groups of refractory materials, based on their chemical composition, is listed in Table 1.

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Table 1 **Chemical classification of dense shaped refractory products according to ISO 1109 1975 (E)**

	Designation	Quality Limits
Acid Group	High-alumina products Group 1	$Al_2O_3 \geq 56$ %
	High-alumina products Group 2	45 % $\leq Al_2O_3 <$ 56 %
	Fire-clay products	30 % $\leq Al_2O_3 <$ 45 %
	Low-alumina fire-clay products	10 % $\leq Al_2O_3 <$ 30 % $SiO_2 <$ 85 %
	Siliceous products	85 % $\leq SiO_2 <$ 93 %
	Silica products	$SiO_2 \geq$ 93 %
Basic Group	Basic products	
	- Magnesite	MgO $\geq$ 80 %
	- Magnesite-Chrome	55 % $\leq$ MgO $<$ 80 %
	- Chrome-Magnesite	25 % $\leq$ MgO $<$ 55 %
	- Chromite	$Cr_2O_3 \geq$ 25 % MgO $\leq$ 25 %
	- Forsterite	MgO, SiO_2
	- Dolomite	CaO, MgO
Special Products	Special products on the basis of: - Carbon - Graphite - Zirconium silicate - Zirconium dioxide - Silicon carbide - Carbides (except silicon carbide) - Nitrides - Borides - Spinels (except chromite) Products based on more than one oxide (except basic prod.) Products based on pure oxides, e.g. alumina, silica, magnesia, zirconium dioxide High purity products	

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As can be observed according to ISO Recommendations, there are

- acidic refractory products
- basic refractory products
- special products

each with a defined percentage of the component oxides in the finished products. In a further subdivision of the fireclay products (i.e. 30% Al_2O_3 < 45 %), the following categories are recognized according to PRE/R 29:

Table 2 Groups of Fireclay Products

Group	Al_2O_3 content (%)
30	30-33
33	33-37
37	37-40
40	40-42
42	42-45

The properties of several ranges of refractories are provided in Table 3.

Table 3 Composition and Properties of Refractory materials

Refractory Materials	Alumina-rich refractories			Magnesite MgO ⩾ 80%			Magnesite Chrome 55⩽MgO⩽80	Chrome-magnesite 25⩽MgO⩽55	Forsterite	Dolomite	Special basic products
				low in Fe	Fe-rich	Bricks					
Al_2O_3	45 - 50	60 - 72	80	<1	<2	⩽8	4 - 11	<22	1 - 10	0.8 - 1.1	3.2 - 4.3
SiO_2	45			0.8 - 3.5	⩽2.5	⩽2.5	1 - 3.5	<6	35 - 40	0.7 - 1	7.1 - 9
Fe_2O_3	1 - 2	1 - 2		0.6 - 2.5	3.5 - 10	⩽12	5 - 12	7 - 15	5 - 12	1	6.5 - 8
MgO	0.2	0.2		92 - 96	>80	77 - 92	55 - 80	25 - 55	45 - 60	37	53 - 63
CaO	0.2	0.2		0.8 - 3.0	<4	<3	1 - 3	<2.5	0.2 - 1.5	60	13.8-18.3
Cr_2O_3	-			<1	<1	⩽0.5	6 - 20	15 - 35	1.5 - 10		6.4 - 8.2
$Na_2O + K_2O$		0.5 - 1.5									
Porosity %	19 - 22		20 - 25	16 - 22	18 - 24	16 - 25	16 - 23	17 - 24	16 - 23	16 - 22	20 - 24
Refractoriness under load t_a °C (DFB)	1480 - 1550	1500 - 1650		1550 - >1700			1550 - 1650		1600 - 1650	1550 - 1760	
Thermal conductivity at 1000°C Kcal/m °C h	1.2 - 1.5	1.5 - 2.2	1.5 - 2.2	2.5 - 3.5	2.5 - 3.5	1.8 - 2.6	2 - 3	1.5 - 2.5	1.6 - 2	2.3	2.9
Application in Transition zone	X	X	X		X		X	X		X	X
Application in Sinter zone			X	X	X	X	X	X	X	X	X
Predominant mineral phases	Corrundum Mullite			Periclase Spinel Monticellite Mg-Ferrite Merwinite			Periclase Monticellite Forsterite Magnesioferrite Spinel Chromite Merwinite		Forsterite Periclase Magnesio-ferrite Spinel	CaO Periclase	

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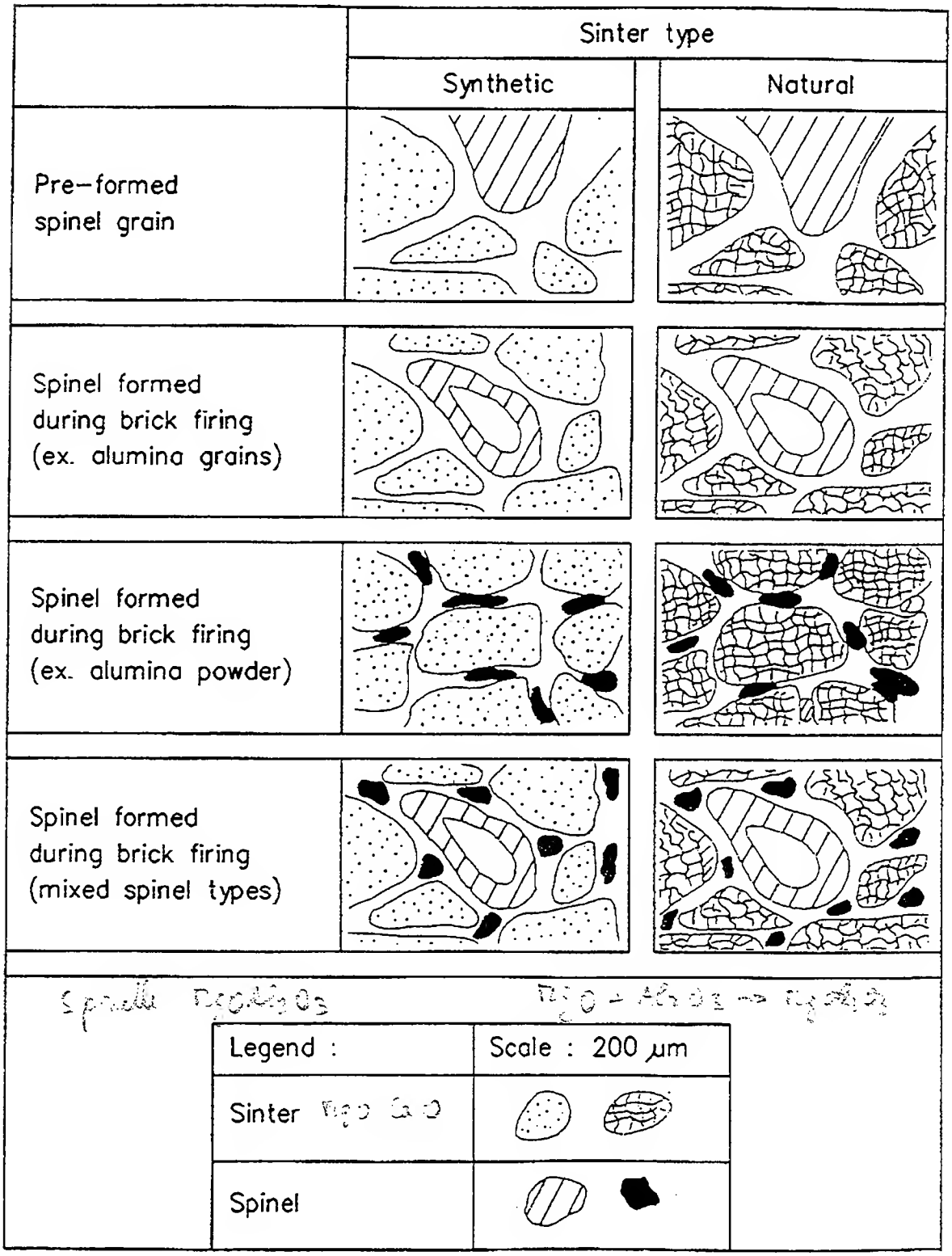
Appendix 1 Structures of "Spinel" Bricks



1.2.3 Nomenclature of Manufacturing Process

To provide further information on the various chemically based classes of shaped refractories, the manufacturer may supply additional characterizing information such as:

- process of shaping
- method of bonding the material

With regard to the shaping process, various methods are employed by the refractory industry in the manufacture of bricks.

Besides the casting of melts of specific compositions (fusion cast products), the methods of fine ceramics, such as slip casting, continuous extrusion or pressing of finely crushed materials, have been used increasingly during recent years.

However, the so-called coarse ceramic method is preferred for the manufacture of refractory bricks. Production by this technique includes the following steps:

- crushing
- classification (grading)
- mixing
- shaping (pressing)
- drying
- firing

1.2.4 Types of Bonding

From the viewpoint of bonding, various methods are recognized which can be classified according to the temperature at which strength properties are developed, e.g.

- unfired products (<150°C)
- fired at moderate temperatures (150-800°C)
- fired at high temperatures (>800°C)

and also according to the material nature of the bond, e.g.

- fireclay bond
- inorganic bond
- organic bond
- hydraulic bond
- ceramic bond

Since the bonding in many refractory materials tends towards ceramic bonding at operating temperatures, it is necessary to state that the previous classification of refractory material bonds is based on products coming from the supplier ready to be put into the kiln.

Examples of the bonding types encountered in refractories commonly supplied to the cement industry are given in Table 4.

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Table 4 Examples of the Various Material Bonds

Nature of bond	Refractory Type	Characteristics
CERAMIC (Eutectic)	Fireclay bricks High-alumina bricks Silicate bonded magnesite bricks	The brick is first heated up above the eutectic temperature of the naturally present bonding oxides, and the structure formed by the liquid on cooling bonds the principal components together
CERAMIC (High fired)	High fired mangesia spinel bricks	The principal components are "dry" bonded by the use of very high temperatures (1700 - 1850°C),which causes the principal solid components to react directly with one another
INORGANIC	Phosphate-bonded alumina bricks	The bricks are bonded by chemical reaction between the added bonding agents(e.g. phosphates, sulfates, chlorides, chromates) and the bricks components at temperatures up to 150°C
HYDRAULIC	Refractory castables Gunning mixes	The refractory materials are bonded by a hydraulic calcium aluminate cement, just like normal concrete

1.2.5 Shaped Insulating Refractory Products

These are shaped products having the same chemical classification as dense shaped refractories (see Table 1) but having a total porosity of at least 45 %. They are further characterized (ISO 22451972 E) by the temperature at which they do not display a permanent linear change of above 2 %, and by their bulk density. Such products are also known as lightweight insulating refractories and are mainly used as insulating backing in the heat exchangers of dry process kilns.

1.2.6 Prepared Unshaped Dense and Insulating Refractories

These are prepared mixtures which contain an aggregate and a binder ready for use either

- as supplied e.g. ramming mixes
- after addition of suitable liquids
  e.g. refractory castables

These may be either dense or insulating, the latter requiring a porosity of ≥ 45 %.

Such materials be used for the following purposes, with further subdivisions being recognized within each group.

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Table 5: Categories of Unshaped Refractories

MONOLITHIC CONSTRUCTION/ REPAIRS	Ramming mixes delivered in a pliable non-coherent state and first harden upon heating Moldable mixes delivered in a plastic form and harden upon heating Refractory castables delivered dry Setting and hardening occurs without heating after mixing H2O or other liquids Gunning mixes delivered in a non-coherent state, suitable for placing by pneumatic projection
MORTARS/ JOINTING MATERIALS	Refractory cements these are placed wet, and harden at high temperature by ceramic bonding Refractory cements (hydraulic binder) as above but harden without heating Refractory cements (chemical binder) harden by chemical reaction at lower temperatures than required for cement bonding

1.3 Manufacture of Refractory Products

1.3.1 Manufacturing Process

A typical flowsheet for the production of shaped and unshaped magnesite based refractories is shown in Fig. 2. As can be seen, the process can be split into four principal stages including

- preparation of sintered magnesia and dried chromite in defined particle size ranges
- preparation of mix having specific proportions of the various magnesite and chromite fractions
- pressing of required format (shaped refractories)
- bonding process (direct, eutectic, chemical)

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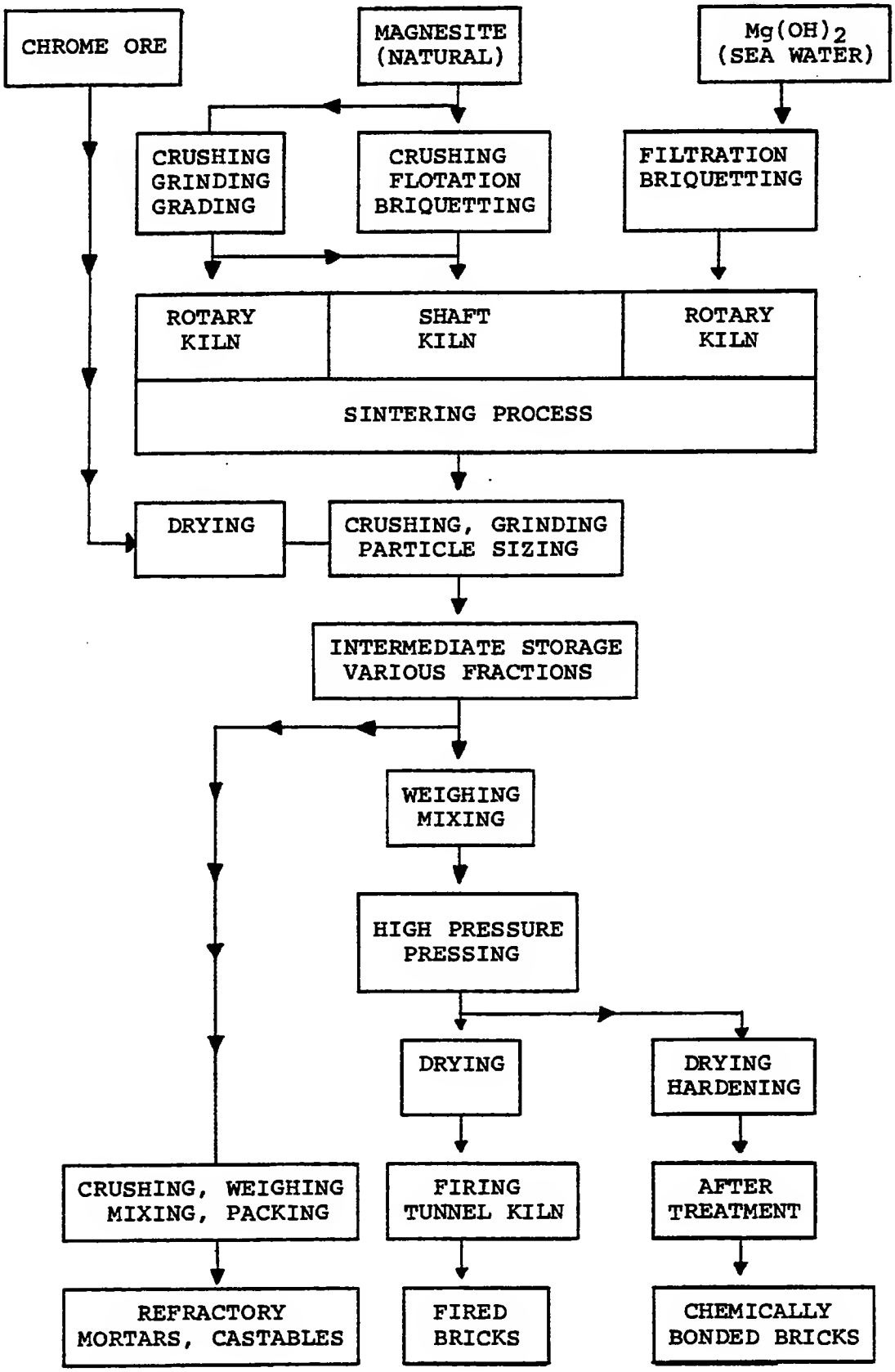
Fig. 2 **Flow Diagram for the Production of Shaped and Unshaped Magnesite Based Refractory Products**



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As can be seen, the magnesite sinter may originate from either natural magnesite or from $Mg(OH)_2$ extracted from sea water. To produce the latter, sea water containing about 200 g MgO/100 liters is reacted with slaked lime slurry to precipitate $Mg(OH)_2$. The resulting slurry is filtered to produce a paste which is calcined at 900-950°C to form "caustic calcined magnesia". This is then fed through briquetting rollers to press dry, dense tablets, which are then dead burnt at 1800°C in rotary kilns. During the burning process, the undesired B_2O_3 content of the MgO is appreciably reduced. In the production from natural magnesite, impurities such as dolomite are removed by floatation after crushing. From the dead burnt sinter the coarser fractions are then prepared by crushing and sieving, with the finest being prepared by subsequent milling.

By using different sizes of sinter and chromite in various proportions, and controlling the total porosity and pore size distribution, a variation of refractory properties may be achieved.

1.3.2 Source of Materials

The production of refractory materials was originally located in areas near to iron works where clay deposits were available with a sufficiently high Al_2O_3 content to allow the production of fireclay bricks. With the development of refractories to include high alumina, magnesite, magnesite chrome, and most recently periclase spinel types, the raw materials are not available in the traditional production areas and have to be purchased on the international market. The refractory manufacturer therefore cannot always produce his product with materials from the same source.

Of the various materials of a grade suitable for refractory production, the most well known sources are indicated in Table 6.

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TABLE 6 Sources of Raw Materials for Refractory Production

Raw Material	Source	
For Basic Refractory Products	Country	Location
Magnesite		
• coarse crystalline(natural)	Austria Canada	Radenthein Veitsch Radium Hot Spring
• crypto crystalline(natural)	Greece Turkey Yugoslavia India Australia South Africa	Euboea
• sea water (synthetic)	Italy England Ireland Japan USA	Sardinia Hartlepool Drogheda
chrome ore (natural)	Philippines Turkey South Africa Greece Iran	Transvaal
magnesia spinel(synthetic)	Japan England Germany	

The key factor deciding whether a raw material can be used for refractory production is the content of the minor oxides, i.e. CaO, SiO_2, Fe_2O_3 and Al_2O_3 which, when they are present in excessive quantities, lead to a reduced refractoriness. Synthetic raw materials (e.g. sea water magnesite) are often too pure and have to have their composition modified by the use of additives to control their CaO/SiO_2 ratio and hence improve their sintering behavior.

Although the critical composition limits at which a manufacturer accepts or rejects a specific raw material are not made available and remain a trade secret, typical compositions of acceptable materials are as indicated in Table 7.

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Table 7 Typical Compositions of Raw Materials

Material	Chemical Composition (wt %)						
	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	Cr_2O_3	TiO_2
BAUXITE (sintered)	6.2	86.9	1.8	0.04	0.08	--	2.7
CHROME ORE	5.4	20.4	18.4	0.6	18.1	37.0	--
DOLOMITE (sintered)	2.3	1.0	0.8	58.8	36.2	--	0.2
MAGNESITE (sea water-sintered)	0.4	0.2	0.6	0.9	98.1	--	--
MAGNESITE (raw)	2.1	0.6	1.5	2.1	93.6	--	--

1.3.3 Production Details

Production details for the manufacture of refractories are seldomly made available. The following points, however, give an indication of typical layout and performance of a refractories plant, which receives already sintered raw materials (see Table 8).

Table 8 Production Details

No of silos per component:	up to 4 for various size fractions
Weighing and mixing process:	usually automatically controlled by computer. Proportions change with brick types and material source
Brick presses:	1500 T total pressure, fully automatic operation. Often up to 5 presses/plant
Tunnel oven (for firing):	capacity 30'000 t/y
Firing data:	time up to 3 days temperature 1550 - 1850°C heat consumption 1500-2000 kcal/kg

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1.4 Testing of Unused Refractories

1.4.1 General

Refractory products are routinely tested at several stages during their manufacture and in some industries (e.g. steel production) are even tested by the user prior to delivery before actually being put into service. Testing may be considered as falling into the following categories and fulfilling the indicated functions.

Table 9 Testing of Unused Refractories

Type of testing	Purpose
Manufacturers Process and Quality Control	Check on Production Process/Equipment/Operations
Standard Refractory Tests (e.g. DIN, ASTM, etc.)	To ensure that product meets manufacturers description
Non-Standard Tests (e.g. at HMC)	To assess whether product is suitable for operations in the specific user industry
Users Acceptance Quality Control	Check whether delivered refractories meet manufacturers specifications or are suitable for users specific requirements

Whereas in the cement industry the used standard ASTM, DIN, BS tests are considered as giving a not unreliable indication of the performance of the mortar or concrete in service, the prescribed standard tests for refractories are not specific for clinker production conditions. It has even been suggested that it would be more useful to test refractories that had already been in service, and not new unused refractories, and so obtain a practice related indication of the suitability of refractories under clinker production conditions.

1.4.2 Manufacturers Process Control

Upon delivery, the refractory manufacturer checks the chemical composition of the supplied chrome ore, sintered magnesite, dolomite, bauxite etc. to ensure that the levels of the minor oxides, e.g. CaO, SiO_2, Fe_2O_3, Al_2O_3, do not exceed the requirements for the refractory type concerned (e.g. direct or indirect bonded basic bricks). In the case of sea water magnesite the boron content is checked, as is the loss on ignition in the case of sintered dolomite, which must be used within approximately 48 hours of delivery.

During the mixing process, the granulometry of the various fractions of each component (e.g. chrome ore, magnesite) is routinely checked as is the mixing ratio of the raw components. These checks are especially necessary when more than one type of for example magnesite-chrome brick are being simultaneously produced. In larger production plants, quality control samples are taken on a statistical basis and are sent to the quality control laboratory. In several companies, all the quality control and production data are stored in a computer databank, thus allowing the production history of any specific delivery to be easily recalled.

In the most modern production lines, the whole process is fully automated from component mixing to packing. After pressing, each unfired brick is automatically checked for its dimensional accuracy, whilst the weights are checked only on chosen samples. After firing, the bricks are all individually tested automatically for dimensions, and by a controller who checks for various subjective factors. The following table lists typical factors evaluated in the subjective tests:

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Table 10 Criteria for "Optical" inspection of Refractory Bricks

OPTICAL INSPECTION OF FIRED REFRACTORY BRICKS

Date:9.3.95		Manufacturer	:Mino Yogyo
Brick Type: CF-83S		Format	ISO 320
No. of Bricks Tested:1			

CLANG TEST

Clang Response	tense	normal	dull
(using small hammer)		1	

APPEARENCE

Aspect	subjective impression		
	none	insignificant	significant
broken corners	1		
broken edges	1		
popouts/holes	1		
melting	1		
irregular surface	1		
cracking	1		
peeling	1		

TEXTURE

Texture	normal/homogeneous	cracking	inhomogeneous
broken section			
cut section	1		

numbers refer to individual bricks

ATTACHMENTS

Aspect	comment
steel plate	
correct face	yes
accurately placed	yes
excessive glue	no
cardboard	
correct face	yes
accurately placed	yes
excessive glue	no

MARKINGS

stamped/ colouring	yes
brick type	yes CF-83S
format	yes ISO 320
production data	yes 61

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1.4.3 Standard Refractory Tests

The product description of refractories is defined in terms of various standard test methods (ASTM, BS, DIN, ISO) which measure their chemical composition and physical properties. Testing is performed both at room and elevated temperatures and for shaped refractories includes the following:

Table 11 Standard Tests for Refractory Materials

Temperature	Property	Standard		
		ASTM	BS	DIN
Ambient	Dimensions	C-134	1902	
	Density	C-134	1902	51'065
	Porosity	C-20	1902	51 '056 51'058
	Cold crushing strength	C-113	1902	51'048 51'067
	Chemical composition	C-572, 573,574	1902	51'070
Elevated	Pyrometric cone equivalent	C- 24	1902	51'063
	Hot rupture modulus	C-583	1902	51'048
	Refractoriness under load	C-16	1902	51'053
	Creep in compression		1902	51'053
	Thermal expansion	C-832	1902	51'045
	Permanent linear change	C-113	1902	51'066
	Shock resistance	C38, 107,122	1902	51'068
	Thermal conductivity	C-201, 202	1902	51'046

Testing procedures have also been established by ASTM for unshaped refractories and include the following in addition to those applicable for shaped products:

- workability index (C-181)
- dry firing shrinkage (C-179)
- cold bond strength

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It should be appreciated, however, that various test methods supply data which cannot be numerically come pared with that obtained by another standard test method, e.g. thermal conductivity. This must be kept in mind when selecting refractories from international suppliers, who often define the quality of their product according to their national standards. A higher catalogue value does therefore not always necessarily imply a higher quality product, and makes a rational selection even more difficult. To avoid such difficulties, international standard specifications are in the process of being established, but progress in this direction has been slow. So far only test procedures for Pyrometric Cone Equivalent (ISO R 528) and refractoriness under load (ISO R 1893) are internationally defined. Test procedures for high temperature modulus of rupture, bulk and true density, and permanent linear change have reached a "draft international standard" status. For the majority most important properties no international standard yet exists.

The previously listed standard tests, although they do to a moderate extent indicate the general performance of a refractory, are not wholly specific for conditions representative of those encountered in cement rotary kilns. It is therefore not surprising that in the cement industry no detailed generally known and accepted guidelines have been established for the exact properties of the refractory types required in the various kiln zones.

Although qualitative rather than quantitative, a hypothetical relationship can be proposed (Table 12), indicating which physical and chemical properties exert an influence on the rate of wear by the common mechanical, thermal and chemical wear factors. From this it can therefore be deduced that the standard tests are more suited for determining the resistance to mechanical attack, less for thermal attack, and almost no indication against resistance to attack by chemical influences.

Table 12 Significance of Characteristic Refractory Properties for In-service Life Expectancy

SIGNIFICANCE / CHARACTERISTIC	WEAR FACTORS								
	MECHANICAL			THERMAL			CHEMICAL		
	TH. STRESS	STRESS	ABRASION	OVERHEATING	SHOCK	FATIGUE	INFILTRATION	ALKALI CONDENSATION	$pO_2.SO_3.CO_2$
DIMENSIONS / ACCURACY	●								
DENSITY			○						
POROSITY			○		○		○	○	
COLD CRUSHING STRENGTH	◐	◐	◐						
CHEMICAL COMPOSITION				○			○		
REFRACTORINESS				○					
HOT RUPTURE MODULUS	◐	◐							
THERMAL EXPANSION	●				○				
PERMANENT LINEAR CHANGE	●								
SHOCK RESISTANCE					◐				
THERMAL CONDUCTIVITY									

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1.4.4 Non-Standard Tests

As a means of assessing the performance of refractories under more practice related conditions, almost all manufacturers have developed their own non-standard test procedures. These include methods for the determination of

- coating adhesion thermal
- shock resistance
- resistance to alkali attack

all of which can be tested by not too complicated test procedures.

1.4.5 Sinterzone Coating Adhesion

The tendency towards coating adhesion can be determined at HMC/MA by means of its own recently developed Fig. 3, and fully described in various publications. In developing this method, all the previously developed procedures were evaluated, their deficiencies eliminated and their advantages combined in a single large scale laboratory method.

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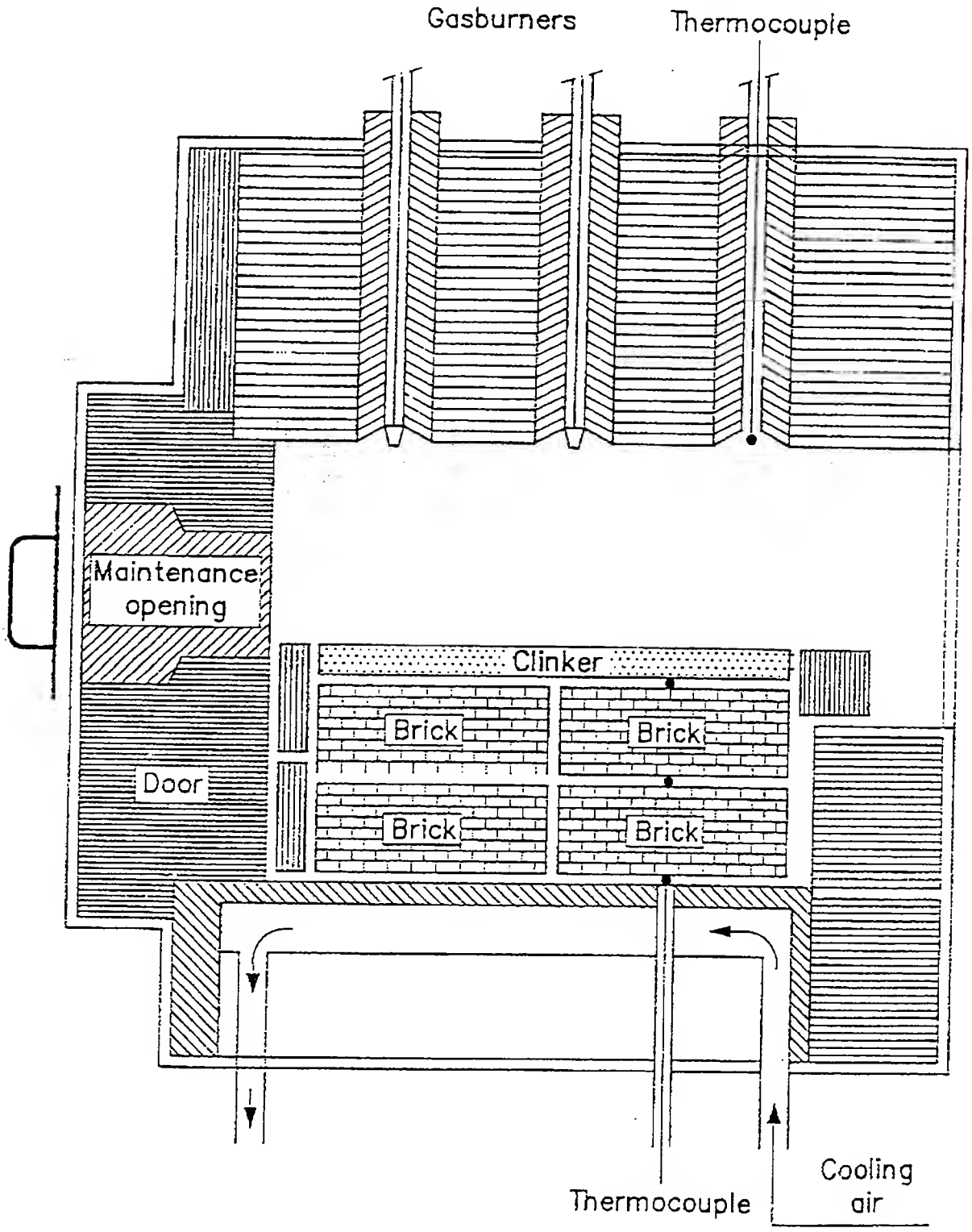
Fig. 3: Coating Test Furnace - Longitudinal section



Examples representative of the type and degree of various observed contact surfaces obtained by this method are given in Fig. 4.

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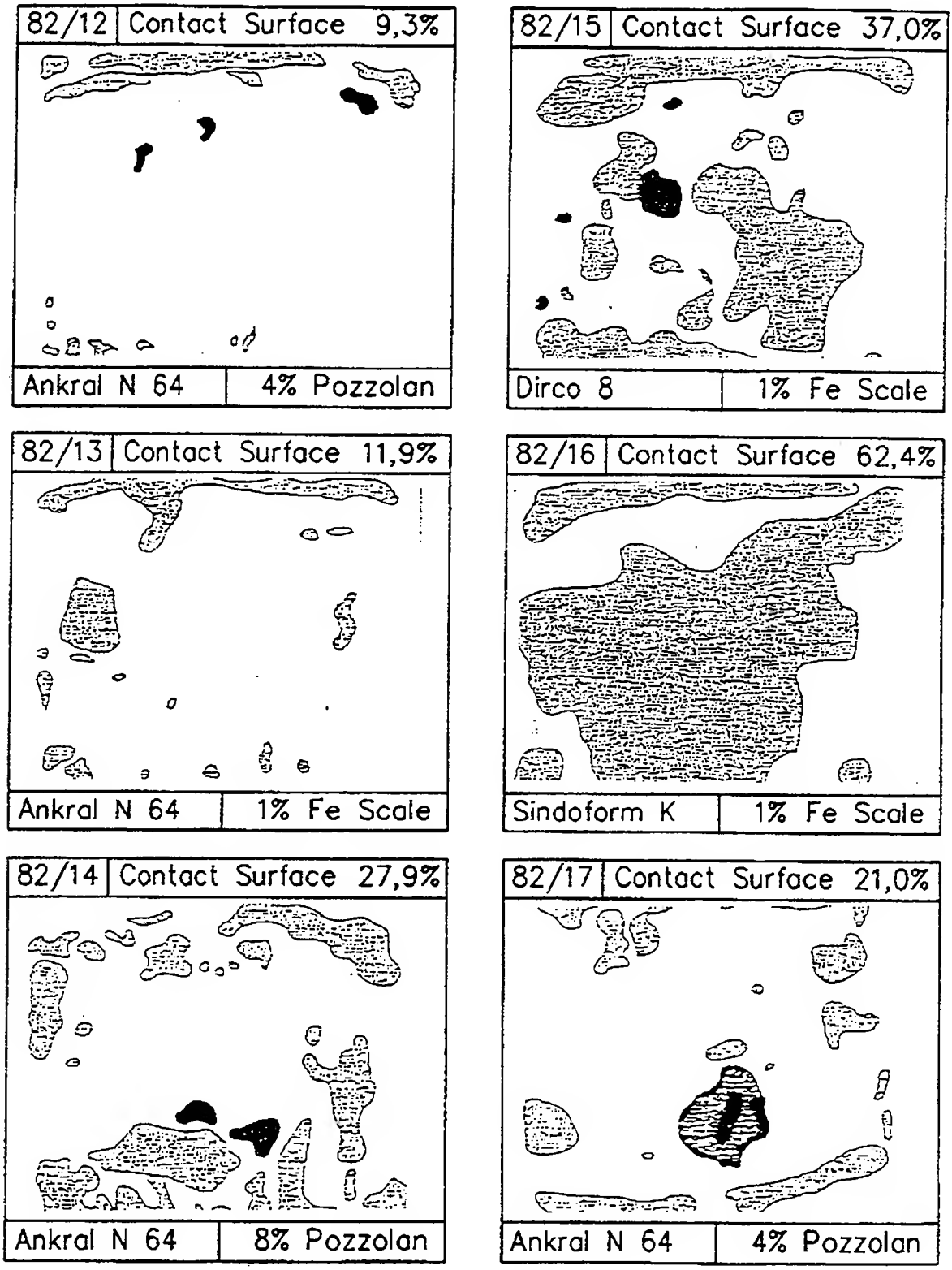
Fig. 4: Typical Contact Zones in Coating Zones



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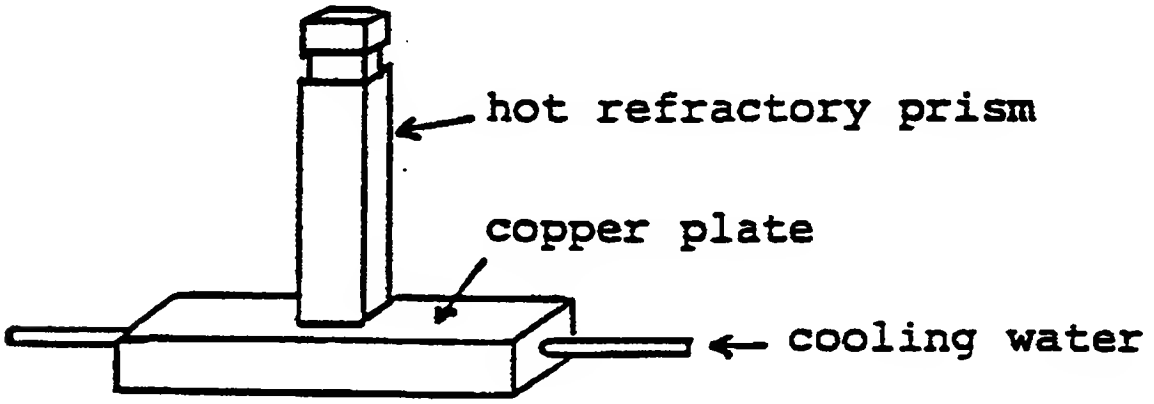
1.4.6 Thermal Shock Resistance

A simple, but extremely aggressive thermal shock resistance test, employed internally in the German steel industry, requires a heated prism of the refractory material under test to be placed on its end on top of a water cooled copper plate (see Fig. 5).

Fig. 5 Arrangement of VDEhL Thermal Shock Test



This test is considered to be more practice relevant than the temperature shock caused by spraying the complete refractory test piece with water or air, conditions never encountered in practice. The test, because only one end of the refractory material is cooled, is said to be much more demanding than either of the two DIN methods and at the same time more practice-like.

1.4.7 Alkali Attack

To determine the resistance of high alumina refractories to alkali attack, many manufacturers use a modified form of the DIN 51'069 test procedure originally developed for testing the resistance of refractories to blast furnace slags. In this method 50 mm diameter x 35 mm deep holes in the test pieces are filled with K_2CO_3 and the material heated for 5 hr. at 1100°C. After cooling, the test samples are cut with a diamond saw and attack judged according to depth of penetration and to whether the bricks show cracking or not. Examples showing a weak and a strong resistance to attack by this simple test method are shown in Figs. 6a and 6b.

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Fig 6a Strong Alkali Attack

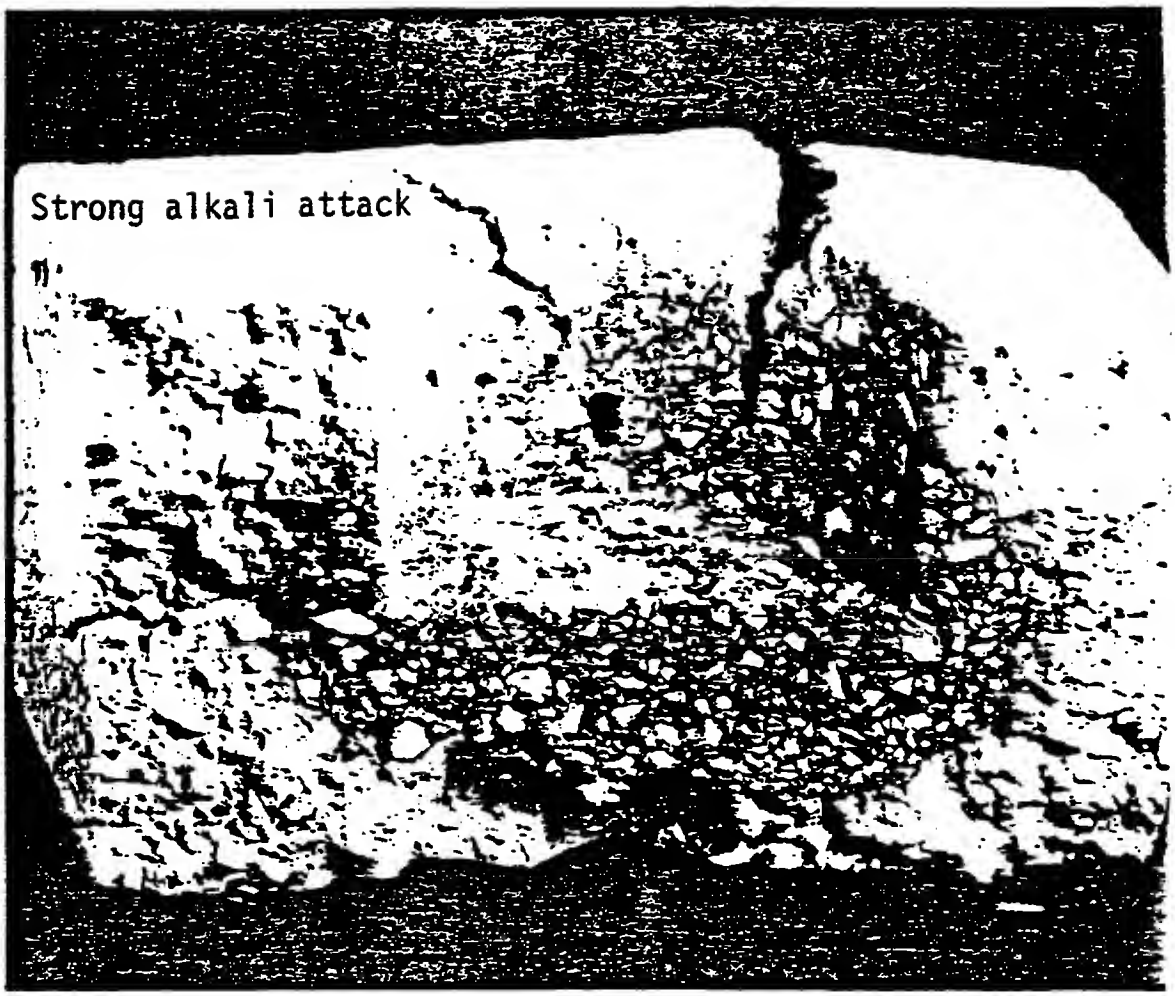

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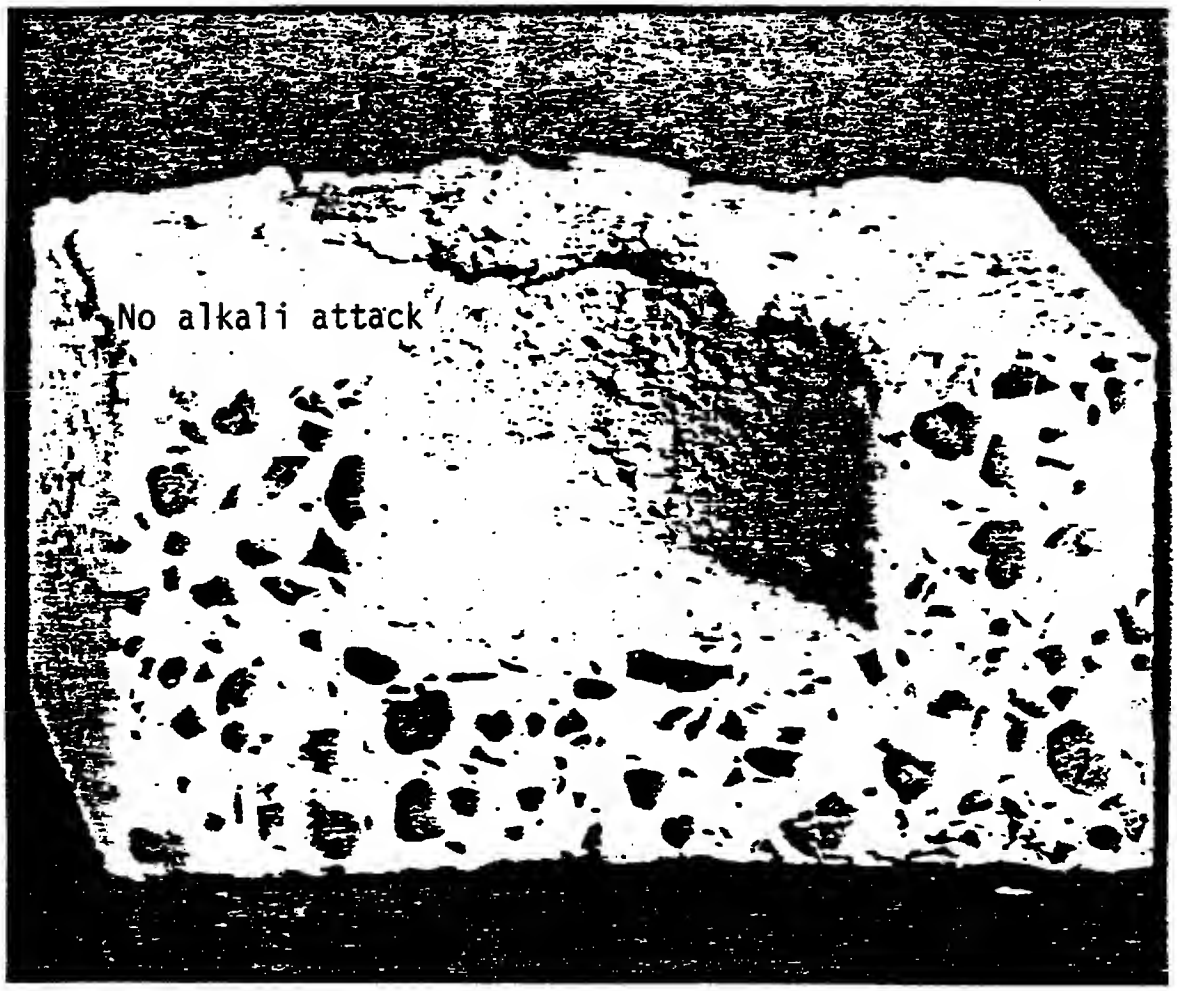
Fig 6b No Alkali Attack



1.4.8 Acceptance Testing Criteria

Although acceptance testing of refractories is not practiced in the cement industry, this is not the case in steel. Refractories are not, however, tested just for testings sake, but are viewed from the aspect of their costs and from the demands placed upon them, and quality control testing carried out accordingly. A principal objective of quality control is to prevent cases of failure in sensitive aggregates which in addition to immense financial consequences may in the steel industry also endanger human lives.

For the cheaper refractory grades, for less critical applications, the manufacturers data sheet specifications and quality control are accepted, with no essential input on the part of the user. Simple optical inspection of the state of the delivered materials is considered as being sufficient. Even for moderate cost refractory grades in service under moderately critical applications, testing is confined to carrying out spot checks, after delivery, on random samples, determining such conveniently measurable properties as dimensional accuracy, cold compressive strength. The results are collected to detect trends over long periods and are also if anomalous communicated to the supplier, for the lather's "information".

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Only for applications for which the refractory properties are critical are quality control examinations carried out systematically on a statistical basis. In such cases, the refractories are controlled prior to dispatch from the suppliers plant and checked according to specifications and tolerance limits previously agreed upon between supplier and client. The chosen test samples are normally divided into 2 sub-samples, one portion being tested in the supplier's laboratory, the other in the user's. This class of refractory is ordered well in advance of the specified delivery date e.g. 6 months, and this long range planning assures that the material to be supplied lies within the acceptance limits. The costs for statistically significant quality control are borne equally by supplier and user and often involve 3 - 4 man days working time for a quantity of 150 - 200 tons, the costs corresponding to 1.5 % of the delivery value.

1.5 ZONING of Cement Kilns

Because the refractories in the cement kiln have to perform several duties, as already listed in 1.1, no single refractory type can now fulfill all the requirements. As factors such as

- the changing temperature profile in the kiln
- the changing physical and chemical nature of the kiln feed ranging from raw meal to clinker

must be taken into consideration, the zoning of the kiln, with respect to the refractory lining, is a necessity. Indeed in some kiln zones it is normal to have two refractory types at any one position, namely a "working lining" and a backup insulating lining.

In a commonly established practice, the kiln zones are defined according to the changes occurring in the kiln charge as indicated in Table 13.

Table 13 Characterization of the Various Kiln Zones

Kiln Zone	Process	Material Change	Temp.Range
Drying	Wet and semi-wet	Expulsion of slurry or nodule moisture	≤100°C
Preheating	Wet and dry	Expulsion of chemically bonded OH groups	100-700°C
Calcining	"	Expulsion of CO_2 from carbonates	600-900°C
Transition	"	Formation of intermediate and clinker minerals	900-1350°C
Sintering	"	Granulation of clinker and formation of permanent coating	≥1350°C
Discharge	"	Cooling of clinker within kiln	1400-1250°C

Variations to the previous scheme include the definition of a chain zone (i.e. a combined drying and preheating zone) and the division of the transition zone into the safety zone and the transition zone, often described as the upper and lower transition zones.

In Figs. 7 and 8 a typical zone plan can be seen for wet and dry process operations with indications of the kiln temperature and various types of attack to be expected.

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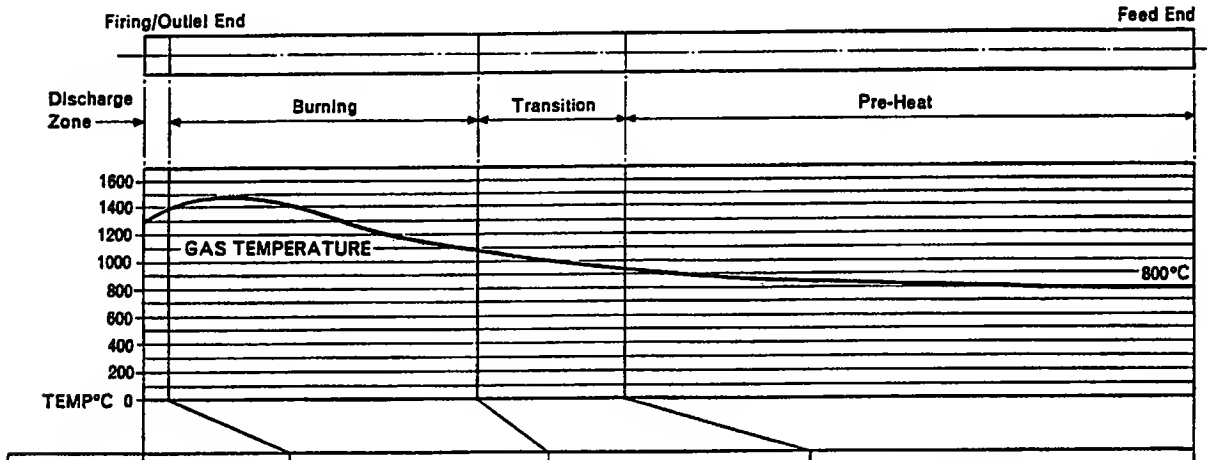
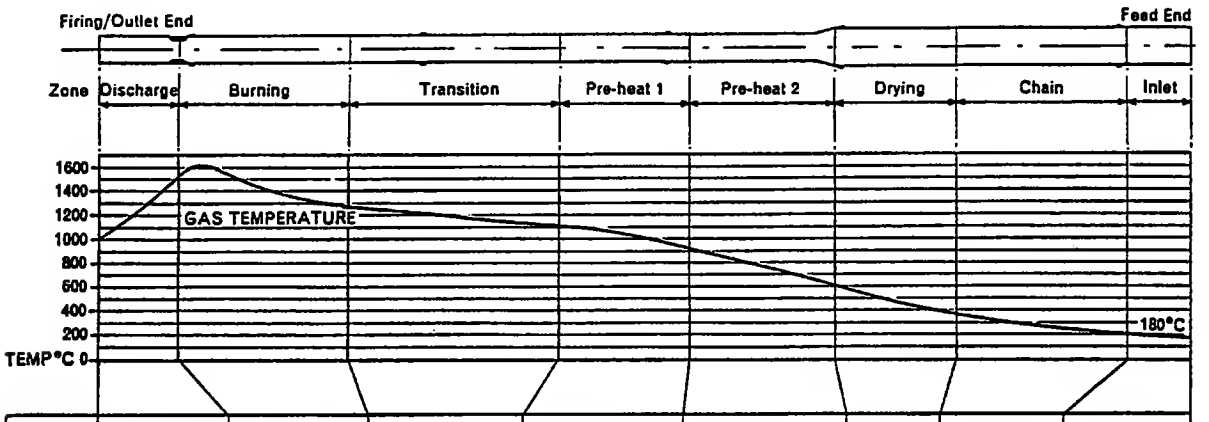
Fig. 7 Typical Rotary Kiln Data - Dry Process



Chemical Attack	Slight	Very Severe	Severe	Fairly Severe
Spalling Tendency	Very Severe	Very Severe	Severe	Slight
Abrasion Tendency	Very Severe	Severe	Fairly Severe	Fairly Severe
Refractory Quality	High-alumina brick Magnesite Silicon-carbide	Magnesite Magnesite-chrome Dolomite	Magnesite-chrome High-alumina brick	Fireclay brick Special light-weight refractory brick

Fig. 8 Typical Rotary Kiln Data - Wet Process



Chemical Attack	Slight	Very Severe	Severe	Fairly Severe	Slight	Slight	Slight	Slight
Spalling Tendency	Very Severe	Very Severe	Severe	Slight	Slight	Slight	Slight	Slight
Abrasion Tendency	Very Severe	Severe	Feirly Severe	Fairly Severe	Feirly Severe	Slight	Very Severe	Slight
Refractory Quality	High-alumina brick Magnesite Silicon-carbide	Magnesite Magnesite-chrome Dolomite	Magnesite-chrome	Alumina-rich brick	Alumina-rich brick	Fireclay brick Special light-weight refractory brick	Fireclay brick	Fireclay brick

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1.6 Wear of Kiln Refractories

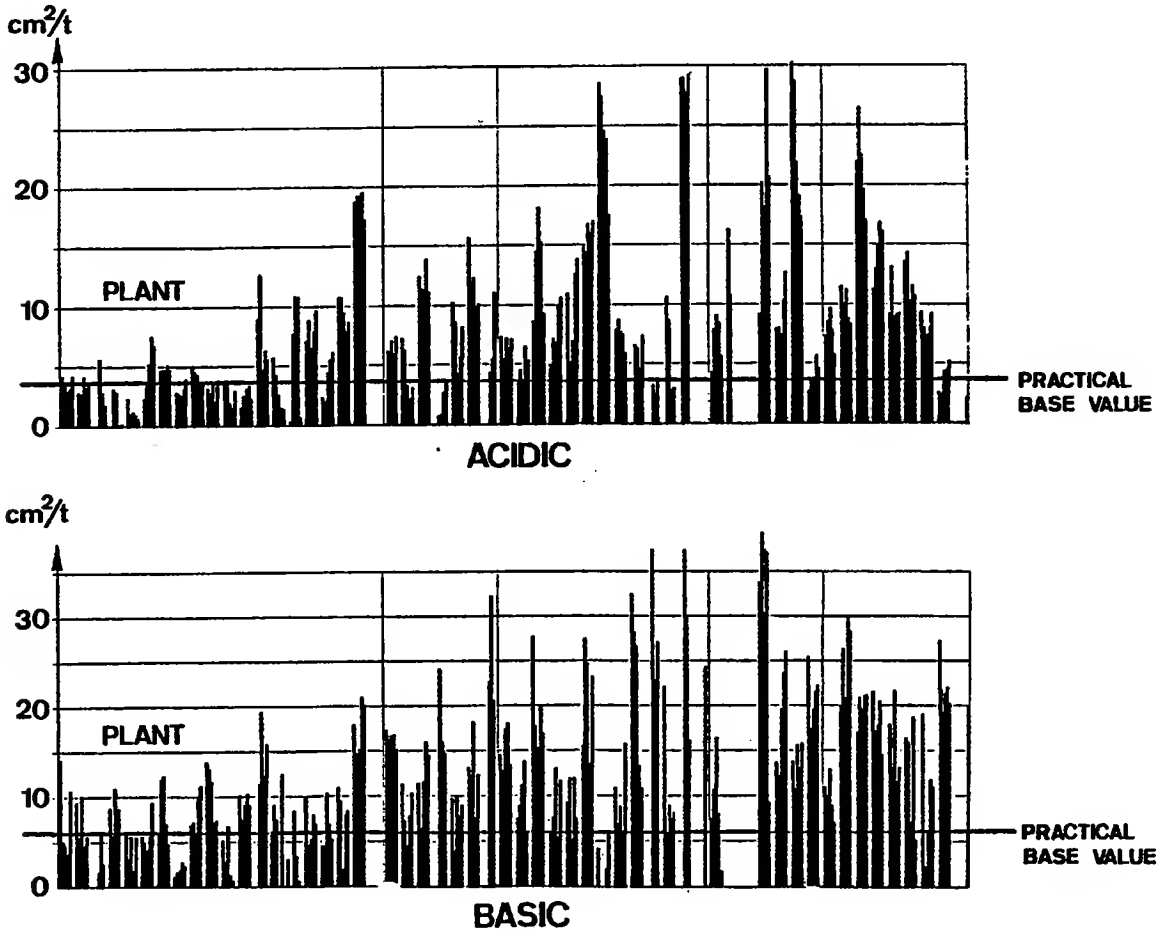
1.6.1 General

Even under the most favorable operating conditions, refractory linings are subject to a definite measurable rate of wear. Based on plant experience (Fig. 9), the minimum consumption rate achievable in practice, for basic and acidic types may be considered as follows:

- basic refractories ≈ 6 cm^2/t clinker
- acidic refractories ≈ 4 cm^2/t clinker

It should be considered that these values are average values for the total basic and acidic linings and that at various positions inside the kiln both higher and lower specific consumption rates are observed. Whereas some plants can consistently approximate to these desired low consumption rates, in most cases refractory consumption is higher with average consumption rates of 30 cm^2/t clinker and above being not uncommon.

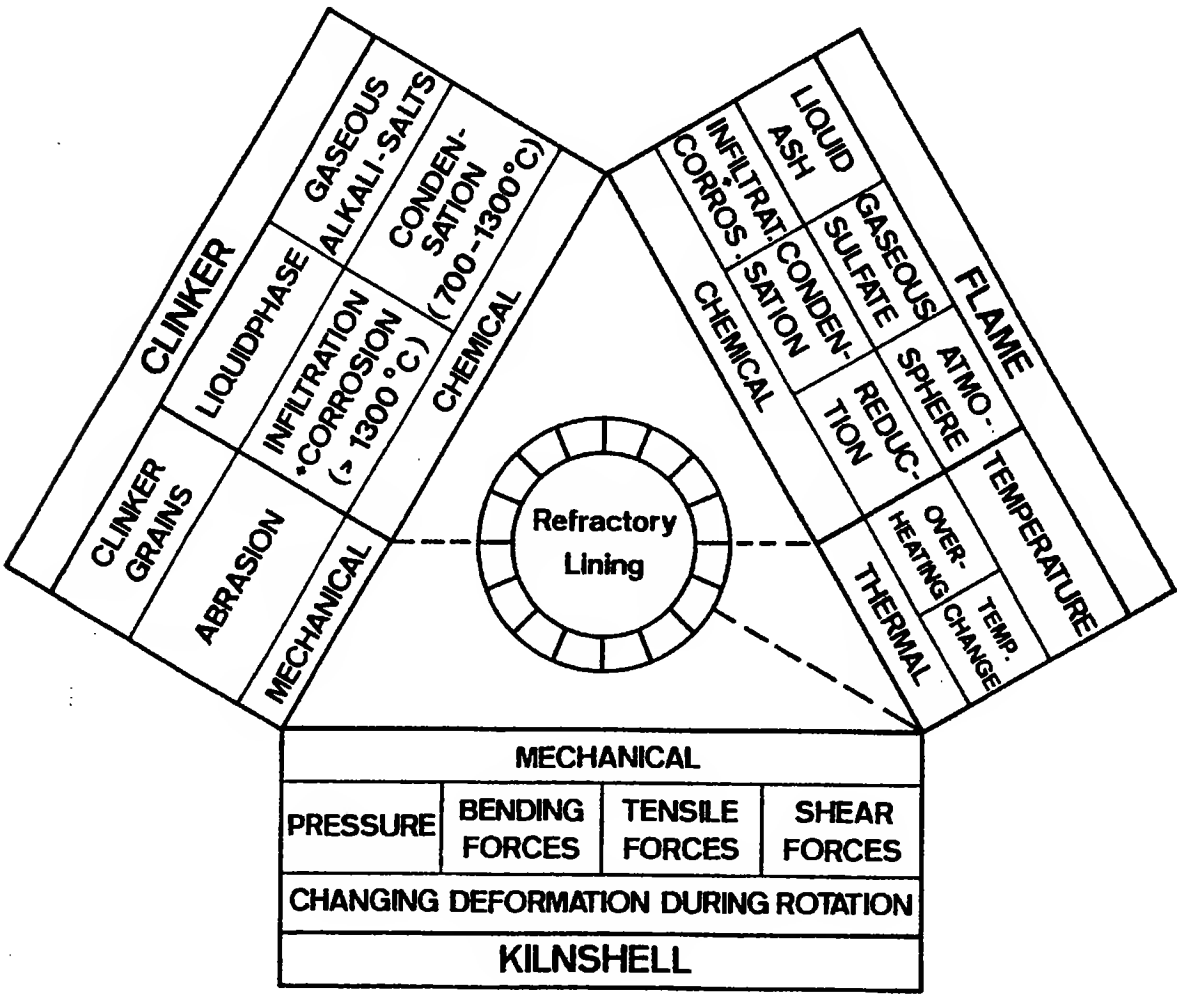
Fig. 9 Consumption of Kiln Linings



1.6.2 Causes of Wear

The principal causes of wear may be attributed to thermal, mechanical or chemical factors, acting either alone or more frequently in combination. Details concerning these various wear types are given in Fig. 10.

Fig. 10 Influence of chemical, thermal and mechanical factors on kiln refractory linings



Depending on the period for which observations are based it can be observed that the frequency of wear, due to any particular mechanism, is not constant but varies. This can be seen in Fig. 11 showing the frequency of the wear mechanisms from 1964 - 1968 and from 1968 - 1974, i.e. since the introduction of large diameter process kilns. As can be deduced, the identified causes of wear before 1968 were principally chemical (silicate and alkali infiltration) in nature with mechanical and thermal factors being of equal but secondary importance. Between 1968 and 1974 mechanical stresses became by far the principal failure cause, with chemical and thermal being of equal importance. Whereas before 1968 the causes could clearly be identified in only 50 % of the investigated cases, the percentage of those not identified dropped to 25 % between 1968 and 1974.

Although no statistics are presently available, the principal causes of wear may again be chemical in nature, originating from silicate attack and reducing conditions stemming from the widespread reintroduction of coal firing.

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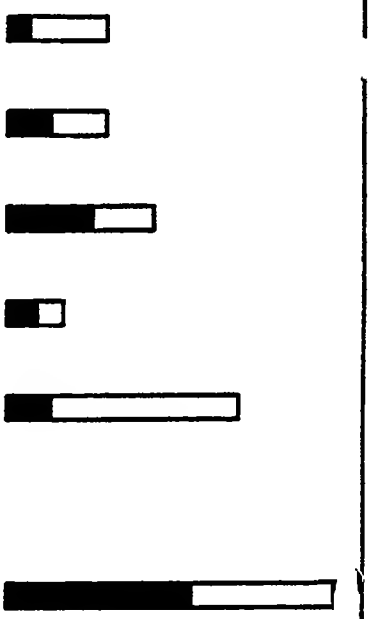
Fig. 11 Main Causes of Wear In Sintering Zones

EXAMINATIONS BEFORE 1968 ■ AND 1968-74 □

CAUSE OF WEAR	EXAMINATIONS NUMBER	%
OVERHEATING	10	10
SILICATE INFILTRATION	11	10
ALKALI INFILTRATION	16	15
THERMAL SHOCK	8	8
MECHANICAL STRESS (THERMAL EXPANSION• KILN SHELL)	26	24
MISCELLANEOUS (NOT DETERMINED, CANNOT BE CLASSIFIED)	35	33
	106	100



1.6.3 Mechanisms by which Damage Occurs

The chemical, thermal and mechanical factors responsible for wear cause changes to occur in the refractory materials by means of material transport stress and erosion. This results in either a modification or a destruction of the refractories texture leading to discontinuous wear by spalling, or to a continuous wear by abrasion. The relationship between these factors are indicated in Fig.12.

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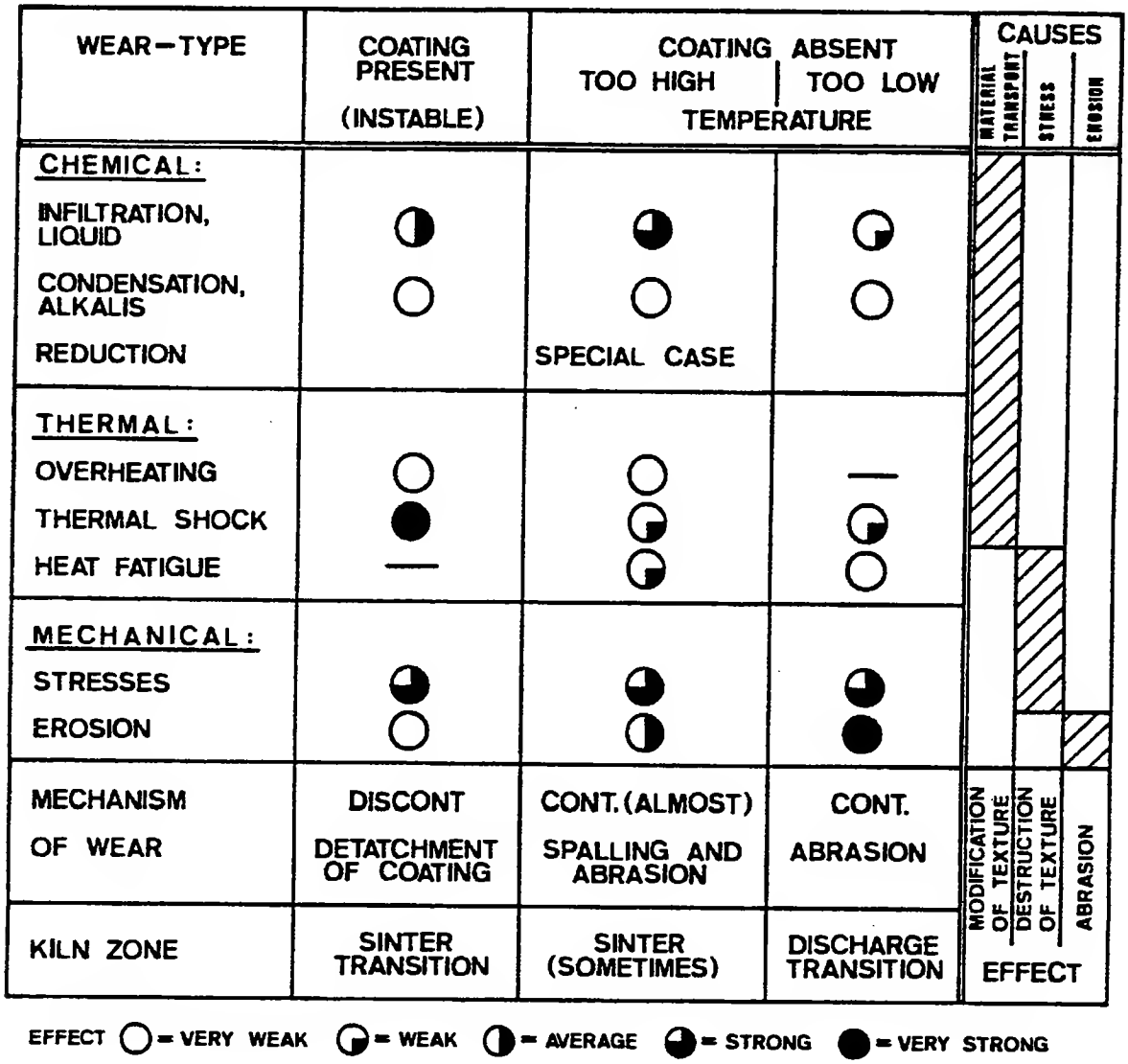
Fig. 12 Characterization of Wear Magnesite-Type Linings



1.6.4 Recognition of the Various Wear Causes

To help identify the cause or causes of refractory wear, a range of characteristics can be proposed summarizing the changes resulting from the various attack mechanisms. A list of these is presented in Tables 14a and 14b. As can be seen, many of the changes can be observed macro or microscopically or by chemical analysis, and so a preliminary conclusion as to why refractories have failed can often be drawn by plant personnel.

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Table 14a CHARACTERISTICS OF THE VARIOUS DAMAGE MECHANISMS FOR ACIDIC AND BASIC BRICKS

- Chemical Damage
  - *Liquid infiltration* (clinker / coal ash
    - lower porosity due to penetration
    - change of colour
    - change of mineralogy
    - increase in CaO, SiO_2, (Al_2O_3, Fe_2O_3)
  - *Alkali condensation*
    - increaes in Na_2O, K_2O, Cl, SO_3 levels
    - porosity decrease
    - presence of light coloured layers
    - alkali bursting (Al_2O_3 refractories)
    - change of mineralogy
  - *CO_2/SO_3 attack (on CaO containing compounds)*
    - decrease of porosity
    - presence with whie coloured layers
    - increase in CO_2, SO_3
    - change of mineralogy

Table 14b CHARACTERISTICS OF THE VARIOUS DAMAGE MECHANISMS FOR ACIDIC AND BASIC BRICKS

- Thermal damage
  - *Overheating*
    - melting of hot face
    - change (increase / decrease) in type of porosity
    - elongation of periclase crystals
    - reduction of chrome content
    - diffusion of low melting silicates, aluminates
    - change of colour
  - *Shock*
    - spalling of hot face
- Mechanical damage
  - *Kiln deformation / thermal expansion*
    - porosity unchanged
    - chemistry / mineralogy unchanged
    - spalling over wide areas
    - crack formation on micro to macro scale

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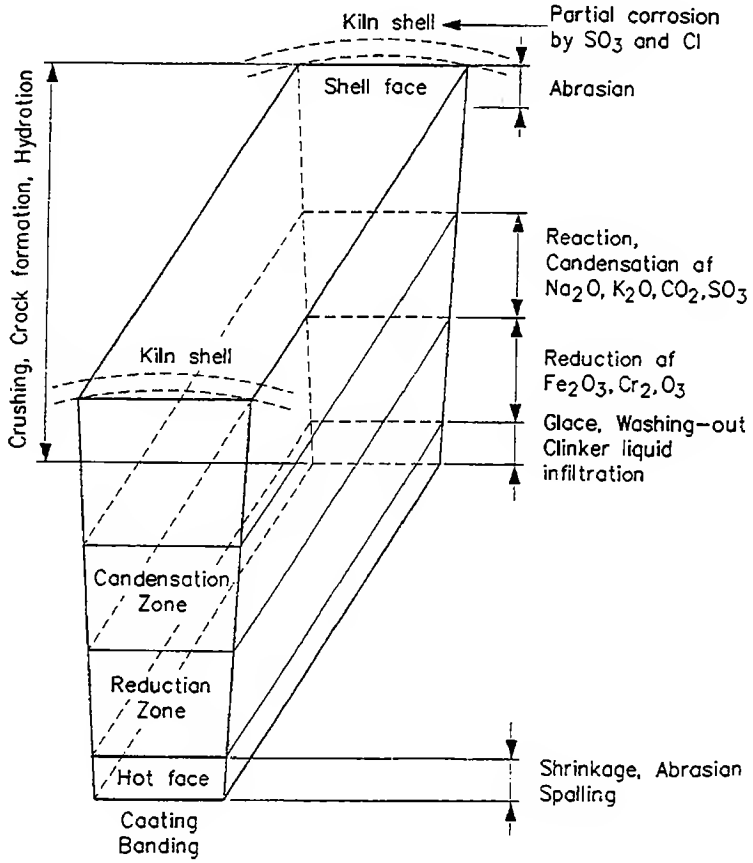
Appendix 2: A graphical representation of the damage zones



1.7 Chemical Reactions Producing Refractory Damage

The chemical reactions occurring when refractories are attacked by siliceous or alkali sulfate infiltration's or by reducing atmospheres have been thoroughly investigated. Examples will now be given of those reactions occurring at various positions in the kiln.

1.7.1 SO_3 Attack on Castables in Heat Exchangers

The bonding agent of many refractory castables is monocalcium aluminate (CA) which exhibits both hydraulic properties (e.g. high alumina cement) and a high melting point (> 1600°C). At service temperatures in the range of 800 1000°C, CA reacts with SO_3 from the kiln gases and on cooling (during shutdowns) a further reaction with moisture can occur.

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The foil owing reactions have been demonstrated to take place:

- In Service
  $2\ CaO.Al_2O_3 + SO_3 \rightarrow CaO.2Al_2O_3 + CaSO_4$ (fast)
  (monocalcium aluminate) → (calcium dialuminate) + (anhydrite)
  $CaO.2\ Al_2O_3 + SO_3 \rightarrow 2\ Al_2O_3 + CaSO_4$ (slow)
  (calcium dialuminate) → (alumina) + (anhydrite)
- During Shutdowns
  $CaSO_4 + 2\ H_2O \rightarrow CaSO_4 . 2\ H_2O$
  (anhydrite) → (gypsum)

As a consequence of all these reactions their occurs

- a destruction of the bonding and a change of structure
- a loss of strength

which are claimed as promoting conditions beneficial for the enhanced adhesions of build-ups.

1.7.2 Alkali Bursting of Alumina Refractories

The most well known method by which high alumina bricks are damaged is the so-called alkali-"spalling" or "bursting". By the pick up of alkalis, the formation of feldspathoids takes place which, because of an increase in volume, causes a disruption of the brick at a given depth below the hot face. The ensuing cracking over a large area causes the heads of the brick to spall off. For the various components of high alumina bricks different volume changes occur.

- Mullite Based Refractories
  $3Al_2O_3\,2SiO_2 + 3\ K_2O + 4\ SiO_2 \rightarrow 3\ K_2O.Al_2O_3 . 2\ SiO_2$
  (Mullite) → (Kalsilite)

ΔVolume = + 29 %

- Bauxite Based Refractories
  $Al_2O_3 + K_2O + 2\ SiO_2 \rightarrow K_2O.Al_2O_3 . 2\ SiO_2$
  (Bauxite) → (Kalsilite)

ΔVolume = + 2.5 %

As a consequence, should it be necessary to install high alumina refractories in zones in which the danger of alkali spelling occurs (e.g. transition zone in the vicinity of the kiln type), preference should be given to a bauxite rather than mullite based product.

1.7.3 Reducing Atmosphere Attack in Magnesite Refractories

The occurrence of local reducing conditions in the burning and transition zones can, when no protective coating is available, bring about a dramatic loss of brick strength. Damage by this mechanism has been more frequent since the reintroduction of coal firing and has been one of the factors decisive in the necessity to develop new Fe_2O_3 and Cr_2O_3 free magnesia refractories i.e. magnesia-spinel products.

Typically, this type of chemical attack takes place on the Fe-containing magnesia ferrite ($MgO . Fe_2O_3$) component, present as a precipitate within the periclase crystals, when the refractories have been in service at approx. 1100°C over an extended period in an oxidizing atmosphere.

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Chemically, the following reaction can be considered to occur:

$MgO.Fe^{III}_2O_3 \rightarrow (Mg/Fe^{II})O$ ss
(Magnesia Ferrite) → (Magnesia Wüstite)

Δ Volume = -20 %

The reduction in volume leads to an increase in porosity with a resulting decrease in strength due to the fo rmation of voids.

1.7.4 Gaseous Attack on Dolomite Bricks

Although they give excellent service when protected by a permanent coating, without this protection, the reactive free CaO component of dolomite refractories is very susceptible to reaction by various reagents, all of which result in an increase in volume. Examples of these include:

Reaction	Volume Increase
$CaO \rightarrow CaS$	+ 155 %
$CaO \rightarrow Ca(OH)_2$	+ 200 %
$CaO \rightarrow CaCO_3$	+ 221 %
$CaO \rightarrow CaSO_4$	+ 275 %
$CaO \rightarrow CaCl_2$	+ 313 %

The MgO component remains in comparison much less reactive to gaseous attack.

Depending on the temperature conditions and the kiln atmosphere, the following reactions occur in the various sections of a dolomite lining:

- Middle Brick Zone
  $CaO + SO_3 \rightarrow CaSO_4$
  $CaO + S^{-2} \rightarrow CaS$
  $2\ CaO + 3\ SO_3 + K_2O \rightarrow 2CaSO_4.K_2SO_4$
- Lower Brick Zone
  $CaO + C0_2 \rightarrow CaCO_3$

Because of the increase in volume at a specific temperature and hence depth the increase in volume results first in the formation of cracking and then in the spelling of the hot brick face.

Dolomite bricks are also subject to hydration in air when their temperature is under 450°C. Because used bricks have different concentrations of newly formed minerals, their hydration rate is not the same over the whole structure. It has been shown that the middle CaS containing zones hydrate faster than the hot face, which in turn hydrates faster than the hot face, which in turn hydrates faster than the cold one. This is shown in Fig. 13 showing the rate of destruction of a single used dolomite brick exposed to laboratory air at 22°C and 65% r.h.

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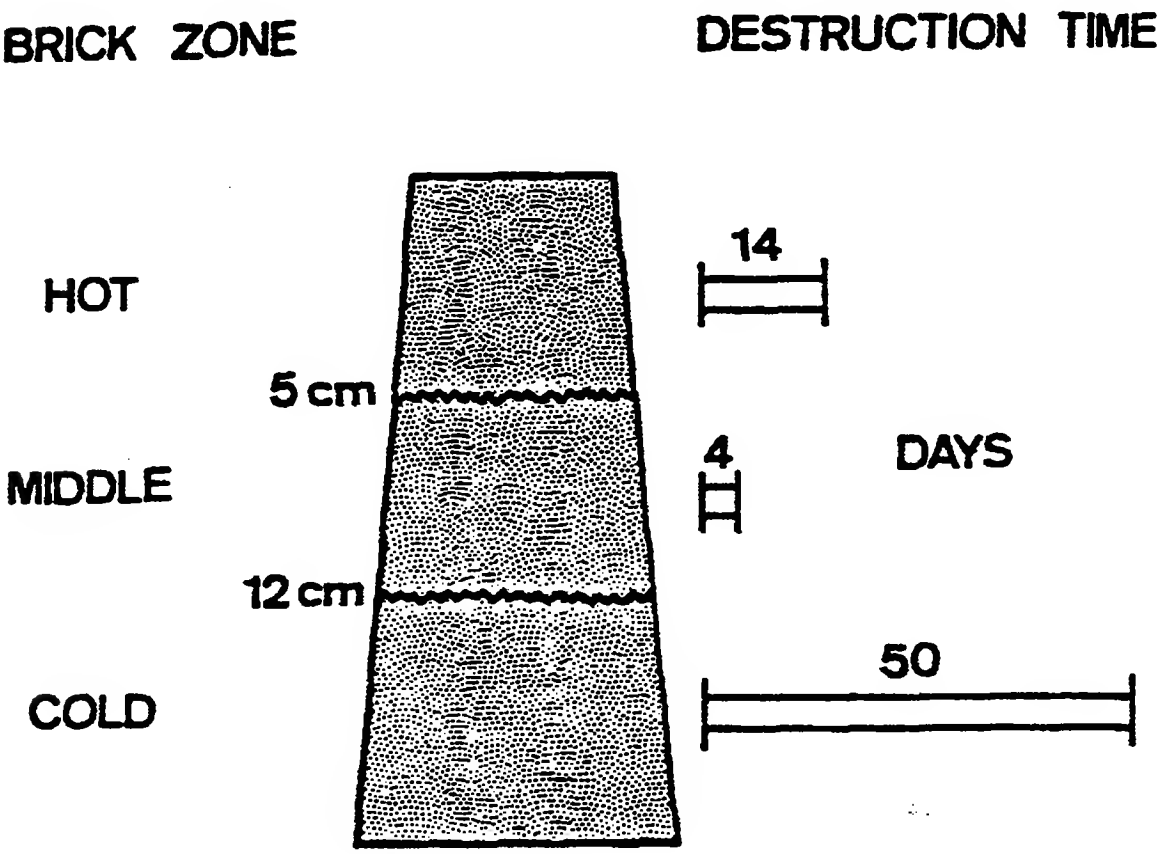
Fig. 13 Destruction rates of Dolomite Brick Zones



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2. COATING: PROPERTIES AND BEHAVIOR

2.1 General

The formation of durable, protective coating on the refractory lining of a cement kiln is desirable, in a technical as well as in an economical respect, to guarantee steady kiln operations.

Basically, it is known that the properties and behavior of coating have to be regarded as the result of a combination of constructional, process technological and material inherent parameters. The three partial-problems:

- Formation of coating
- Adhesion and strength of coating
- Prevention of ring formation

are closely related to the material parameters as

- Refractory type
- Clinker dust and ash characteristics
- Gas phase and volatile elements

and to the constructional/operational aspects as:

- Kiln system
- Kiln operation
- Operating personnel

2.2 Coating

2.2.1 Definition

The term "coating" is applied in the cement industry for different phenomena. It is therefore necessary to distinguish between the following:

- Coating in the cement kiln is the desired protective layer on the refractory lining, which is formed during kiln operation under the influence of high temperature from the traveling material on the refractory lining in the transition and sinter zones.
- Ring formation is an undesired degree of growth of coating.
- Incrustations, referred to as build-ups, are "coatings" which are formed below 1100°C by alkali salts and which occur particularly in pre-heater systems.
- Refractory lining encompasses all construction materials which are intended to eliminate or reduce the effects of thermal strain.

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2.2.2 Significance of Coating

Coating has the following functions to fulfill:

- Protection of the refractory brick against overheating and abrupt temperature changes (improvement of heat distribution).
- Protection of the refractory lining against the abrasive effect of the moving clinker bed and the dust.
- Protection of the refractory lining against chemical attack of clinker constituents, fuel ash and gas atmosphere.
- Augmenting the mixing effect, i.e. improvement of the blending effect of the clinker mass due to unevenness of the coating surface.
- Reduction of heat radiation and protection of the kiln shell against thermal overloading.

According to experience with large kilns, coating characteristics and behavior were found to be the most significant parameters associated with steady kiln operations. The operating life of refractory linings is substantially longer, when stable coating is achieved over the whole length of the kiln.

Protective coating begins normally in the transition zone which is also the zone, where ring formation may frequently occur.

2.2.3 Formation of Coating

Two basic hypothesis are plausible as to the mechanism of coating formation:

- Formation of clinker melts (above cat 1280°C) and their sticking together with the refractory lining with simultaneous cementing of solid particles.
- Adhesion of solid crystalline phases on the refractory lining.

In both cases, chemical and/or physical reactions (impregnation/infiltration) with the upper zone of brick strengthen the adhesion of the coating.

The temperature gradient of coating refractory lining kiln shell effects a further stabilization of the coating as a result of cooling and crystallization. The formation of coating increases to that point at which its inherent insulation characteristics prevent further growth.

2.2.4 Thickness of Coating

Usually, the following classification is applied as to the thickness of coating:

30 - 50 cm	thick coating
~ 20 cm	normal coating
0 - 10 cm	thin coating

Coating thicker than 50 cm is classified as ring formation.

2.3 Influencing Parameters

Coating is preponderantly formed from the components of raw material and clinker, whose characteristics and composition primarily determine the quality and quantity of the coating. In addition, the type of fuel and diffusion products of the refractory lining also exert a certain influence.

2.3.1 Chemical Composition

Industrial experience demonstrates that a correlation exists between the characterizing moduli (silica ratio, alumina ratio, lime saturation) and the rate and ability of coating

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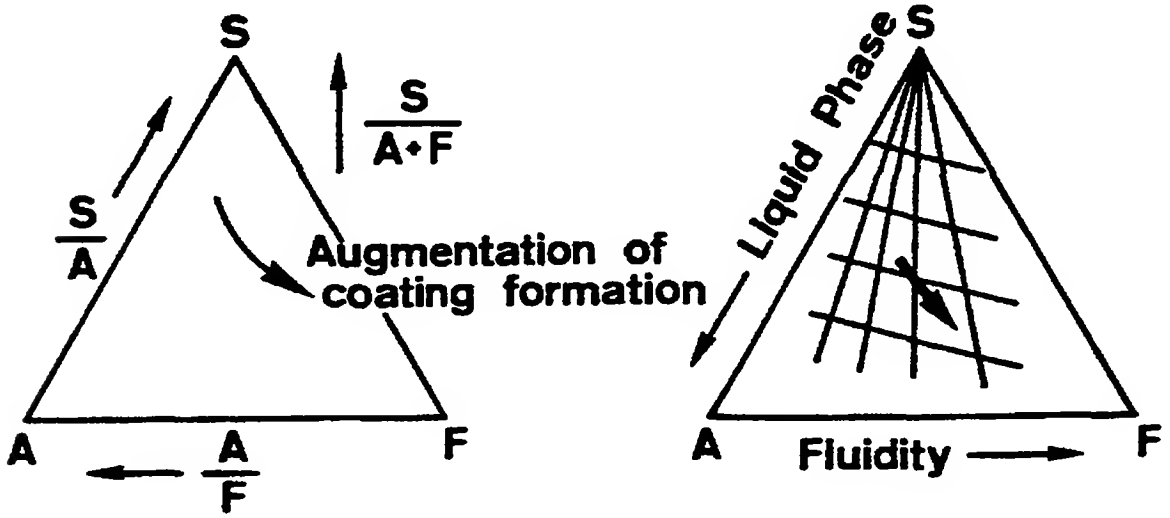
formation. Within the system SiO_2 Al_2O_3 Fe_2O_3, i.e. S A F, the dependence of coating formation and chemical composition has been demonstrated as occurring in Fig. 14.

Fig. 14 Correlation of coating formation and chemical parameters



Coating formation is facilitated by decreasing the ratios A/F and S/A.

In a second triangle (right) indicating two series of lines, viz. equal percentage of liquid phase at 1450°C and equal fluidity of the liquid phase it is demonstrated that, disregarding the lime saturation, the tendency to coating formation increases with increasing proportion and fluidity of the liquid phase.

The liquid phase of a clinker is defined as that material present in liquid form, after beginning of the formation of clinker minerals. This usually first forms at temperatures of approximately 1250°C. This liquid phase content at 1450°C may be up to 35 %, but on average lies around 25 %.

The quantity of liquid phase and its physical properties (viscosity, surface tension, etc.) are dependent on chemical composition and temperature.

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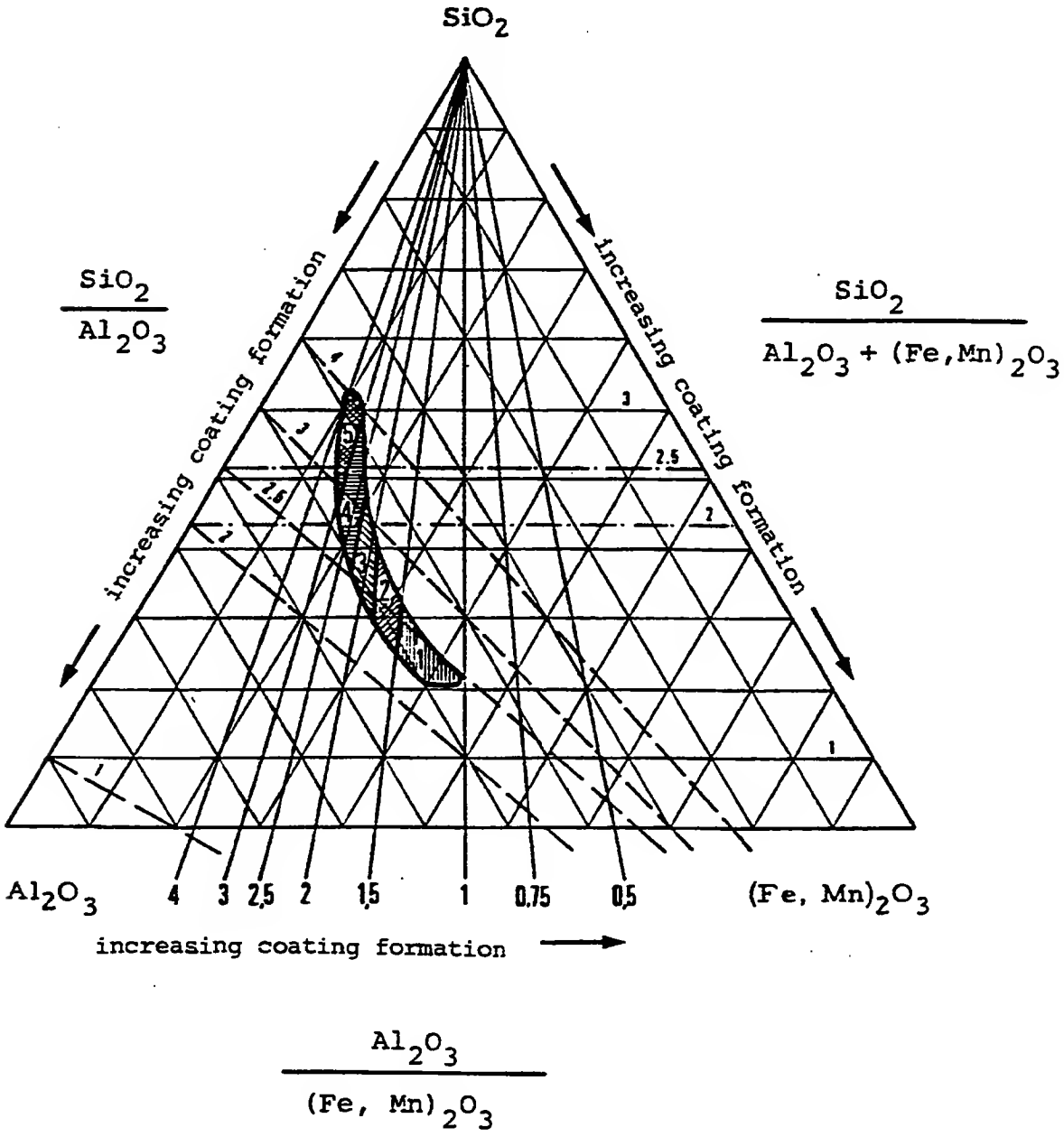
From experience, a correlation is known to exist between the quantity of liquid phase (C_3A + C_4AF + MgO + K_2O + Na_2O + SO_3) and the thickness of the coating. If a raw mix with less than 20 % potential liquid phase is employed, the formation of coating in the burning zone is expected to be weak, whereas liquid phase contents of more than 25 % lead to thick coating and frequent ring formation.

Another display divides the field of potential coating formation into five areas (Fig. 15).

Fig. 15 Clinker composition and coating formation as influenced by the ratios of SiO_2 Al_2O_3 $(Fe, Mn)_2O_3$



Coating formation is subdivided as follows:

field 1	strong
field 2	medium strong
field 3	medium
field 4	low
field 5	very low

Generally, the following can be deduced from Fig. 14:

- Increasing alumina ratio and SiO_2/Al_2O_3 ratio impair coating formation and vice-versa.
- Favorable coating conditions are achieved with a SiO_2/Al_2O_3 ratio of 2.5 to 3.3 and an alumina ratio of 1.8 to 2.3.
- Favorable coating conditions are achieved at constant medium or high SiO_2/Al_2O_3 ratios, by lowering the alumina ratio.
- Extremely SiO_2/Al_2O_3 high ratios necessitate extremely low alumina ratios.
- Main influencing parameter is the alumina ratio.

In addition, a number of other measures to improve the tendency to coating formation have been proposed:

- Lowering of the $CaCO_3$ content in the raw mix
- Addition of CaF_2 (fluorspar)
- Addition of pyrites ash

etc.

However, the chemical characteristics of the raw material should not be over-emphasized, since the characteristics of the refractory lining represent additional, determining parameters as to coating formation.

2.3.2 Mineralogical Composition

Experience shows that raw mixes containing elevated clay mineral contents and low quartz contents lead to better coating ability than raw mixes of reversed mineralogical composition. Since the mineralogical composition influences the ratios and liquid phase contents, the chemical composition is already a conclusive criterion.

2.3.3 Homogeneity of Raw Mix

Inhomogeneity of the raw mix composition is normally responsible for fluctuations in coating formation. This applies not only to chemical inhomogeneity, but also to the variations of the raw mix fineness and to the characteristics of the fuel.

The effects of changes of raw mix properties on the already formed coating layer may be deduced with the aid of Fig. 16.

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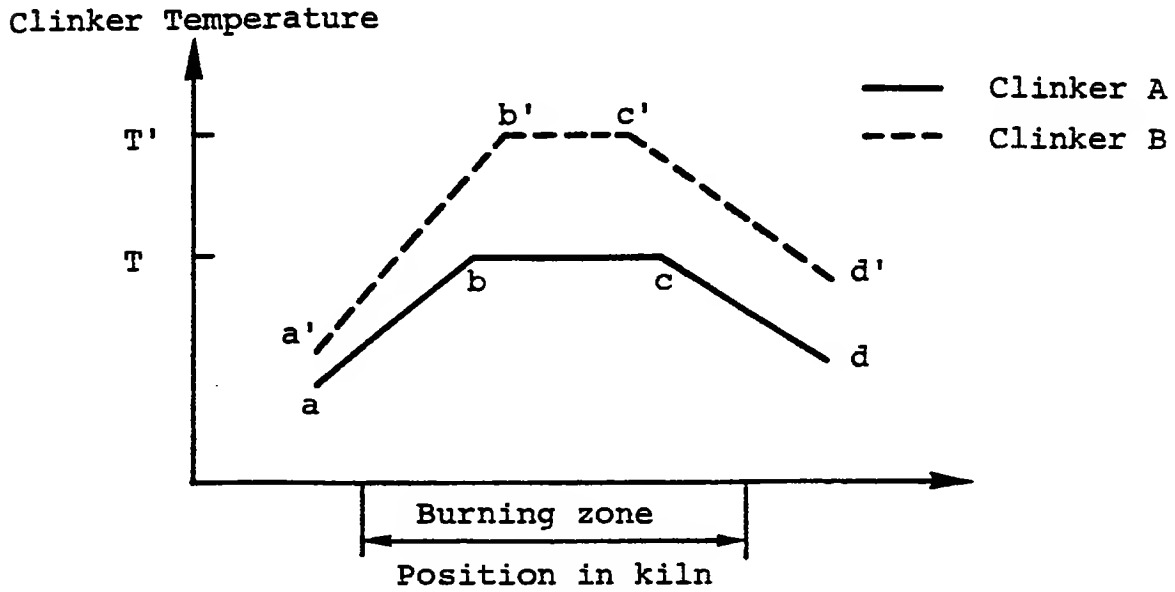
Fig. 16 Schematic Temperature profile of kiln clinker as a function of material properties



Clinker A of good burnability upon up-heating along a - b enters the burning zone at temperature T (point b). It remains during clinker formation at this temperature and then cools down, along the temperature profile c - d. During the contact of clinker A with the refractory lining, a layer of coating is assumed to have been formed.

Any change of the raw mix composition, which impairs its burnability, often leads to a change of clinker melting properties. For the more difficult to burn clinker b let us assume that an increased amount of energy will be necessary resulting in a new temperature profile a' - b' - c' - d'. With this changed profile, the already formed coating layer becomes softer, more fluid, and is easily scraped off by clinker B. If the refractory linings are already worn out, the kiln shell will become hotter and perhaps even red spots may occur. In any case, the refractory lining in the burning zone will be weakened as a result of the new temperature at the brick surface. Frequent changes and loss of coating (due to fluctuation of composition of raw mix) particularly in big kilns often results in a considerable decrease in refractory life caused by spelling.

2.3.4 Circulating Elements

Circulating elements are the alkalis (K_2O, Na_2O), chlorine (Cl^-) and sulfur (SO_2, SO_3, eventually S^{2-}). They evaporate from the raw materials (raw mix and fuel) and condense on the colder raw mix and penetrate into the refractory lining. They play a significant role with regard to the stability of coating and refractory lining life, but influence coating formation to a lesser extent.

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Alumina-rich refractory bricks are easily infiltrated by alkalis which react with the brick constituents forming minerals in this zone, such as leucite and kalsilite. These minerals crystallize upon cooling and lead to spelling of the brick surface. Such reactions don't occur between the alkalis and the basic bricks. Corrosion of bricks is also observed, if sulfates are present in notable amounts. The destructiveness of alkali sulfates (and chlorides) on coating is caused by the following mechanism:

- The sulfate-melt penetrates and becomes enriched in the pores and cracks of the brick, the position of enrichment depending upon the thermal gradient.
- Periodic changes of the temperature gradient produce both melting and recrystallization, which results in crack formation vertical to the temperature gradient, due to pressure of crystallization.

Such an infiltrated zone is the cause for the spelling of 4 - 6 cm thick layers of brick and adhering coating. As a rule of thumb it can be said that large quantities of alkali and sulfate are always especially harmful, if a molar excess of sulfate exists.

2.3.5 Crystal phases

As already discussed, coating formation is mainly considered as being controlled by the properties of the clinker liquid phase.

However, there is also another possibility to be considered: the compaction of powdery materials at high temperatures by sintering or recrystallization. This effect may be explained by the preferential grow of bigger crystals at the expense of smaller ones, a diffusion process which takes place at temperatures below the melting points of the substances.

In nearly all cases in laboratory tests as well as under plant conditions a small layer of C_2S, less than 1/2 mm in thickness can be found in the contact zone between brick surface and coating, whereby the crystals of C_2S in this zone seem to be glued together without interstitial phases. The C_2S zone extends into coating as well as into the upper parts of the (porous) brick to about 1 mm in depth, thus connecting both, brick and coating.

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SIFCA™, which is an acronym for **S**lurry **I**nfiltrated **F**ibre **C**astable, has some unique properties, which allow it to successfully replace both refractories and heat resistant cast irons in many difficult applications. **SIFCA™** has extended the life and reduced the cost dramatically when used in such applications as

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I look forward to hearing from you and to co-operating with you in future in the supply of your requirements

David A. Jarvis Managing Director Thermatex Wahl Europe Limited Redhill House Hope Street Chester CH4 8BU Telephone/ Fax 0124 465 9909 Mobile 0771 421 8712

Chapter 4

Rings, Balls and Build-Ups

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Rings, Balls and Build-Ups

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1. INTRODUCTION

Rings and deposits are accumulations of solid materials (from the powdery kiln charge) in the rotary or static sections of clinker production lines. They have been encountered since the earliest days of rotary kiln production, with each development in process technology, e.g. grate and cyclone pre-heaters, grate cooler, bringing with them their own specific type of deposit.

Rather than being of academic interest, ring and deposit formation has an appreciable influence on plant operations, frustrating operations personnel by their impairing or even impeding production, and annoying the company management by lowering production (and sales) and increasing production costs.

As a direct consequence of rings and deposits, the gas and material flow through the kiln is restricted, resulting in a reduced kiln output. Especially in the sinter zone, the presence of rings can interfere with combustion of the fuel and can result in improper combustion. From time to time unstable rings and deposits can break away leading to blockage or mechanical damage in the cooler, or in cyclone blockages. The partial shedding of coating from the exhaust fan blades results in severe vibration which mostly requires a short shutdown for complete removal. The breaking of a ring almost always causes a flush of material into the burning zone and a temporary loss of stable kiln operations.

The formation of deposits in cyclones results in extra costs for the labour needed to remove the deposits by poking. The introduction of air canons (big blasters) provides a method for their regular automatic removal and has been installed in Group plants with persistent pre-heater blockages. High pressure water jets may also be employed.

In the worst cases, a complete shutdown is necessary to allow entrance to the affected area and mechanical removal of the blockage with compressed air drills. This shutdown invariably weakens the sinter zone refractories, and accelerates the next shutdown for rebricking.

2. LOCATION OF RINGS

2.1 Classification

Unwanted build-ups may be classified with regard to the type of material from which they are formed, either sintered or unsintered. Within these two groups the various types can be classified as follows:

unsintered:

- exhaust fan deposits
- cyclone and grate preheater deposits
- slurry or mud rings
- meal rings

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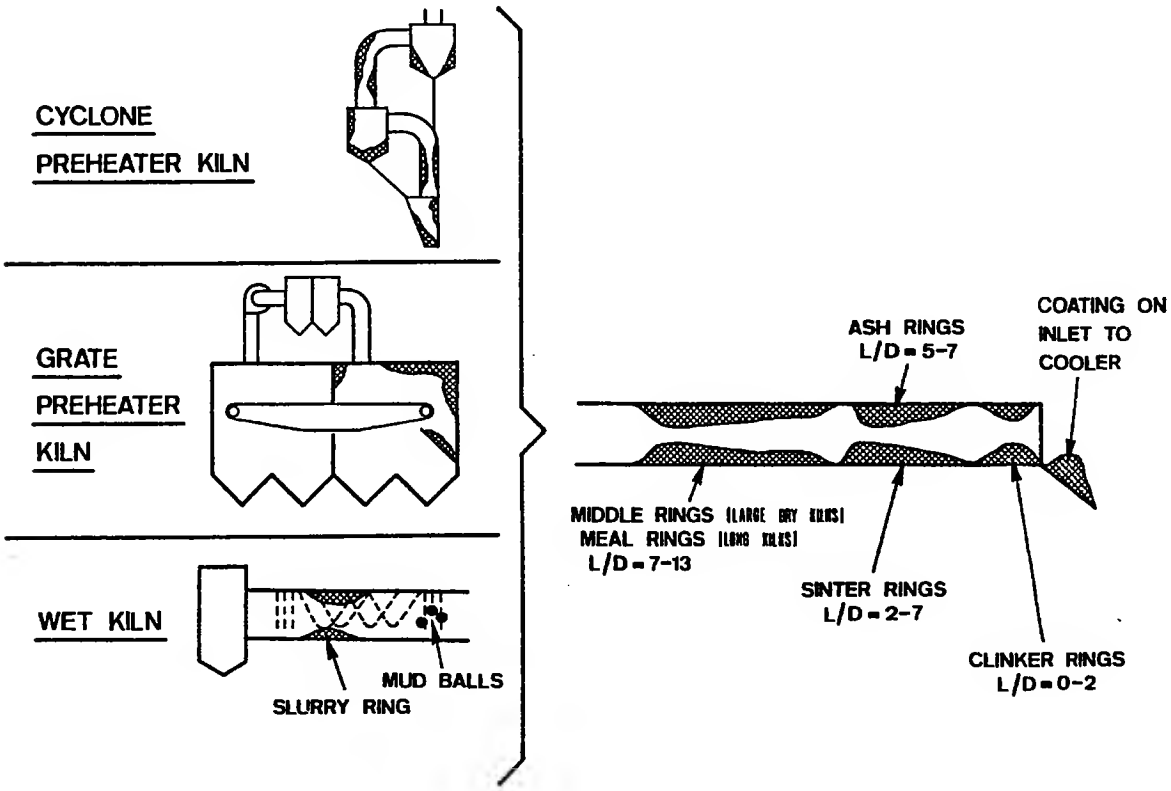
sintered:

- middle ring
- sinter ring
- clinker ring
- "snowman" in grate cooler
- kiln charge ball

Process technological characteristics of such build-ups e.g. kiln type, location, temperature of gas and kiln charge can be seen in Table 1. Material technological characteristics e.g. state of kiln charge, enrichment in various elements, and type of texture are summarised in Table 2.

The location of the various types of the above rings and deposits can be seen in Fig. 1.

Fig.1 Rings and Build-ups in Different Kiln Systems



3. THEORETICAL ASPECTS OF RING AND DEPOSIT FORMATION

Although of much practical significance, little quantitatively based, fundamental knowledge is available on the formation of deposits from solids suspended in gas streams. In a qualitative way, however, the more important features of such processes are known.

The formation of a deposit is always a dynamic process in which the factors responsible for formation outweigh the forces of degradation. In general, the stronger the forces of destruction, the more unlikely the chance of deposit formation, but when this does occur, a strong, hard to remove agglomeration is the result.

After the transport of material to the area of deposition, a definite force is required to make it adhere to the wall. This can range in magnitude from that caused by turbulence within the gas stream, increasing to centrifugal forces when the stream changes direction, to that due to mechanical pressure. Whereas pre-heater deposits involve the first two, mechanical pressure certainly plays a part in ring formation within the rotating kiln.

The forces according to Rumpf considered to cause deposit formation can be grouped as follows:

a) melting or softening of surface due to friction or collision

b) melting or freezing due to addition or removal of heat

c) interlocking of aggregates built up of finer particles held together by surface forces

d) interlocking of long fibrous particles

e) electrostatic attraction

The mechanisms b, c and d are the ones encountered in kiln operations. In general, the finer the powder, the greater the tendency towards agglomeration, and in many cases the absence of particles under a critical size (e.g. 5 um) ensures freedom from deposition.

4. CHARACTERISTICS OF VARIOUS RING AND DEPOSIT TYPES

Tables 3 and 4 contain a list of typical properties of rings and build-ups encountered within the "Holderbank" Group plants (with full chemical analysis being provided in Appendices I, II and III). Included are such factors as volatile element concentrations and moduli of the deposited materials. As an indication of the texture, the size of the pores and the particles or aggregates of particles, from which the materials were built up, is given. In many cases the mineralogical composition is also given.

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4.1 Exhaust Fan Deposits

In the case of kilns with pressure filter systems, in which unfiltered dust-laden gas passes through the exhaust gas fan, deposit formation causes problems. These arise when the deposit falls off one blade, and brings the rotating fan out of balance. Deposits of up to 3.5 kg/blade are known to occur.

4.1.1 Characteristic properties:

Exhaust fan deposits, composed of the finest raw meal particles are usually red-brown, hard and quite brittle. They exhibit a compact layered structure and have a very low porosity of 8%.

Their chemical and mineralogical composition is basically that of the raw meal but often the plate-shaped clay particles are preferentially deposited parallel to the blade surfaces. Due to their long stay in the system, fan deposits are enriched in the volatile components K_2O, Na_2O, SO_3. Typical values include the following:

LSF	20 - 100
SR	1.0 - 1.5
$K_2O + Na_2O$	2.1 -3.0 %
SO_3	4.7 - 6.0 %

The SO_3 is usually present as anhydrite ($CaSO_4$).

Binding mechanism:
In this case, the temperature is such that liquid phase involvement - aqueous or molten salts - can be ruled out.

The dust particles, because of the fan rotation, strike the blade surfaces with a high velocity and are so compacted. As the texture of the surface, after even a short time in operation, possesses undulations in the order 0.5 - 20 um, the smaller dust particles can be mechanically "locked-on". Subsequent development of the deposit follows by an identical mechanism.

4.2 Slurry Rings (including mud balls)

Characteristic properties:

These occur in long wet kilns and are composed of the partially dried kiln charge somewhat enriched in alkalis and SO_3. They are soft and can usually be broken - and hence prevented by - heavier chains. The H_2O content lies between 20 and 30%, a range in which clay materials exhibit a sticky, plastic consistency. The content of the alkalis which greatly increases the tendency to adhesion (influence on rheological properties) can be up to 10% $K_2O + Na_2O$, and about the same level of SO_3. In many cases, balls form (in addition) on the chain links by the same mechanism. A typical example of a mud ball is plant I in Table 4.

Binding mechanism:

The binding mechanism is the well known ability of clays to form a sticky, plastic mass when containing the correct quantity of H_2O, and to harden on the further water loss. To this mechanism must also be added the crystallization of K_2SO_4 solution and the further strengthening of the structure by formation of $CaSO_4$. Photo 1a gives an example of such a ring.

4.3 Cyclone and Grate Preheater Deposits

Characteristics:

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These deposits form on the roofs, walls, outlet and riser pipes of cyclone preheaters, in the hot chamber of grate preheaters, and vary considerably in appearance and homogeneity. In general, they have a light colour varying from cream to brown to pink, indicating that the component particles had not been heated higher than 1200°C. In some cases, darker zones of harder burnt material can be observed. Depending on their place of deposition, they range from a dense, compact, definitely layered structure, hard to break to a porous (30%) material with only moderate strength with less obvious layering. The former type is typical of cyclone cones and discharge pipes while the latter is to be found in the transition and swirl chambers. Soft deposits can, however, also be found in the cyclones.

From a chemical viewpoint, this deposit type usually is characterized by a concentration of the volatile elements in the following range:

K_2O = 1 - 30%
SO_3 = 1 - 35%
Cl = 1 - 25%
Na_2O = 0 - 2%

In some cases, therefore, deposits can occur with no appreciable increase in concentration. Typical analyses found for deposits are given in Table 3 and Appendix I.

The mineralogical composition of preheater deposits differs as would be expected from that of the raw meal in that the clays are essentially decomposed, and a reaction to form intermediate minerals has taken place. Minerals containing only the volatile elements can also be found.

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Amongst the minerals found in preheater deposits are the following:

raw meal:	
calcite	$CaCO_3$
quartz	SiO_2
normal intermediate phases:	
free lime	CaOf
periclase	MgO
mayenite	$C_{12}A_7$
belite	C_2S
typical deposit phases with low melting point:	
sylvite	KCl
halite	NaCl
langheinite	$2CaSO_4 . K_2SO_4$
arcanite	K_2SO_4
typical deposit phases without melt involvement:	
carbonate spurrite	$2C_2S . CaCO_3$
sulfate spurrite	$2C_2S . CaSO_4$
anhydrite	$CaS0_4$

Formation mechanism:

The binding substance in this deposit type is the low melting point Na_2O, K_2O, SO_3, Cl based compounds. These are molten in the kiln gas and are deposited on the cyclone walls and pipes, or first on dust particles which then themselves are deposited out of the gas stream in these areas. Cooling on contact or with increasing thickness results in an appreciable strengthening of the originally sticky deposit. Because of the extensive duration of stay in the kiln system, a reaction takes place with gaseous CO_2 and SO_3, resulting in the formation of lath shaped spurrite and sulfate spurrite which additionally strengthen the texture. Typical textures for unsintered, pre-heater and kiln inlet deposits can be seen in photos Ib - Id.

4.4 Meal Ring (Calcining Ring) in Long Kilns

Characteristics and formation:

The meal rings, often called "calcining rings" in long kilns, are in their properties and mechanism of formation very similar to those of pre-heater deposits in heat exchanger kilns. This is perhaps not surprising in that both build-ups occur in the same temperature zone. Meal rings are mostly less troublesome than pre-heater deposits because often, due to their relatively poor strength, thermal fluctuations, kiln deformation and the action of the material stream, they fall off periodically under their own weight. A typical example of a calcining ring is plant J in Table 4.

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4.5 Middle Rings in Large Pre-heater Kilns

Characteristics:

Unlike meal rings, middle "rings" are dense (fine grained) of low porosity, very hard and seldom fall off during operations. Although termed as a ring they are rather more elongated, like a band, being often some 15 - 20 m long extending from 7 to 11 diameters from the outlet, e.g. 35 - 55 m for a 5 m ∅ kiln. Unlike previous types, this deposit is clinker like in colour indicating it being composed of well burnt kiln charge. Perpendicular to the direction of deposition, the fine layered structure can be seen showing the curvature of the kiln shell.

The chemical composition of middle rings is very similar to that of clinker. This is surprising because considering the long duration of the stay in the kiln, no increase in concentration of the alkalis or SO_3 takes place, and often the ring shows lower volatile element values than for clinker. Typical analyses of a middle ring are given in Table 5.

The minerals found in middle rings are the clinker minerals elite, belite, aluminate, ferrite and free CaO, the elite having often decomposed into microscopically mixed belite and free CaO, resulting from the temperature at the site of the ring being under the lower stability temperature of elite (i.e. 1260°C).

Formation Mechanism:

The mechanism of bonding is the freezing of the clinker alumino-ferrite melt. Due to a long cool flame, the clinker has a tendency to be fine, and the smallest clinker particles of 150 - 450 um are carried back by the flame and deposited onto the kiln wall in a zone where temperatures of below 1250°C exist. The particles immediately freeze in place, and because the kiln charge is still fine, it does not possess sufficient abrasive action to remove the growing ring. The typical compact structure of a middle ring can be seen in photo 1e.

4.6 Sinter Rings (excluding coal-ash rings)

Characteristics:

These occur at the beginning of the sinter zone some 4 - 5 D from the kiln outlet. They are greyish-black in colour, strong and are (usually) agglomerations of small clinker pellets and clinker dust. No layer structure is obvious because of the presence of large pores and voids.

In general, the chemical composition is that of the clinker with no appreciable concentration of volatile elements.

From a mineralogical viewpoint, the normal clinker minerals elite, belite, aluminate, ferrite and free CaO are observed, with reactions to form belite and CaO_{free}, spurrite and belite being found with increasing depth in the ring, i.e. decreasing temperature, similar to the case of middle rings.

Bonding mechanism:

The bonding is created by the freezing of the alumino ferrite clinker liquid in the case of pure sinter rings. This phenomenon occurs especially at the beginning of the sinter zone, where the liquid phase is just starting to form (approx. 1280°C). Due to the rotation of the kiln, the charge in this zone freezes with each kiln revolution: a new wet layer sticks on, and with time a thick deposit builds up consisting of particles of less than 1mm.

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4.7 Coal Ash Sinter Rings

Characteristics:

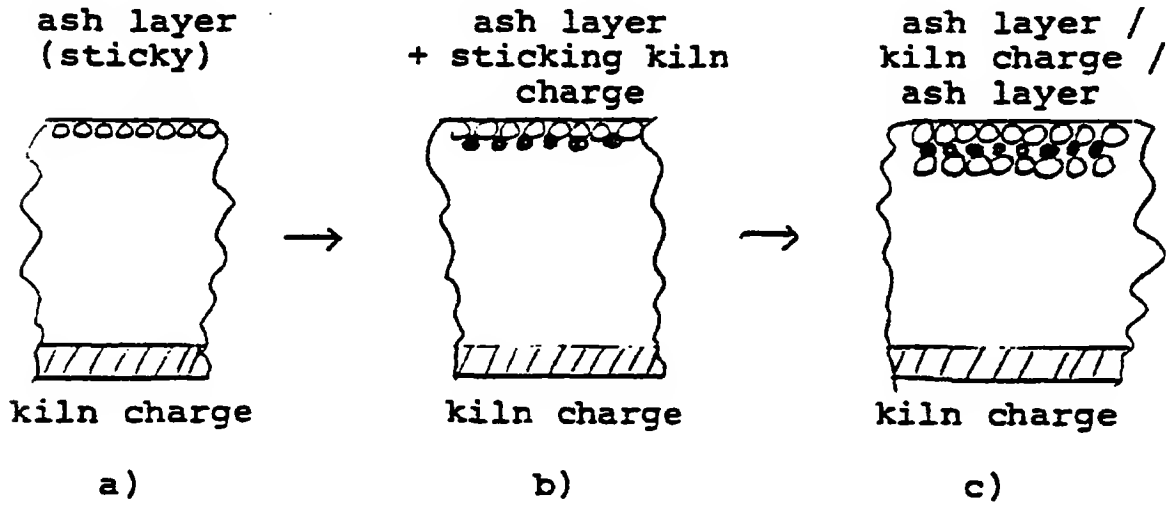
In kilns fired with a high ash content coal, sinter/coal ash rings can form at 7 - 8.5 D from the kiln outlet. They are dense, often layered and sometimes glassy in appearance and built up from particles some 150 250 um in size. They are rather less dense and have larger pores and voids than middle rings. Photo 1f gives an example of the microstructure of such material, showing the coal ash layers.

From the viewpoint of their chemical and mineralogical composition they are essentially similar to clinker, exhibiting the minerals elite, belite, aluminate, ferrite and free CaO. With decreasing temperature (increasing ring depth) reactions to form spurrite and calcite take place, and also the transformation of alite ⇒ belite + CaOf and β belite ⇒ α belite. Details of the chemistry and mineralogy are given in Table 6 . No enrichment of the volatile phases can be observed. Because of the enrichment in coal ash, the belite content is higher than that of the clinker, and tends to be found in layers.

Formation mechanism:

The bonding medium here is the sticky molten coal ash particles and perhaps to a slight extent, the alumino ferrite clinker liquid phase occurring by a mechanism such as in Fig. 2 showing the typical build up during kiln rotation.

Fig. 2: Mechanism for ring formation



The molten coal ash droplets adhere to the exposed kiln lining at a point and temperature at which they are still partially fluid and sticky. When this sticky layer passes under the kiln charge on each rotation, it is assumed that a single layer of the still very fine kiln charge adheres to it. Because of the presence of fine crystalline elite and the overall occurrence of liquid phase, it must be assumed that the material temperature at the position of the ring lies above 1260°C.

The elite crystals are very small and certainly much smaller than those of the clinker. Because of this, it can definitely be said that the ring is not formed from clinker dust blown back down the kiln.

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4.8 Clinker Rings / Cooler Inlet Deposit (snowman)

Such rings and deposits are formed from normal size clinker granules and have a high porosity containing many voids. They are usually not troublesome to kiln operations as they can easily be removed. Their composition and mineralogy is identical to clinker, but in some cases, rings of up to 3.5% K_2O and 3.0% SO_3 have been observed.

The mechanism of bonding is the freezing of the clinker liquid phase as the clinker passes through the cooling zone (ring) or on falling down the chute into a grate cooler, grate kilns being usually operated so as to have no cooling zone within the kiln itself.

4.9 Kiln Charge Balls

Kiln balls occur in cases where a tendency to meal or sinter ring already exists and can be up to 1m in diameter. The chemical composition is, thus, an important factor. They are usually found upstream of meal or sinter rings. They are usually made up of already calcined material and can have a porosity of up to 55%, consisting of many fine pores. Often they consist of a hard dense porous core, surrounded by the majority of porous material. The core usually is a piece of coating from say the lower heat exchangers or the transition chamber, and often has a come position different from the kiln charge in the area of formation. Differences in composition can be seen in Table 7.

The mechanism of meal ball formation can be due to either, or a combination of the following factors:

- stripping and subsequent "balling" of old, excess coating
- agglomeration enhanced by available clinker and/or salt melt
- ring section acts as a dam, retaining "pieces" of material for long periods. Radial growth of the pieces occurs by compaction and adherence of fresh surface due to continual rolling action of the pieces/balls over the charge.

In most cases, no liquid phase participation in sufficient quantities is possible so that the balls behave like a snowball and by their own pressure material sticks to the surface. This mechanism is similar to that of deposition on the exhaust fan blades.

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5. METHODS OF REMOVAL/ELIMINATION

An important prerequisite for minimizing the tendency to form objectionable coatings and rings, is stable kiln operation. This applies to the composition, fineness and feed rate of the raw material and fuel, and burning zone heat control.

The tendency to form coatings in the kiln is reduced by lowering the dust load of the kiln gas.

Objectionable coatings and rings which are formed as a consequence of high concentrations of various circulating elements can be obviated by appropriate reduction of the cycles in question.

This can be achieved by:

- Employing different raw materials and/or fuel with lower concentration of the offending element. This is generally not practicable.
- Control of the raw meal milling so as to reduce the concentration of the very fine particles of sizes under 20 µm.
- Intervention into cyclic process by either discarding dust in which the circulating elements have become concentrated, or by means of a bypass installation which extracts a portion of the kiln gas.

The penetration of false air into the preheater and kiln inlet chambers should be avoided, as such cold areas will act as sites for preferential build-up.

In order to reduce the tendency to form sinter rings, it is in the first place necessary to reduce the proportion of fusible matter in the clinker, i.e. the lime standard and silica modulus should be increased. "Coating-inactive" bricks have also proved successful in certain cases, in reducing the tendency toward sinter ring formation.

In coal-fired kilns a coal with a normal ash content should be employed as coals having ash contents of 40% are characterized by a very high tendency to ring formation. No general approach can be given to the effectiveness of other measures, e.g. alteration to firing conditions, as these represent variables which are peculiar to the particular installation.

Clinker rings can be avoided by shifting the flame further back, thus increasing the clinker temperature at the kiln outlet. As a result of this, however, the "stickiness" range of the clinker is shifted towards the cooler inlet. Coatings can then be formed on the cooler inlet chute. This is particularly problematic with satellite coolers. In instances where this occurs with grate coolers, these coatings can be eliminated by the use of water cooled plates on the inlet chute.

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Table 1 **Time and Location of Rings and Coating (according to Opitz, 1974)**

Table 1: T E AND LOCATION OF RINGS AND COATIN (according to Opitz, 1974)

type	kiln type(s)	location	temperature (°C) gas	charge
coating	all	exhaust fan blades	128/180	-
slurry ring	long wet	drying zone	150/300	<100
coating	dry (preheater)	stage 3 & 4	700/1100	700/800
coating	Lepol grate	walls/roof of hot chamber	1000/1100	--
coating	long dry	inlet chamber	1000/1100	--
meal ring middle ring	preheater long dry wet	calcining zone " " " "	1100/1400 <1400 <1400	800/1200 <1200 <1200
kiln charge ball	all	calcining zone	<1400	<1200
sinter ring	all	beginning of sinter zone	1400/1600	1250/1350
sinter zone coating	all	sinter zone	1600/1800	1350/1450
clinker ring	all	end of sinter zone	800/1600	1200/1400
coating	grate cooler entrance		600/800	1200

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Table 2 General Characteristics of Various Ring Types

Table 2: GENERAL MATERIAL CHARACTERISTICS OF VARIOUS RING TYPES

Type	Kiln	Location	Character Base Material			Enriched in				Texture	
			Raw Meal	Int. Mod.	Clinker	Na,K	SO_3	Cl	CO_2	Packing	"Particle" size
Coating	all	Exhaust fan Blades	+ (clay)			+	+			dense	fine
Slurry ring	WL	Drying zone	+ (dust)			+	+			dense	fine
Coating	DS	Stage 3+4	+	(+)		+	+	+		porous	fine
Coating	DG/P	Walls/roof of hot chamber	+	(+)		+	+	+		porous	fine
Coating	DL	Inlet chamber	+	(+)		+	+	+		porous	fine
Meal ring Middle ring	WL DS DL	Calcining zone " " " "	+	+	 + +					dense " "	fine " "
Kiln charge ball	ALL	Calcining zone		+*	+					porous	fine
Sinter ring	ALL	Beginning of sinter zone			+					porous	large
Sinter zone coating	ALL	Sinter zone			+					porous	large
Clinker ring	ALL	End of sinter zone			+					porous	large
Coating		Grate cooler			+					porous	large

* coating center

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Table 3 Properties of Build-ups Examined

Plant	Location	Volatile Elements				Alkali Sulphur Ratio	Moduli			General appearance			Mineralogy	Texture		
		K_2O	Na_2O	Cl	SO_3		SR	AM	LSF	Strength	Colour	Colour of streaks		Layer sampled	Pore size (µm)	Bld.Units (µm)
A	1st cycl.	7.6	0.18	1.6	29.7	0.16	1.76	2.77	62	strong	yellow	red/brown	-		1 - 10	>1 - 10
A	2nd cycl.	11.4	0.16	1.2	24.0	0.36	2.05	2.77	86	strong	yellow	red/brown	-	dense porous	>1 - 5 1 - 10	2 - 3 2 - 10
A	3rd cycl.	4.8	0.09	1.8	2.56	0.85	2.25	3.44	124	weak	yellow	none	-		-	-
B	" "	1.3	0.65	1.09	36.0	0.02	2.8	2.5	149	strong	yellow/ brown	dark brown	-	dense porous	<1 - 3 2 - 5	1 - 5 1 - 5
C	4th cycl. (Pipe)	0.8	0.11	1.2	0.61	0.87	2.4	1.69	339	strong	yellow	none	-		-	-
C	" (Cone)	0.19	0.09	0.38	0.40	0.38	3.33	1.70	180	strong	red/brown	none	-		10 - 20	10 - 20
D	"	3.6	0.78	3.1	7.9	0.07	1.46	1.45	198	-	-	-	-	dense porous	1 - 5 10 - 20	1 - 5 25 - 50
E	"	28.0	1.0	19.3	0.02	(164.9)	3.01	2.28	103	medium	brown	white/green	-		pores filled	KCl-200 charge 1-5
B	"	23.8	1.8	19.3	0.33	2.31	3.2	2.8	104	weak	brown	none	-		pores filled	KCl cont. charge 2-3
F	"	31.5	0.33	23.4	0.16	4.85	2.62	2.36	89	strong	yellow/ brown	none			pores filled	KCl coat. charge 0-5
D	Riser Pipe	3.6	0.98	2.7	15.9	0.08	1.61	1.59	157	-	-	-	major: calcite, sulphate spurrite, KCl minor: spurrite, CaOf		-	-
G	"	4.8	0.85	5.0	20.2	-0.03	2.5	1.3	196	strong	yellow/ brown	none	-	porous	10 - 15	5 - 10
A	Kiln Inlet	8.4	0.19	2.1	7.3	0.69	3.2	2.13	114	weak	yellow/ brown	none	-		-	-
B	"	7.6	1.1	0.27	4.6	1.65	2.09	5.89	52	weak	khaki	none	-	dense porous	1 - 3 20 - 30	20 - 30
H	"	6.9	0.34	0.09	27.6	0.22	1.48	0.68	75	strong	red/brown	black	-	dense porous	>1 100	lu,KCl cont. 20 - 30

Table 4 Properties of Rings Examined

Plant	Ring Type	Location	Volatile Elements				Alkali Sulphur Ratio	Moduli			General appearance		Mineralogy	Texture	
			K_2O	Na_2O	Cl	SO_3		SR	AM	LSF	Strength	Colour		Pore Size (µm)	Bld.Units (µm)
I (wet)	Mud Ball	110 m	4.1	0.40	0.23	8.5	0.44	2.19	1.48	110	strong	yellow/ brown	-	2 - 3	2 - 5
J (wet)	Meal	100 m	2.18	0.16	0.01	14.2	0.14	2.36	1.46	78	strong	green/ grey	Major - Sulphate Spurrite	100	50 - 100
													Minor - C_2S, C_4AF, Spurrite, various sulphates	100	50 - 100
K	Middle	45 m	0.16	0.06	0.01	0.13	1.56	2.2	1.5	92	strong	yellow/ brown	C_3S, C_2S, C_3A, C_4AF, CaOf	150 - 250	150 - 450
F	Middle	37 m	0.72	0.07	0.05	0.77	0.84	2.73	2.32	88	strong	yellow/ brown	-	50	50 - 100
L	Ash/ Sinter	35 m	1.07	0.24	-	1.07	1.14	2.52	1.86	87	-	-	C_3S, C_2S, C_3A, C_4AF, CaOf	200	100 - 200
N	Ash/ Sinter	32.5 m	0.34	0.03	-	0.27	1.21	2.4	1.3	79	strong	grey/ black	C_3S, C_2S, C_3A, CaOf, spurrite	100 - 150	200
R	Ash/ Sinter	18.5 m	0.95	0.28	0.28	1.56	0.55	2.88	1.55	88	strong	brown/ grey	C_3S, C_2S, C_3A, C_4AF, CaOf	200 - 500	150 - 300

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Table 5 : Typical analyses of a middle ring at various depths

sample	ring internal	ring middle	ring external	clinker
Loss on ign.	0.47	1.82	0.94	0.37
SiO_2	23.0	17.8	21.9	21.9
Al_2O_3	4.9	5.1	5.5	5.6
Fe_2O_3	3.0	2.9	3.1	3.2
CaO	66.4	70.7	66.8	66.1
MgO	1.1	1.0	0.95	0.95
SO_3	0.13	0.07	0.27	0.48
K_2O	0.16	0.09	0.30	0.35
Na_2O	0.06	0.06	0.12	0.14
TiO_2	0.25	0.23	0.25	0.25
Mn_2O_3	0.02	0.02	0.02	0.02
P_2O_5	0.24	0.21	0.20	0.20
Cl-	0.01	0.01	0.01	0.01
Total	99.74	100.36	100.01	99.57
LS	92	122	96	93
SR	2.91	2.23	2.55	2.49
AR	1.63	1.76	1.77	1.75

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Table 6: Chemistry and Mineralogy of a (coal ash) ring

kiln point	29.5	32.5	33.5
depth in ring	under hot surface	middle	near shell
chemical analysis:			
SiO_2	24.65	24.61	21.77
Al_2O_3	5.68	5.70	5.48
Fe_2O_3	4.23	4.29	3.53
Cao	61.27	61.92	65.31
MgO	0.49	0.79	0.64
K_2O	0.58	0.34	0.17
Na_2O	0.36	0.03	0.20
SO_3	0.15	0.27	0.34
Loss on ign.	2.61	1.63	2.48
Total	100.0	99.68	100.0
LS	78.1	79.0	93.7
SR	2.49	2.46	2.42
AR	1.34	1.33	1.55
free lime	1.0	4.0	2.7
minerals det.: (wt. %)			
alite	~ 30	~ 15	~ 40
α and β belite	~ 35 - 40	~ 55 - 60	~ 30
liquid phase (aluminate + ferrite)	~ 15	~ 15	~ 15
carbonate spurrite+CaOf	~ 15	10	~ 15
λ belite	< 5	< 5	
texture:			
average pore size (estimated) µm	400 - 800	400 - 800	100 - 300

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Table 7. Chemical comopsition of core and rim of meal ball and the kiln charge composition at the corresponding zone

	kiln ball rim	kiln charge core	calcining zone
Loss on ign.	3.9	4.3	1.9
SiO_2	20.9	17.2	21.2
Al_2O_3	5.6	8.0	5.7
Fe_2O_3	2.8	4.4	2.7
Cao	63.0	62.5	62.1
SO_3	1.9	1.7	2.2
K_2O	2.1	1.5	2.3
Cl-	0.1	0.1	0.1
LSF	94	104	91
SR	2.4	1.4	2.5

Appendix I: Chemical analysis of preheater cyclone deposits

MA-Mat. Nr.	54'138	54'140	54'142	54'119	47'504	47'505	50'397	54'272	54'120	52'838
Plant	A	A	A	B	C	C	D	E	B	F
Location	1.Cycl.	2.Cycl.	3. Cyclone		4. Cyclone					
Loss on ign.	6.9	6.9	23.8	4.3	35.7	37.3	15.0	28.9	21.7	11.66
SiO_2	14.6	12.3	13.4	9.0	5.0	9.0	7.9	9.1	10.8	9.7
Al_2O_3	6.1	4.2	2.7	2.3	1.3	1.7	3.2	2.1	2.5	2.6
Fe_2O_3	2.2	1.8	1.2	0.91	0.77	1.0	2.2	0.92	0.88	1.1
CaO	30.5	34.9	51.5	42.5	54.4	50.1	54.0	29.5	35.0	27.5
MgO	1.5	1.4	1.3	0.67	1.3	0.24	1.1	0.59	0.49	0.87
SO_3	29.7	24.0	2.56	36.0	0.61	0.40	7.9	0.02	0.33	0.16
K_2O	7.6	11.4	4.8	1.3	0.80	0.19	3.6	28.0	23.8	31.5
Na_2O	0.18	0.16	0.09	0.65	0.11	0.09	0.78	1.0	1.8	0.33
TiO_2	0.22	0.17	0.11	0.11	0.18	0.16	0.10	0.06	0.11	-
Mn_2O_3	0.02	0.01	0.01	0.03	0.05	0.04	0.09	0.03	0.03	0.02
P_2O_5	0.07	0.07	0.05	0.10	-	-	0.11	0.04	0.09	-
Cl^-	1.6	1.2	1.8	1.09	1.2	0.38	3.1	21.5	19.3	23.4
Total	101.19	98.51	103.32	98.96	101.42	101.2	99.08	(121.76)	(116.83)	(108.84)
SR	1.76	2.05	3.44	2.80	2.42	3.33	1.46	3.01	3.20	2.62
AR	2.77	2.33	2.25	2.53	1.69	1.70	1.45	2.28	2.84	2.36
LSF	62	86	124	149	339	180	198	103	104	89

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Appendix II: Chemical analysis of riser pipe kiln inlet deposits

APPENDIX II: CHEMICAL ANALYSIS OF RISER PIPE AND KILN INLET DEPOSITS

MA-Mat. Nr.	50'393	53'550	54'155	54'118	53'956
Plant	D	G	A	B	H
Location	Riser Pipe		Kiln Inlet (DS)		Kiln Inlet (DG)
Loss on ign.	11.8	10.1	7.1	2.6	0.57
SiO_2	9.2	7.4	15.6	25.9	12.4
Al_2O_3	3.5	1.7	3.4	10.6	3.4
Fe_2O_3	2.2	1.3	1.6	1.8	5.0
CaO	49.3	46.1	55.4	45.1	39.8
MgO	1.2	2.0	1.5	0.83	2.3
SO_3	15.9	20.2	7.3	4.6	27.6
K_2O	3.6	4.8	8.4	7.6	6.9
Na_2O	0.98	0.85	0.19	1.1	0.34
TiO_2	0.09	0.15	0.15	0.41	0.19
Mn_2O_3	0.07	0.01	0.03	0.04	0.16
P_2O_5	0.11	0.09	0.05	0.38	0.08
Cl^-	2.7	5.0	2.1	0.27	0.09
Total	100.65	99.7	102.82	101.23	98.83
SR	1.61	2.47	3.12	2.09	1.48
AR	1.59	1.31	2.13	5.89	0.68
LSF	157	196	114	52	95

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Appendix III Chemical analysis of rings from rotary section of Kiln

MA-Mat. Nr.	53'952	52'040	52'037	52'178	49'643	45'061	50'524
Plant	I	K	F	M	L	N	J
Location	100-110m	45m	37m	32.5m	35m	18.3m	100m
Loss on ign.	10.7	0.47	2.2	1.63	0.12	8.11	1.5
SiO_2	14.7	23.0	22.7	24.6	22.84	21.3	20.3
Al_2O_3	4.0	4.9	5.8	5.7	5.9	4.5	5.1
Fe_2O_3	2.7	3.0	2.5	4.3	3.2	2.9	3.5
CaO	52.3	66.4	63.1	61.9	63.9	58.8	50.8
MgO	0.79	1.1	2.1	0.79	0.92	1.5	1.13
SO_3	8.5	0.13	0.77	0.27	1.07	1.56	14.2
K_2O	4.1	0.16	0.72	0.34	1.07	0.95	2.18
Na_2O	0.40	0.06	0.07	0.03	0.24	0.28	0.16
TiO_2	0.18	0.25	0.31	-	0.36	0.28	0.22
Mn_2O_3	0.06	0.02	0.04	-	0.15	0.02	0.04
P_2O_5	0.23	0.24	0.08	-	0.24	0.08	0.24
Cl^-	0.23	0.01	0.05	-		0.28	0.01
Total	98.89	99.74	100.44	99.68	100.25	100.56	99.38
CaOf		7.4			7.1		0.62
SR	2.19	2.91	2.73	2.46	2.51	2.88	2.36
AR	1.48	1.63	2.32	1.33	1.84	1.55	1.46
LSF	110	92	88	79	88	88	78

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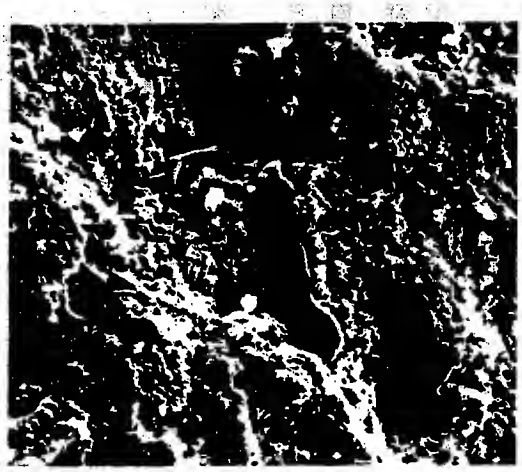
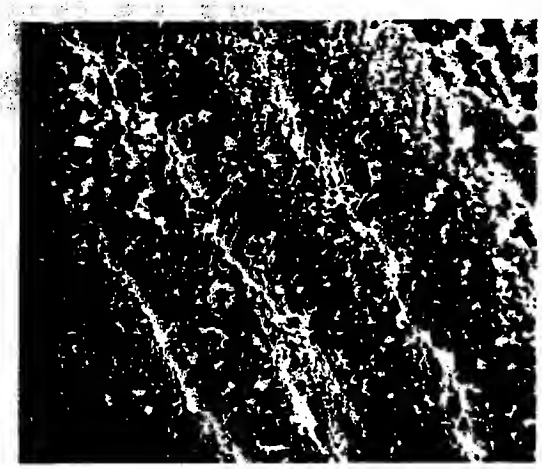
Phototable 1 S.E.M Micrographs of Deposits



1 a) 10µ REM 84/108

Mud ball in chains



1 b) 20µ REM 84/506

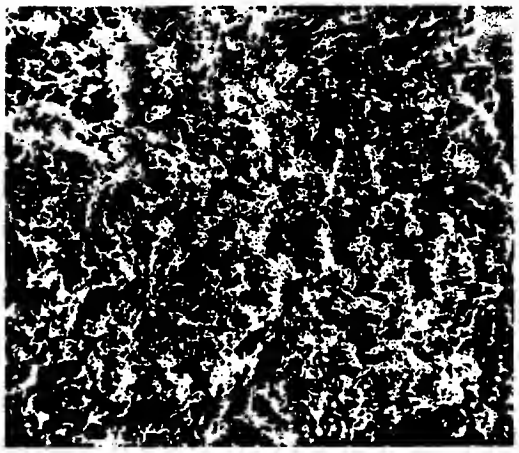
Compact cyclone 2 deposit

1 c) 10µ REM 84/545

KCl crystals

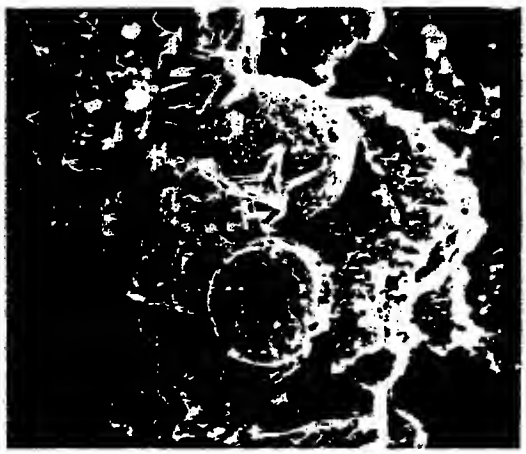
1 d) 2µ REM 84/69

Spurrite crystals



1 e) 200µ REM 84/79

Texture of middle ring



1 f) 10µ REM 84/72

Fly ash particles

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Chapter 5

Fuels for Cement Kilns

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Fuels for Cement Kilns

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1. INTRODUCTION

By common definition a fuel is any material that liberates heat when it reacts with oxygen. In practice, however, only those materials which ignite at moderate temperatures burn with comparative rapidity are obtainable in quantity at reasonable prices are regarded as fuels.

According to their physical state fuels are divided into 3 main groups, i.e. solid, liquid and gaseous. All 3 are employed in the cement industry. Originally coal was used almost exclusively. Around 1960 the conversion to oil and gas firing began and proceeded quickly because these two fuels had become sensibly cheaper than coal, and furthermore they were much easier to handle. However, after 1973 oil (and gas) prices rose again steeply and soon surpassed coal by far. Plants in countries without own deposits of oil or gas began to return to coal, and by now most of them are back again to nearly 100 % of coal firing. Fig. 1 shows the consumption of different fuels in the cement industry of Switzerland, a country which has neither coal nor oil of her own. For countries which dispose of oil (or gas) deposits the return to coal is not so urgent, of course, but on the whole coal is gaining ground world-wide as can be seen e.g. from figures 2 and 3 which show the development in the "Holderbank" Group since 1974.

Fig. 1 Consumption of Coal and Fuel Oil in the Swiss Cement Industry from 1959-1985

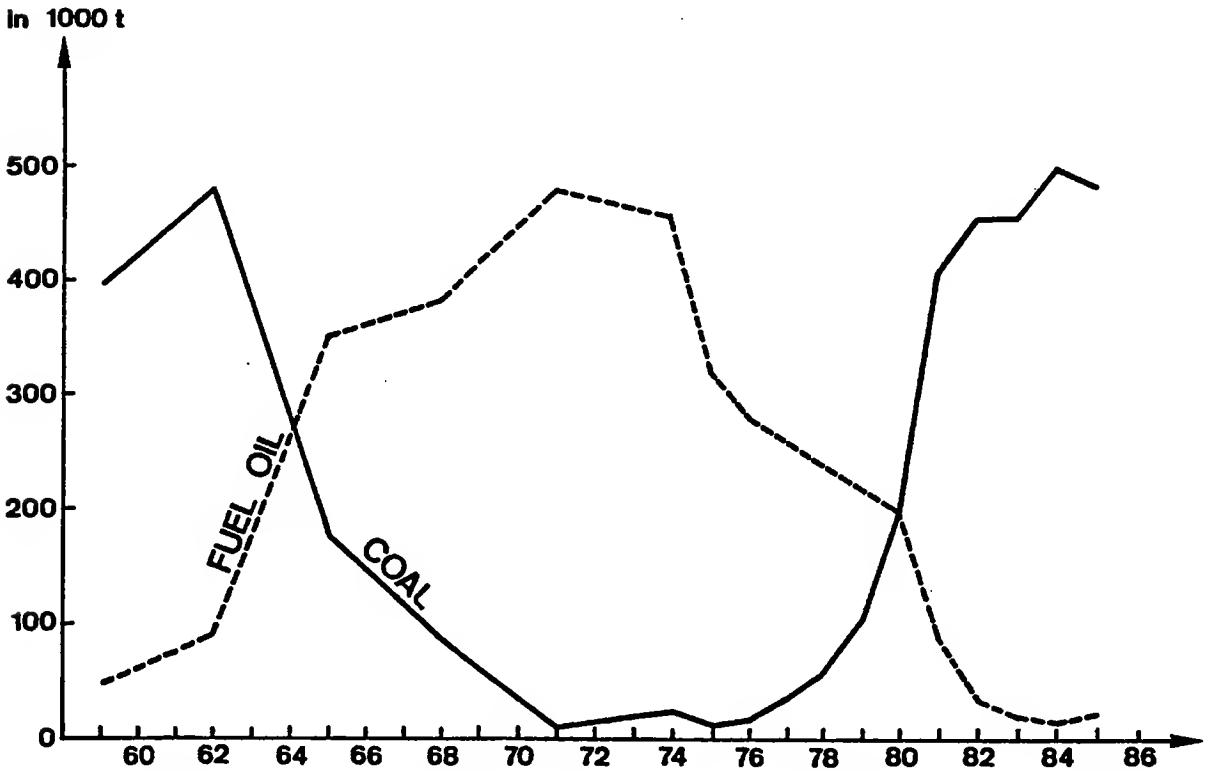

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Fig. 2 Percentage of Clinker Produced with Different Types of Fuel

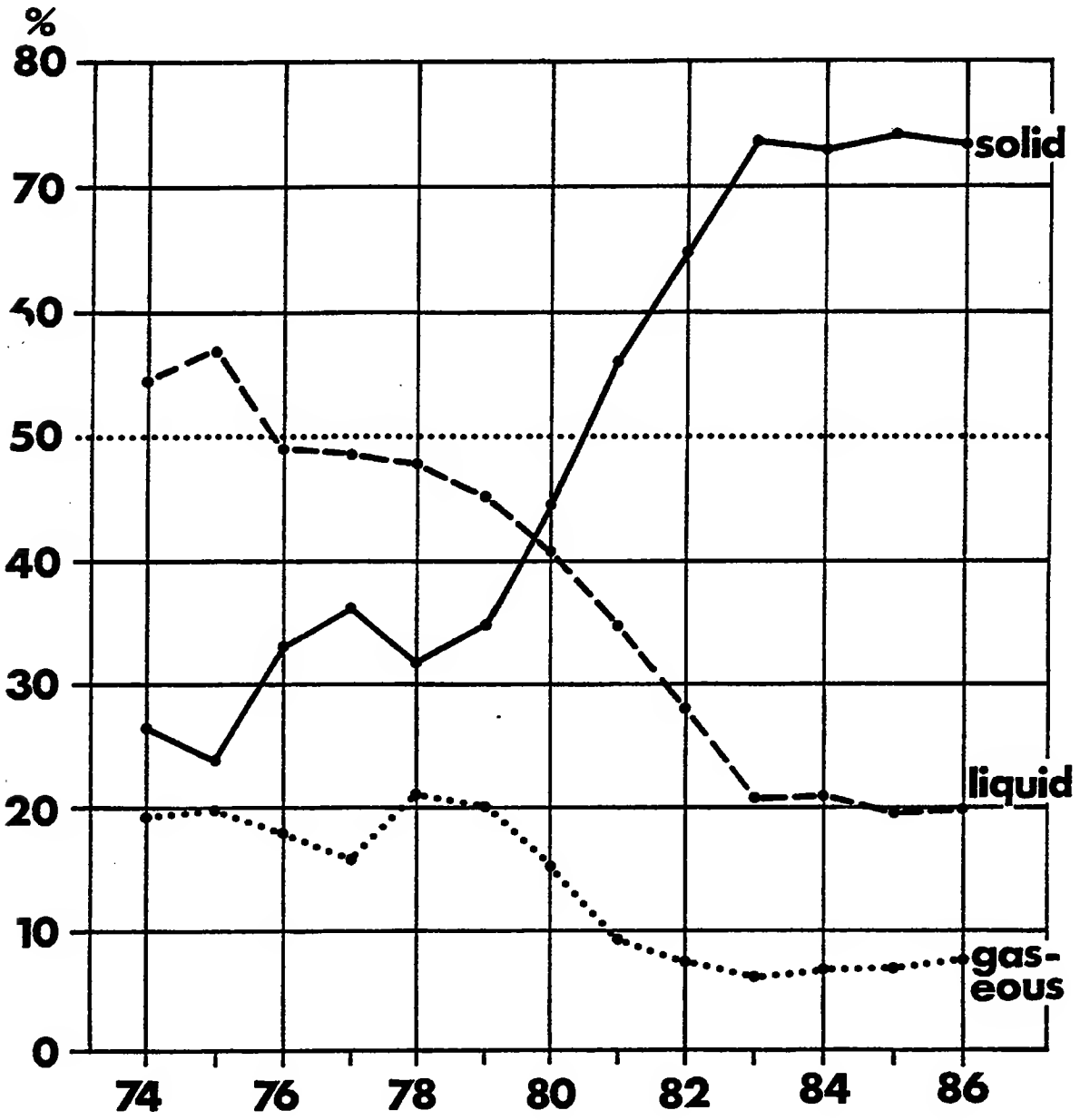
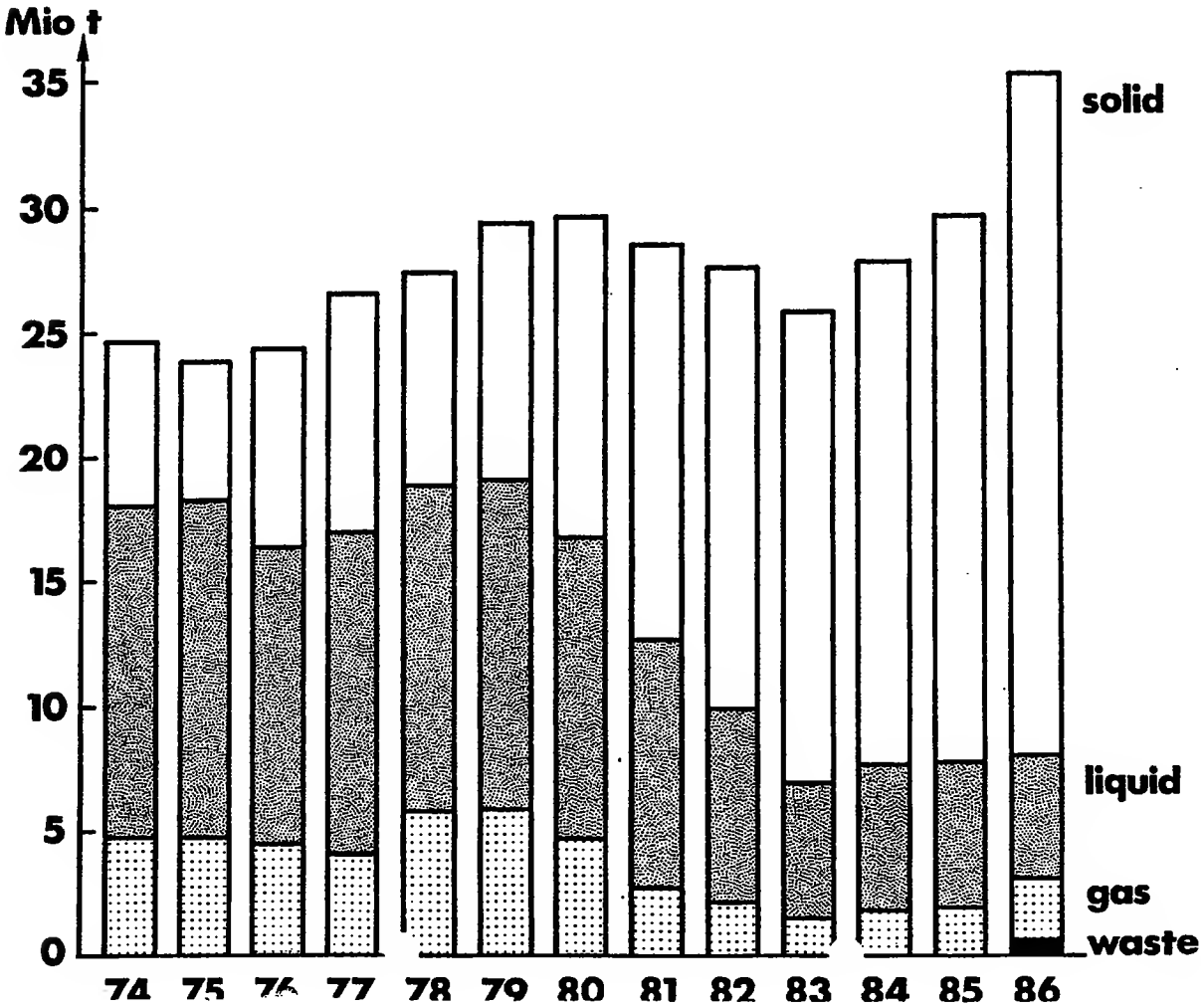


Fig. 3 Quantity of Clinker Produced with Different Types of Fuel



2. GENERALITIES

2.1 CHEMICAL COMPOSITIONS

All ordinary fuels consist mainly of carbon and hydrogen accompanied by minor amounts of nitrogen, oxygen, sulfur and sometimes inorganic minerals. For example, table 1 shows the analyses of the mostly used fuels on a dry, mineral-free basis and without inert components of natural gas like N_2 and CO_2.

Table 1: Ultimate analyses of fuels (weight percent)

	bituminous coal	fuel oil	natural gas
C	88 - 90	83 - 87	75 - 77
H	4 - 5	11 - 13	23 - 25
O	3 - 4	0.1-0.4	traces
S	1 - 2	1 - 3	traces

Appreciable amounts of minerals which form ashes after combustion occur only in solid fuels and shall be discussed in chapter 3.5.

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Sulfur, although a minor component of fuels is of particular importance because of its effects on clinker quality and, above all, the environment. Furthermore, owing to its volatility, it can accumulate in certain parts of the kiln system and form build-ups and rings. Therefore it is absolutely necessary to supervise closely the sulfur contents of both raw mix and fuel. Depending on the raw mix composition and on kiln conditions (temperature, atmosphere) the clinker can absorb considerable quantities of sulfur in the form of sulfates. When its capacity is exceeded the excess sulfur is emitted by the stack as SO_2 which can have detrimental effects on organisms and buildings. Therefore in most industrialised countries the authorities have restricted the emission of SO_2 and in some cases even demand monitoring of emissions.

2.2 HEAT OF COMBUSTION

The carbon and hydrogen contents of the fuels determine their most important property, namely their heat of combustion or calorific value. Chemically speaking combustion is nothing else but oxidation. The elements of the fuel, mainly carbon and hydrogen, combine with oxygen from the air and a certain amount of energy is set free as heat. The heat is measured in

- Joule (J) (International Standard Unit)
- calorie (car) (no longer valid but still widely used)
- British Thermal Unit (BTU) (United Kingdom, USA etc.)

The conversion factors are

- 1 cal = 4.187 J
- 1 BTU = 1055 J

For the requirements of the cement industry a larger unit is more suitable i.e. the Mega-Joule (MJ)

- 1 MJ = 106 J

The heat generated per unit weight of fuel is called specific "heat of combustion" or "calorific value". For example, 1 kg of carbon will combine with 2.67 kg of oxygen to form 3.67 kg of CO_2, releasing 33.9 MJ of heat:

$1kg\ C + 2.67kg\ O_2 \rightarrow 3.67kg\ CO_2 + 33.9\ MJ$

Accordingly the calorific value of carbon is 33.9 MJ/kg.

The calorific value is normally calculated for complete combustion to the highest possible state of oxidation. If not sufficient oxygen is present toxic CO will be formed with much less heat development:

$1kg\ C + 1.33\ kg\ O_2 \rightarrow 2.33kg\ CO + 10.0\ MJ$

This means that the emission of carbon monoxide is not only a pollution of the atmosphere but also a waste of energy.

For hydrogen one obtains for the formation of water vapour

$1kg\ H_2 + 8kg\ O_2 \rightarrow 9kg\ H_2O + 119.9\ MJ$

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This is called the 'net calorific value'. By condensation of the vapour another 20.3 MJ would be released bringing the total heat development of the reaction to 140.2 MJ. This is the 'gross calorific value'. Of course, for cement kilns only the net values are important because the flue gases leave far above 100°C and therefore the heat of condensation of the water vapour cannot be utilised. Nevertheless in some countries, including the US, the gross calorific value is generally used.

For conversion from gross (G) to net (N) heats of combustion (MJ/Kg) the following formula is valid:

$N = G - (W_B + W_M) \cdot H_C$

where W_M stands for the moisture content of the fuel, W_B for the water generated by combustion (all in kg), and H_C for the heat of condensation of water vapour which is 2.26 MJ/Kg.

The contribution of sulfur to the calorific value of fuels is very small:

$1kg\ S + 1\ kg\ O_2 \rightarrow 2\ kg\ SO_2 + 9.3\ MJ$

Besides MJ/kg other units for calorific values are still in use such as

- J/g (or kJ/kg)
- cal/g (or kcal/kg)
- BTU/lb (British Thermal Units per pound)

For gaseous fuels the calorific values usually refer to units of volume under standard conditions (0°C, 760 mm Hg) instead of mass (Nm^3 = cubic meter or cf = cubic foot. 1 Nm^3 = 35.31 cf). The conversion factors of these units are listed in table 2.

Table 2: Conversion factors for calorific values

a) Units for solid and liquid fuels			
	J/g	cal/g	BTU/lb
J/g	1	0.2388	0.4299
cal/g	4.187	1	1.800
BTU/lb	2.326	0.5555	1
b) Units for gaseous fuels			
	kJ/Nm^3	$kcal/Nm^3$	BTU/cf
kJ/Nm^3	1	0.2388	0.0268
$kcal/Nm^3$	4.187	1	0.1122
BTU/cf	37.25	8.896	1

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Typical calorific values of the main fuels employed in the cement industry are shown in table 3.

Table 3: Net calorific values of various fuels

a) solids + liquids	MJ/kg	cal/g	BTU/lb
bituminous coal (dry, mineral-free)	33 - 35	7800 - 8400	14100 - 15200
heavy fuel oil	40 - 41	9600 - 9900	17300 - 17800
b) gaseous	**MJ/Nm**	**kcal/Nm3**	**BTU/cf**
natural gas	34 - 38	8000 - 9000	900 - 1000

From the foregoing it can be inferred that the calorific value of a fuel may be calculated from its elementary composition. However, the energy to break up the chemical compounds of the fuel has to be deducted. In general, this is not known with sufficient accuracy. Therefore the direct determination of the heat of combustion by calorimetric methods is to be preferred. It is simple and quick and requires no special skill if a modern calorimeter is available. Fuel prices depending directly on calorific values frequent checks are to be strongly recommended. Owing to the tremendous heat requirements of a cement kiln even small variations of the calorific values may become very costly.

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3. COAL

3.1 FORMATION AND CLASSIFICATION OF COALS

Collier's Encyclopedia defines coal as a "combustible, sedimentary rock formed of vegetable matter by physical and chemical alteration through geologic processes". The first intermediate product of these processes is peat which is not yet considered as coal. Then follows lignite (or brown coal) several types of bituminous coal, and finally anthracite. During this transformation the carbon content and with it the calorific value increase continuously while the volatile matter (mainly hydrocarbons), decreases. Thus coals are classified according to their contents of volatile matter and/or their calorific value. For example, table 4 shows the ASTM classification (Standard Specification D 388 - 66).

Table 4 Classification of Coals by Rank

Class	Group	Fixed Carbon Limits, percent (Dry, Mineral-Matter-Free Basis)		Volatile Matter Limits, percent (Dry, Mineral-Matter-Free Basis)		Calorific Value Limits, Btu per pound (Moist,[a] Mineral-Matter-Free Basis)		Agglomerating Character
		Equal or Greater Than	Less Than	Greater Than	Equal or Less Than	Equal or Greater Than	Less Than	
I. Anthracitic	1. Meta-anthracite	98	...	...	2	...	...	nonagglomerating
	2. Anthracite	92	98	2	8	...	...	
	3. Semianthracite[c]	86	92	8	14	...	...	
II. Bituminous	1. Low volatile bituminous coal	78	86	14	22	...	...	Commonly agglomerating[d]
	2. Medium volatile bituminous coal	69	78	22	31	...	...	
	3. High volatile *A* bituminous coal	...	69	31	...	14 000[e]	...	
	4. High volatile *B* bituminous coal	...	...	...	...	13 000[e]	14 000	
	5. High volatile *C* bituminous coal	...	...	...	...	11 500	13 000	
						10 500	11 500	agglomerating
III. Subbituminous	1. Subbituminous *A* coal	...	...	...	...	10 500	11 500	nonagglomerating
	2. Subbituminous *B* coal	...	...	...	...	9 500	10 500	
	3. Subbituminous *C* coal	...	...	...	...	8 300	9 500	
IV. Lignitic	1. Lignite *A*	...	...	...	...	6 300	8 300	
	2. Lignite *B*	...	...	...	...	...	6 300	

[a] This classification does not include a few coals, principally nonbanded varieties, which have unusual physical and chemical properties and which come within the limits fixed carbon or calorific value of the high-volatile bituminous and subbituminous ranks. All of these coals either contain less than 48 percent dry, mineral-matter-free fixed carbon or have more than 15,500 moist, mineral-matter-free British thermal units per pound.
[b] Moist refers to coal containing its natural inherent moisture but not including visible water on the surface of the coal.
[c] If agglomerating, classify in low-volatile group of the bituminous class.
[d] Coals having 69 percent or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of calorific value.
[e] It is recognized that there may be nonagglomerating varieties in these groups of the bituminous class, and there are notable exceptions in high volatile C bituminous group.

3.2 ANALYSIS

Proximate Analysis

Proximate Analysis covers the determination of moisture, volatile matter, fixed carbon, and ash by determining the weight losses at certain temperatures. For example ASTM-Standard D 3172 prescribes

- for moisture 107°C in air
- for volatile matter 950°C with exclusion of air(covered crucibles)
- for fixed carbon 750°C in air The residue in ash.

Proximate analysis is simple and fairly quick. Together with the calorific value and sulfur content it is sufficient for all practical purposes.

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Ultimate Analysis

Ultimate Analysis determines the elementary composition. It requires special equipment and can only be carried out in laboratories with appropriately trained personnel.

Sulfur

Sulfur is analysed by ignition and subsequent determination of SO2 in the combustion gases by infrared absorption or colorimetry.

Chlorine

Chlorine can be determined in the liquid from the calorimetric bomb by titration with silver nitrate.

Ash composition

Ash composition which is important for correct calculation of the raw mix (see chapter 3.5) is analysed by the same methods as other raw materials.

3.3 VOLATILE MATTER

Not all types of coal are equally suitable for cement kilos. If the volatile contents are too low ignition of the coal particles is delayed and the flame becomes too long. On the other hand, high volatile contents increase the risk of self-ignition and explosions. For a long time about 20 % of volatile matter was considered an acceptable compromise, and preferably bituminous coal of that composition was used, or, if that was not available, a corresponding mixture of coals with higher and lower volatile contents. In the meantime engineers have learned to master the risk of self-ignition and explosions and brown coal containing 50 % of volatiles or even more can be employed.

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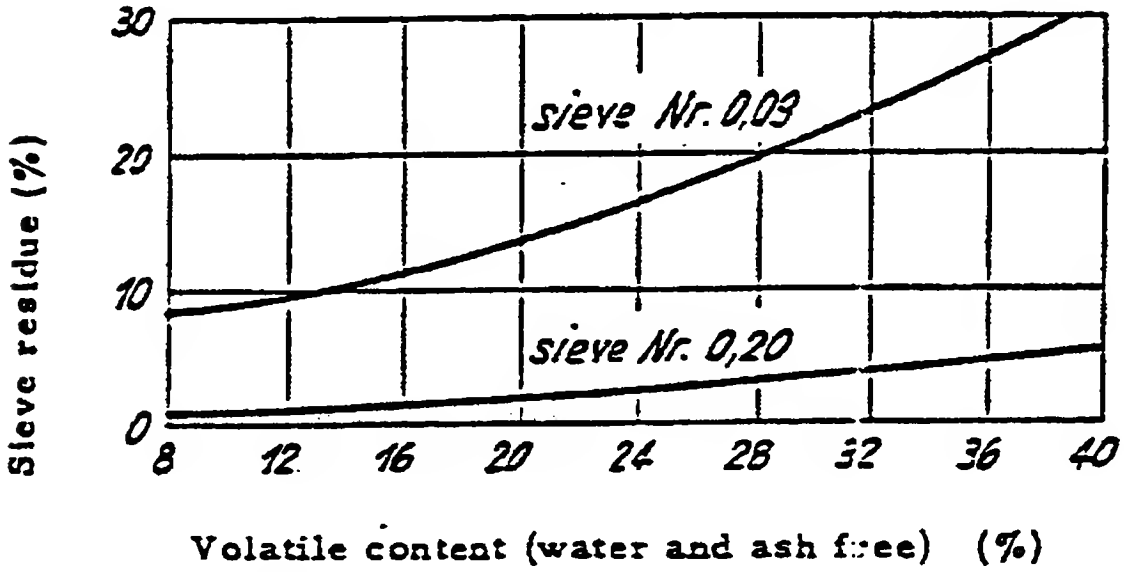
3.4 GRINDING AND DRYING

Besides a certain content of volatile matter the fineness of the coal dust is of prime importance for the generation of a good flame in the kiln. In fact, a lack of volatiles can to a certain extent be compensated for by finer grinding of the coal as can be seen on fig. 4:

Fig. 4 Required fineness of coal as a function of volatile content (Sieve numbers indicate mesh openings in mm)



Grinding is generally combined with drying of the coal. This drying should not go too far because completely dry coal could be difficult to ignite. In the absence of water the reaction of carbon with oxygen is very sluggish. 1 - 1.5 % of residual moisture in the coal will accelerate the reaction sufficiently. The exact drying temperature for a certain coal can be determined in the laboratory by establishing the desiccation curve, i.e. residual water as a function of the temperature.

3.5 ASH CONTENT

It has already been mentioned that coal contains mineral components which leave solid remainders after combustion, the ash. Bituminous coal which is generally used for clinker burning gives a siliceaous ash in the form of molten drops. These are carried away by the flame gases and finally settle on the clinker being formed. There is thus a higher SiO_2-concentration on the surface of the kiln charge. Depending on how far the ash drops are carried into the kiln they will be more or less assimilated with the clinker granules. Due to the deposition of ash drops rich in silica on the kiln charge there is a local lack of CaO. Here only belite can be formed and thus the deposition of ash leads to typical belite streaks in the clinker as shown in chapter 13.2.3.5, plate 19.

Because the coal ash is incorporated into the clinker it is absolutely necessary to take it into account when calculating the raw mix. The following example may serve to demonstrate this. Suppose raw materials and ash have the compositions listed in table 5 a) and the coal required for clinker burning leaves 2 % of the raw mix weight as ash. To produce a clinker with 55 % C3S the raw mix shown in the first column of table 5 b) would be required.

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If the coal ash is not taken into account the calculation for 55 % C_3S will give the raw mix of the second column. After incorporation of the coal ash the C_3S - content of the clinker would come down to only 39 %.

Table 5: Consequences of neglect of coal ash at raw mix calculations

a) Analyses of raw materials and ash (%)

	Limestone	Clay	Sand	Coal ash
L.o.i.	42.2	12.5	16.7	---
SiO_2	3.1	52.2	59.4	42.9
Al_2O_3	0-9	15.6	1.5	20.6
Fe_2O_3	0.6	13.2	3.0	22.4
CaO	52.6	2.4	17.2	8.2

b) Raw mix calculated for 55 % C_3S (%)	**with ash**	**without ash**
Limestone	79.2	77.9
Clay	15.0	17.8
Sand	5.8	4.3
Coal ash	2.0	

c) Clinker analyses (%) after absorption of coal ash		
SiO_2	22.2	22.8
Al_2O_3	5 4	6.0
Fe_2O_3	4 7	5.1
CaO	65.6	63.9
Tricalciumsilicate C_3S (%)	55.0	39.0

4. FUEL OIL

4.1 COMPOSITIONS AND CLASSIFICATION

Fuel is one of the products of the refining of petroleum. Petroleum or crude oil is a liquid found in subterranean deposits. Chemically it is a complex mixture of hydrocarbons which range from molecules containing one carbon atom to more than ninety. In addition sulfur, nitrogen, oxygen and traces of some metallic elements appear in varying amounts in the molecules.

By distillation and other processes ("refining") the crude oil is split up into a large number of different products, among them several types of fuel oil. These are classified according to their density, viscosity, and other properties. For cement kiln firing only the heavy quality is used, designated as Bunker C or No. 6 Fuel Oil in the USA. In Germany and Switzerland it is called Heavy Fuel Oil (Schweres Heizöl = Heizöl S).

Heavy fuel oil consists primarily of the residue from distillation and other refinery processes, such a thermal cracking, visbreaking, etc., with the addition of various diluents to meet various specification limits for the viscosity, density and sulphur content of the finished product.

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The chemical and physical properties of heavy residual fuels are dependent both on the origin of the crude oil and on the processing conditions in the refinery. The main properties of fuel oil derived from petroleum are subject to certain standards. For example, table 6 shows the German Standard DIN 51603.

Table 6: Requirements for heavy fuel oil as per German Standard DIN 51603

Flashpoint in closed cup : min	°C	65;100
Kinematic viscosity : max	mm^2/s(cSt)	at 50°C: 450 at 100°C: 40
Pour point :	°C	no specifications
Conradson carbon residue: max	% mass	15
Sulphur content: max	% by mass	2.8
Water content, not removable: max	% mass	0.5
Sediment content: max	% mass	0.5
Calorific value: min	MJ/kg	39.8
Ash (oxide ash): max	% mass	0.15

4.2 DENSITY

The density of fuel oil is strongly dependent on temperature. For 1°C increase in the temperature of heavy oil the density decreases, on average, by 0.63 %. Therefore the density must always be given with a reference temperature, usually 15°C (d_{15}). There are statistical relationships between density and other properties of fuel oil, specially the calorific value. Being relatively easy to measure density is suitable for regular and frequent quality control.

4.3 VISCOSITY

Apart from the calorific value viscosity is the most important property of fuel oils. Both pumping and atomisation require certain viscosities. These can be obtained by heating the oil because the viscosity depends strongly on temperature. Fig. 5 shows this together with the atomisation range and pumping limits. It can be seen that for optimal atomisation the heavy fuel oil must be heated to about 120°C. The oil temperature has to be strictly controlled because even small variations would cause marked changes of viscosity and thereby also atomisation.

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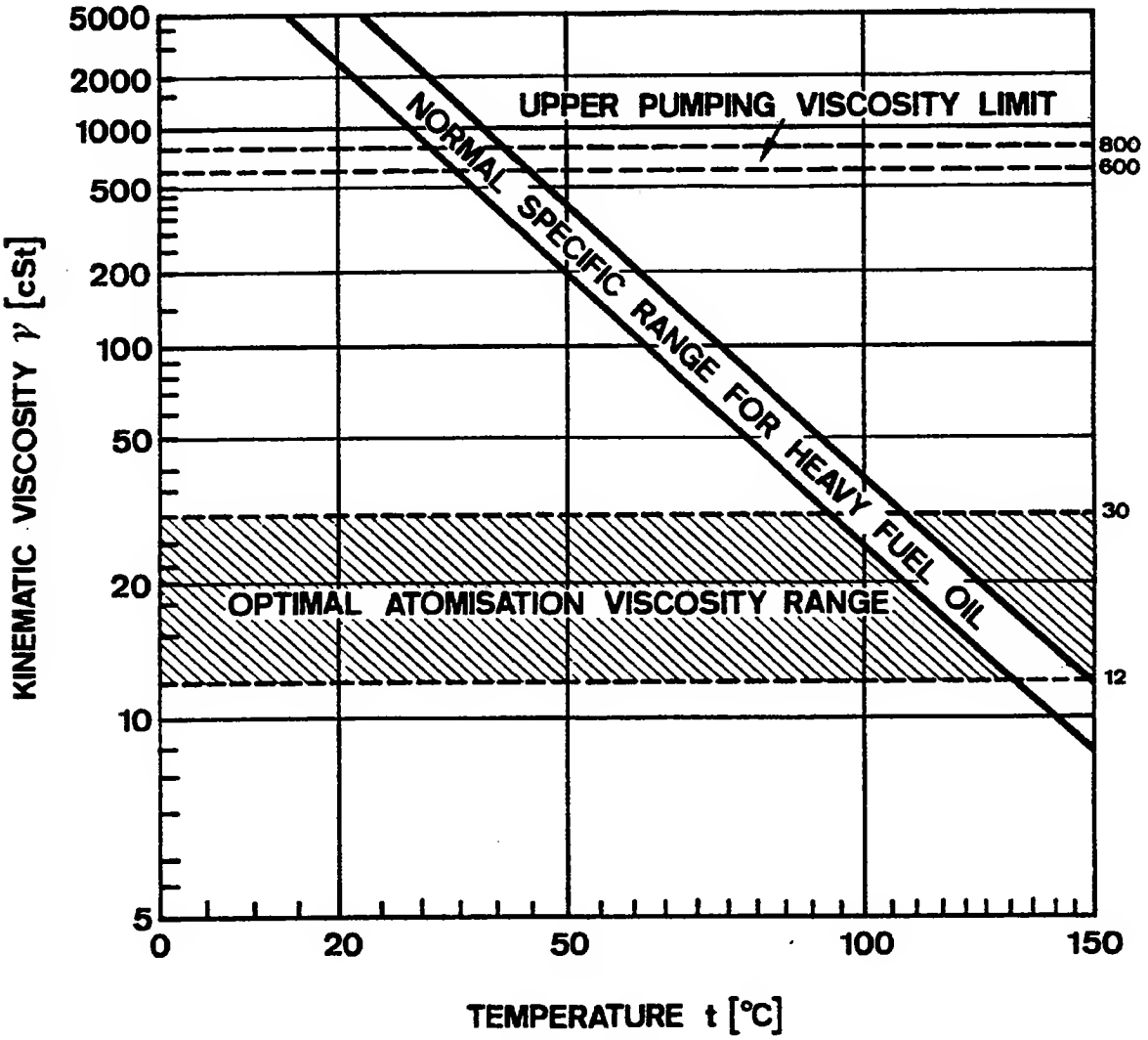
Fig. 5 Kinematic Viscosity of Fuel Oil as a Function of Temperature



The International Standard Unit of viscosity is the Poise (P):

$$1P = 10^{-1}\frac{N.s}{m^2}$$

It is called "dynamic viscosity". The ratio of dynamic viscosity to density is designated as "kinematic viscosity", usually measured in centistokes (cSt). In practice empirical units are often employed such as Redwood-seconds in Great Britain, or Saybolt seconds ("Universal" and "Furol") in the USA. Table 7 contains a comparison of the different viscosity scales.

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Table 7: Comparison of viscosity units

Saybolt seconds		Redwood seconds	cSt
Universal	Furol		
100	15	89	20.6
160	20	145	34.2
300	32	292	65.0
500	52	438	108
800	81	800	173
1500	150	1400	324
3000	300	2800	648

4.4 POUR POINT

Some fuel oils show an anomaly in their temperature/viscosity behaviour: already more or less above the upper pumping limit viscosities rise abruptly and the oil suddenly becomes a stiff gel. This temperature is called the "pour point". Obviously it is most important for storage and transport and depending on the climate upper limits should be fixed between buyer and supplier.

4.5 WATER CONTENT

Heavy fuel oil as delivered to consumers can contain slight traces of water. In general, the water content is well below 0.5 % mass. The relatively small quantity of water found in heavy fuel oil is normally present in a finely dispersed form which does not cause corrosion or problems in storage, transport or combustion.

A water content higher than 0.5 % mass can be caused by water condensation in storage tanks which are not completely full or by leaks from pipes, steam or hot water heating coils. Excess water gradually collects on the floor of the tank together with other constituents of the oil or contaminants (e.g. sediments, dirt) and forms a sludge. Regular checks and proper care of the tanks are, therefore, necessary in order to prevent build up of sludge on the bottom of the tank.

Care must be taken to ensure that when oil is removed from the tank none of the sludge is allowed to get into the burner circuit as it may cause blocking of filters and other problems. In normal circumstances, heavy fuel oil tanks should be cleaned every two to three years.

Even if some water in the oil is not likely to cause any trouble in kiln operation etc. it still means less calorific value and must not be neglected when comparing fuel prices.

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4.6 OTHER PROPERTIES

- calorific values 40 - 41 MJ/kg (see chapter 2.2)
- sulfur contents 1 - 3 % (see chapter 2.1)
- ash contents < 0.05 % (negligible for raw mix calculations)
- flash point:, >100°C (no danger of inadvertent or spontaneous ignition)
- carbon residue *) ~12 %

*) as measured by the Conradson test, i.e. heating of an oil sample with the exclusion of air until a solid residue of oil coke remains which is expressed in per cent mass of the oil sample. The procedure in this test, however, does not correspond to the conditions in a normal burner, where heavy fuel oil is first atomised into very fine droplets and is then burnt in suspension in an excess of air.

5. NATURAL GAS

The term "natural gas" is used to designate gases from below the surface of the earth. They consist of methane and higher paraffin's, nitrogen, carbon dioxide, hydrogen sulfide, and sometimes helium. In a narrower sense only combustible mixtures consisting predominantly of methane are called "natural gas".

Many natural gases, especially those from oil deposits, contain some natural gasoline, mainly paraffin's with 5 - 8 carbon atoms which are liquid at room temperature under atmospheric pressure. Depending on their gasoline content three types of natural gas are distinguished

- dry gas with < 10 9 gasoline per m^3
- lean gas with 10 - 50 9 gasoline per m^3
- wet gas with ; > 50 9 gasoline per m^3

Before feeding the gas into the supply system it is purified. Especially the evil-smelling and poisonous hydrogen suphide which may amount to 15 % of the crude gas has to be eliminated. The valuable gasoline is also taken out and recovered.

Besides sulfur many other things which can cause trouble when employing liquid or solid fuels such as grinding, preheating, atomisation etc. do not exist with natural gas. Supervision of the fuel quality is scarcely necessary. Only the calorific value may vary somewhat mainly because of varying concentrations of inert compounds like CO^2. Abrupt changes of ±300 kJ/Nm^3 have been reported. Therefore the installation of an on-line calorimeter is to be recommended which measures and records continuously the heating value of the gas.

Another disadvantage of gas is the low emissivity of the flame but owing to the dusty kiln atmosphere the heat transfer to the kiln charge is not as bad as might be apprehended.

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6. OTHER FUELS

Cement kilns offer the means and capacity to burn many kinds of combustibles. A large number of unusual "fuels" have already been tried in cement kilns, many of them successfully. Well known examples are oil shale, petrol coke, and used car tires. A lot more are listed in table 8 which can be considered as fairly comprehensive at the moment.

Table 8: By-product and waste fuels

SOLID		
Synthetic	**Vegetable matter or natural**	**Other**
waste tires (whole, Chips)	oil shale	domestic refuse
	coal wastes	refuse derived fuel (RDf)
rubber wastes	Low grade lignites	car shredding wastes
used battery casings	colliery benefaction rejects	oil Contaminated earth
plastic wastes	peat	
paper/Cardboard wastes	wood wastes	
fullers earth (from industrial oil preparation)	bark, wood chips	
	saw dust	
activated bentonite (from oil preparation)	rice husks	
	oil palm shells	
petrol coke (from refineries)	coconut shells	
graphite dust		
charcoal (wastes from steel industry)		

LIQUID	
Easy decomposable, low toxic	**Stable. toxic compounds (hazardous)**
tar (industrial by product)	poly-aromatic hydrocarbons (PAH)
acid tar	
sludge asphalt	polychlorinated byphenyls (PCB)
waste oils (from various sources)	chlorinated aromatics and alicyclics
oil sludge's	
solvents from paint industry	
petrochemical wastes	
chemical wastes (ketones, alcohol's, acetates, benzene. paint slurries)	

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The motive behind these activities was not in the first place to save high grade fuel but the disposal of waste materials and industrial or agricultural by-products for which no other use could be found. Conventional disposal of such materials in open deposits or burning in refuse incineration plants is becoming increasingly difficult. It is costly and may lead to environmental hazards. This does not, or at least only partly, apply to the combustion in cement kilns for the following reasons:

- with flame temperatures around 2000°C all organic compounds, even the most resistant chlorinated hydrocarbons, are completely destroyed.
- most of the acid combustion products and the toxic trace elements are absorbed by the counter-current of raw meal. This natural cleaning process is very effective.
- the flue gases are very efficiently deducted. In modern cement plants the residual dust content is not rarely below 50 mg/Nm^3.
- Thus toxic trace elements which tend to accumulate in this dust are effectively retained and can be eliminated for controlled disposal.
- most of the toxic elements will be incorporated into the cement clinker where they can do no harm because their concentrations are very small and are further reduced in mortar or concrete (cement content only 10 - 15 % of concrete weight) where they remain fixed practically insoluble.

From the macro-economic point of view, too, the burning of waste and by-product fuels in cement kilns is by far preferable to deposition or combustion in special refuse incineration plants. However, not all fuels are suitable for cement kilns, or only to a limited extent. Some of the restrictions which have to be considered are :

- the concentrations of volatile elements like Cl and S must not exceed certain limits in order to avoid circulation or emission problems. In many cases the chlorine content of a fuel limits its possible input into the kiln to avoid circulation troubles. Excessive sulfur input can lead to harmful SO_2-emissions.
- the overall composition of the fuel ash should be such that it can easily be incorporated into the clinker without deteriorating its quality.
- some wastes may cause handling difficulties with regard to transport, storage, safety etc.
- the burning of waste fuels is usually subject to permission by local authorities. The plant has to prove that stack emissions do not exceed existing limitations which may turn out to be rather expensive.

All these points have to be taken into account when calculating the financial aspects. In many cases the expenses saved on high grade fuel will barely be sufficient to cover the extra costs. Even so a cement plant should consider that it can create goodwill and a favourable image in public by helping other people to get rid of their by-products and waste without impairing the environment.

It is clear from the foregoing that no general rules for the combustion of waste or by-product fuels can be given, each case being different from the next and having its own particular aspects. However, to illustrate the subject an example for the successful use of a waste the burning of old tires in cement kilns as practised in a German Cement plant shall be described.

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The calorific value of tires is about 27 MJ/kg which is virtually the same as that of coal with an average mineral content and moisture. The tires are fed whole directly into a SP-kiln. The system begins with a forklift truck movement of tires from a storage pile onto a conveyor. This carries the tires up to an automatic balance. The tires having very different sizes weighing is indispensable to maintain a constant mass flow of the fuel. The number of tires fed per unit of time is automatically adjusted according to their weight. The tires enter the kiln via a chute and through an air lock to minimise heat loss into the transition chamber where temperatures are high enough to ensure complete combustion without any risk of environmentally negative aspects. The iron wires of the tires are oxidised to FeO which is incorporated into the clinker along with the coal ash. Of course, the raw mix has to be adjusted accordingly. Provided this is done the clinker quality will not be affected. Up to 20 % of the fuel has been replaced by old tires.

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Chapter 6

Clinker Burning Behaviour of Heavy Metals

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Clinker Burning Behaviour of Heavy Metals

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1. INTRODUCTION

For hundreds of millions of years the heavy toxic metals remained buried in the ground in non-soluble forms, mainly as oxides and sulphides, and as such were not a pollutant of the biosphere. The use of metals by man started about 6000 B.C. when natural metallic gold, silver and copper supplemented the use of stone, bone and wood. Between 4500 and 3500 B.C. the discovery that metals such as copper could be extracted from native ores, made malleable by heat and mixed with other metals to form alloys, formed the basis of the vast metal industries of today. Accompanying this came potential hazards to health.

The early ores were mined from pits without adequate ventilation and resulted in an exposure to dusts and fumes whose composition was unknown and whose toxicity was on the whole not even suspected. Strabo, in describing the arsenic mines in Pontus, however reported:

"the air in the mines is both deadly and hard to endure on account of the grievous odour of the ore, so the workmen are doomed to a quick death".

From the early beginnings modern technology has made available metals whose toxic properties were first realised after incidents of ill-health or death caused by inhalation of dust and fume or by contact with the skin. Such metals include beryllium, selenium, cadmium, gallium, indium, etc. That cadmium and selenium have toxic properties was first realised after these metals had already achieved commercial success.

What significance do heavy toxic metals have today in the cement industry? Since the mid 1980's strict regulations for gaseous emissions and immissions have started to be implemented in highly developed countries. In Europe, especially in Switzerland and Germany, very low limits have been set for the emission of various elements such as defined in the Swiss (LRV) and German (TA-Luft) Clean Air Acts. These regulations, given in Table 1, classify the toxic elements, mainly heavy metals, into various classes depending on their toxicity.

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Table 1: **The Limits for the Emission of Heavy Toxic Metals as Defined in the German (TA-Luft) and Swiss (LRV) Clean Air Acts**

Element	Class	Class Emission Limits (mg/Nm^3)
Cd	1	
Hg	1	0.2
Tl	1	
As	2	
Pd	2	
Ni	2	1
Se	2	
Pd	2	
Cr	3	
Pd	3	
Pb	3	
Pd	3	
Cr	3	5
Pd	3	
Sb	3	
Sn	3	
V	3	
Zn	n.c.	n. defined

Due to the very low emission limit of 0.2 mg/Nm^3 for the sum of the Class 1 elements (cadmium, mercury and thallium) and the high volatility of the elements mercury and thallium, Class 1 is of utmost importance to the cement industry. That thallium can be emitted from a cement plant is well documented (Figure 3) following the "incident" in 1979 at Dyckerhoff's Lengerich plant in which flora and fauna around the plant were shown to have suffered as a result of its emission. The source of this thallium was mainly the iron corrective used (4.8 ppm) and also the coal (1.2 - 1.8 ppm), both natural mate- rials.

With the now common and ever increasing use of secondary materials, combustible and non-combustible, increased quantities of the heavy, toxic elements just defined are now being introduced into cement kilns. Which of these elements are the most common ones encountered, where do they come from and how do they behave inside the kiln? What happens to them after the cement is hydrated?

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2. ESSENTIAL AND TOXIC ELEMENTS

Whereas some elements are essential, others are decidedly toxic. Which elements fit into each category?

2.1 Essential Elements

Those elements considered to be essential are given in Table 2.

Table 2: Essential and Toxic Elements (Iyengar and Sansoni, 1976)

Structural Elements: C, Ca, H, N, O, P, S

Electrolyte Elements: Ca, Cl, K, Mg, Na

Trace Elements:

- Essential
  - biologically important: Co, Cr, Cu, F, Fe, I, Mg, Mn, Mo, Ni, Se, Si. Sn, V, Zn
  - Clinically significant: Co, Cr, Cu, Fe, Mg, Se, Zn
- Suspected to be Essential: As, Ge, Rb
- Regularly Found in Tissue: Al, B, Br, Ga, Li, Sc, Sr, Ti
- Toxic
  - potentially toxic: As, Cd, Hg, Pb, Sb, Se
  - environmental contaminants: Cd, Hg, Pb
  - industrial hazards: Be, Bi, Cr, Mn, Ni, Sb, Th, Te,U, V, W

Others: Ag, Au, Ba, Ce, Cs, Nb, Pt, rare earths, Te, W, Zr

Radioactive Contaminants: Po, Ra, Rn, Th, U, Am, Cm, Np

Not all elements are toxic, indeed many are even essential to life. As can be seen in Table 2, the elements H, C, N, O, Na, Mg, P, S, Cl, Ca are the major elements necessary, the additional elements B, F, Si, V, Cr, Mn, Fe, Co, Ni, Cu, Zn, Se, Mo, Sn and I being necessary in only trace quantities. The main body of the essential trace elements is grouped between V and Zn in the first transition series of the periodic table. With the exception of Mo, Sn and I, all the essential elements lie within the first four layers of the table. The form in which these elements is present is also important as some compounds, eg. HCN (prussic acid) contain essential elements but must definitely be classed as being highly toxic.

Regarding the quantity of the trace elements required this varies from element to element. As given in Table 3, showing the situation for a 70 kg adult, the daily requirements range from 0.03 mg for Co up to 15 mg for Fe. These elements are not accumulated within the body and are excreted and so must be continually replaced. The total burden of the essential elements is also shown in Table 3 with Fe at 4.1 g (4100 mg) and Zn at 2.3 g (2300 mg) being the most abundant.

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Table 3: Estimated Daily Intake and Human Body Burden of Essential Trace Elements

Element	Class*	Daily Intake (mg)	Burden (mg/70 kg)
Co	2	0.03	1
Ni	2	0.45	10
Se	2	0.06	15
Cr	3	0.06	6
Cu	3	3.2	100
V	3	2.5	30
Fe	n.c.	15	4,100
Mn	n.c.	5	20
Mo	n.c.	0.35	9
Zn	n.c.	12	3,200

* Class in German and Swiss Clean Air Acts

2.2 Toxic Elements

2.2.1 Toxicity

Some of the substances which are emitted to the atmosphere during the burning of cement clinker are annoying and some can be even harmful to the environment. Annoying substances are those which only affect well-being, whereas harmful substances impair the biological activity (metabolism, growth) of an organism. In medical terminology the latter are referred to as "atoxic". While it is usual to speak globally of toxic elements, toxicity itself is by no means a fixed, well-defined property. Firstly it depends to a decisive extent on the chemical compound in which the particular element is found. For instance sulphur is very toxic as dioxide, hydrogen sulphide etc., but many other sulphur compounds are perfectly harmless e.g. sulphates like gypsum. It is also very important whether a compound is easily resorbed by the organism or not. Examples of compounds containing well-known "toxic" elements which on account of their being insoluble cannot be resorbed and are therefore practically harmless are:

- arsenic sulphide
- barium sulphate
- mercury sulphide

The tolerance limits for toxic elements to be found in official regulations are of course determined on the basis of the most dangerous compounds of the element concerned.

The toxic efficacy of a substance also depends to a large degree on whether it is absorbed by inhalation, ingestion or skin exposure. For example, metallic mercury can pass through the stomach and intestines without serious effects whereas inhaling the vapour can be very harmful. The toxicity of mercury compounds also depends on their solubility which in turn determines their absorption and distribution in the organism involved. Thus whilst metallic mercury and almost all mercury compounds are toxic , mercury sulphide is non-toxic because of its practical insolubility in the body.

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Furthermore, a substance which harms one particular kind of organism can leave others entirely unaffected or even promote their biological activity. Moreover, the biological effect depends largely on the dose, as already recognized by the Swiss born physician and alchemist Paracelsus (1493 - 1541) who established the role of chemistry in medicine:

"the dose alone decides whether a substance is poisonous or not"

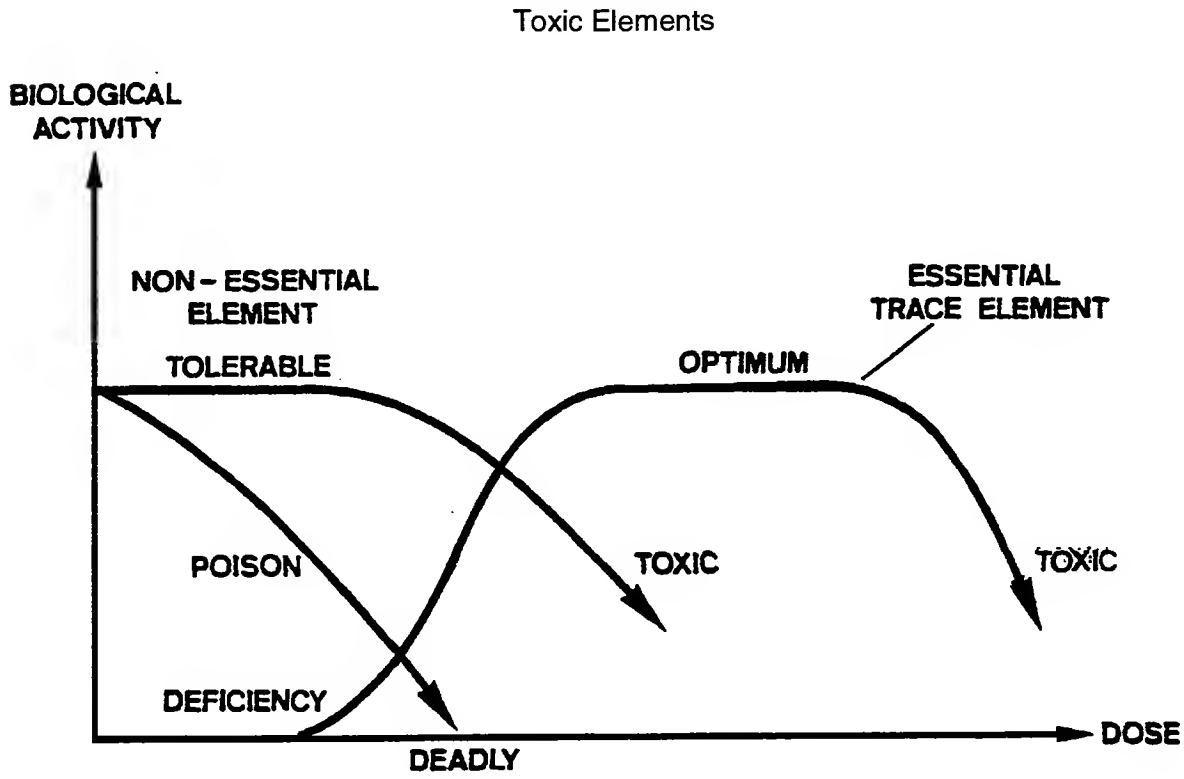
"if given in small doses what makes a man ill also cures him"

"miners disease results from inhaling metal vapours and is not a punishment for sin administered by mountain spirits"

No substance is absolutely harmless as there is always an upper limit to the concentration which may not be exceeded without serious consequences for the organism. For example, common salt is a dangerous poison when consumed in too large quantities. On the other hand, there is no absolutely toxic substance because below a certain minimum concentration even the strongest poison becomes ineffective.

Figure 1 shows schematically the basic relationship between dose and effect. In order to assess the toxicity of a substance, all the above factors have to be taken into account.

Figure 1: Troxicity is a Question of the Dose



In many cases it has been found that elements which are essential to life can also be toxic depending on the dose and the form in which the element is present. In general it can be said that the toxicity of elements roughly parallels their abundance in the Earth's crust and in sea water. Elements that are present in low levels are more likely to be toxic as shown in Table 4 in which those metals for which toxic properties have been reported are given. In this table mainly the toxic metals are included, the toxic non-metals having being omitted.

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Table 4: Composition of the Lithosphere indicating Essential and Toxic Elements

Concentration (%)	Element
46.6	**O**
27.7	**Si**
8.1	*Al*
5.0	**Fe**
3.6	**Ca**
2.8	**Na**
2.6	K
2.1	**Mg**
10^{-1}	*Ti* **H, Mn, P**
10^{-2}	**F, S, C, Cl**, *Rb, Sr, Ba, Zr*, **Cr, V, Zn**
10^{-3}	**Ni, Cu**, *Li*, **N, Sn, Co, Pb**, *Th*
10^{-4} (ppm)	*As*, **B, Mo**, *Br, W, U*
10^{-5}	*Sb, Bi, Ag*
10^{-6}	*Hg*, **Se**
10^{-7} (ppb)	*Au, Pt*, He
10^{-8}	**I**

essential elements in **bold text**, toxic elements in italics

Also presented in Table 4 are those elements previously shown in Table 3 which are essential for life. As can be seen many of the elements reported are considered as being both essential and toxic. Included in this category are the elements B, Mg, V, Cr, Mn, Co, Ni, Cu, Zn, Se, Mo and Sn. When, for example, a manganese deficiency occurs in humans this can be accompanied by anaemia, and bone changes in children. Considering the toxic effects of manganese these are listed as including metal fume fever, psychological and neurological disorders, the two latter being contracted after long term exposure to air containing freshly drilled dust particles rich in manganese ore. When a deficiency of copper occurs this is accompanied by a severe anaemia. Metallic copper is much less toxic than its salts and for humans only 1 g of $CuSO_4$ is highly toxic, causing vomiting, diarrhoea, convulsions and is often fatal.

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3. ANALYSIS OF TOXIC METALS

In order for a cement plant to be able to safely use secondary materials containing toxic metals without the risk of excessive emissions, it is essential that the materials received conform to the reference samples initially submitted. To do this it is necessary to carry out control analyses for those elements listed in the applicable Clean Air Act. This is especially the case, because of their volatility, for the Class 1 elements Hg, Tl and Cd but also for the elements of Classes 2 and 3 should these be present in appreciable concentrations. To avoid the necessity of carrying out a full quantitative analysis, which would be costly and time consuming and would attempt to determine elements even when they were not present, it is advisable to first carry out a qualitative analysis. This is the method used at HMC/TC.

3.1 Qualitative Analysis of Toxic Elements

A qualitative analysis (spectrum) of all the toxic elements with the exception of Be can be carried out using such methods as X-ray fluorescence (XRF) and inductively coupled plasma emission (I.C.P.) spectroscopy, provided that the instrument is of the sequential and not of the multi-channel type. Whereas the I.C.P. method requires the sample being measured to be in liquid form, both solid and liquid samples can be measured by the XRF, the latter being possible when the spectrometer is flushed with helium instead of operating under vacuum. Because of the comfort of the operating software and the general ease of operations the XRF technique is preferred at HMC/TC.

An example of an XRF spectrum is given in Fig. 2. This was taken using a sample of slightly contaminated excavated material being considered for disposal by incineration in a Swiss cement kiln. In this diagramme the wave length of the characteristic radiation (expressed as two theta in degrees for the measuring crystal) is presented on the x-axis and the intensity of the radiation (expressed as counts/sec) on the y-axis. As can be seen the presence of Pb (410 ppm) and Zn (7700 ppm) can easily be determined in this figure. The peak heights observed are not the same for equal concentrations of different elements. The peak height for different elements is not proportional to the quantity present. The values for these elements were found in subsequent quantitative determinations. Attempts to ascertain the qualitative presence of Cd, Hg and Tl, which were later shown to be present in quantities of 4.4, 0.14 and < 1 ppm, were not successful by this technique and should in any case always be quantitatively determined.

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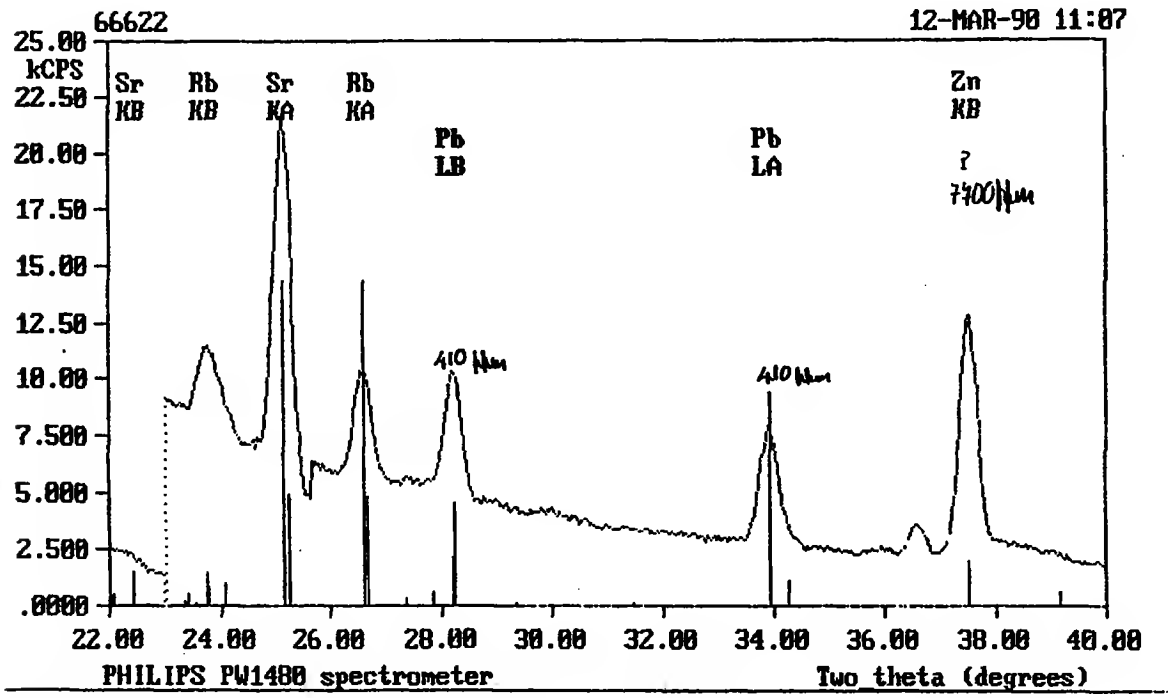
Figure 2: XRF Specturm of excataed material

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3.2 Quantitative Analysis of Toxic Metals

Several methods can be used for the quantitative analysis of toxic metals and include those shown in Table 5. Of these the most sensitive of the methods that can generally be used is certainly the AAS technique with a graphite furnace which allows a lower limit of detection than the flame AAS, XRF and ICP methods. The limits of detection of the cold vapour and hydride AAS techniques lie within those of the two groups of methods previously mentioned. With regard to the limits of measurement these are some 4 to 5 times higher than the limits of detection.

Table 5: Detection Limits (ppm) of Environmentally Sensitive Elements by Various Measurement Techniques

Element	Class	AAS Flame	AAS Furnace	AAS Cold Vapour	AAS Hybride	ICP	XRF
Be	1 Car.	1	0.003			0.003	
Cd	1	1	0.0002			1.5	1 - 10
Hg	1	200		0.1	0.05	12	
Tl	1	30	0.01			27	
As	2	100	0.08		1	30	
Co	2	5	0.008			3	
Ni	2	8	0.05			4	1 - 10
Se	2	100	0.05		1	30	
Te	2	70	0.03		0.50	20	
Cr	3	3	0.01			3	
Cr	3	2	0.005			1	
Pb	3	20	0.007			25	
Pb	3	20	0.05			13	1 - 10
Pt	3	50	0.20				
Rh	3	2	0.01			3	
Sb	3	50	0.05		5	40	
Sb	3	20	0.03		5	30	
V	3	20	0.10			3	
Zn	n.c.	0.6	0.003			2	1 - 10

In HMC/TC laboratories, where all the above methods are available Hg is measured by the hydride process and all the others with a few exceptions using the graphite furnace AAS technique. These methods were chosen because, being wet chemical techniques, there is no matrix correction and standard calibration samples are easily made by the addition method.

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3.3 Evaporation of Mercury during Handling in Laboratory

It should be pointed out that sampling and sample preparation are most important aspects of a trace element analysis. Contamination or the loss of a portion an element are an omnipresent danger. This is actually the reason why there are often doubts about the validity of values given in literature, particularly at low concentration levels. An evaporation test demonstrated that varying amounts of mercury can escape during the preparation and drying of a sample (see Table 6).

Table 6: Hg-Concentrations in Precipitator Dust and Sewage Sludge Samples after Treatment at Different Temperatures

Temperature (°C)	ESP Dust (ppb)	Sewage Sludge (ppb)
20	430	6,400
100	90	6,400
200	60	6,300
300	40	700

4. SOURCES OF TOXIC METALLIC SUBSTANCES

4.1 Common Sources of Toxic Metallic Substances

To provide an overview with regard to the normal uses to which the elements of Classes 1, 2 and 3 of the previously mentioned Clean Air Acts are put, Appendix II gathers together some relevant information. As can be seen all of the elements are extracted from ores in which they are present as either oxides or sulphides and are thus found in nature in water insoluble forms. In nature they are often only present in low concentrations but after extraction are then present in much higher concentrations and may be in a soluble form. The information given in Appendix II is intended to inform the reader which industries are likely to have by-products that could contain the listed elements. Although it is very unlikely that any of the primary products listed in the "uses" table would ever be considered as secondary materials the same cannot be said for the by-products of these manufacturing industries, which because of restrictions on dumping materials, are now looking for other means of disposal. Today some Group plants are in fact disposing of preservatives, pigments, dyes etc. which enter the kiln via secondary materials such as waste oils or woods.

4.2 Natural Cement Raw Materials Components

As shown earlier in Table 4 the heavy toxic metallic elements make up a small but definite proportion of the materials in the Earth's crust. As such their introduction into a cement kiln via the natural raw materials cannot be prevented. This question of interest is therefore how much of the classified and non-classified elements (at the ppm and ppb levels) can be present in the various rock types used for raw mix components?

The answer to the above question is given in Table 7 for the main rock types limestone, limey marl and clays and in Table 8 for the commonly used corrective, iron ore. As can be seen in Table 7 the limestone and limey marl components tend to have lower contents of heavy metals than clay components. This does not mean, however, that all limestones can be assumed to be low in heavy metals. In fact Zn levels of 1900 ppm and V levels of 3000 ppm have been encountered in extreme cases.

For the limey marls reported the values were all low and in the same magnitude or lower than the average levels in the Earth's crust. The highest level was 55 ppm reported for Zn.

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When considering the heavy metals present in iron ore, an often used corrective, as shown in Table 8 such materials can often contain concentrations of heavy metals appreciably higher than the average for the crust. Such elements include Tl (400 ppm), As (680 ppm), Cr (1500 ppm), Pb (8700 ppm) and Zn (9400 ppm). It is fortunate that only small quantities of iron corrective have to be used in cement kiln raw mixes.

In the case of the clays values very much higher than in the Earth's crust were reported for Cd (500 ppm), Cr (1500 ppm), V (3600 ppm) and Zn (1300 ppm).

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Table 7: **Range (ppm) of Heavy Metals Reported for Rock Types Used as Raw Mix Components**

Element	Class	Limestone	Limey Marl	Clay	Earth's Crust
Cd	1	0.02 - 0.50	0.04 - 0.35	0.02 - 500	0.2
Hg	1	0.005 - 0.10	0.009 - 0.13 0.02 - 0.15	0.08	
Tl	1	0.05 - 1.8	0.07 - 0.68	0.20 - 1.6	0.5
As	2	0.1 - 12		3 - 25	1.8
Co	2				25
Ni	2	1.4 - 13	5.9 - 21	11 - 71	75
Sb	2				0.05
Te	2				0.01
Cr	3	0.70 - 16	4.6 - 35	20 - 1500	100
Co	3				55
Pb	3	0.27 - 21	1.3 - 8.5	1.0 - 219	13
Sb	3				0.2
Sn	3				2
V	3	5 - 3000		30 - 3600	135
Zn	n.c.	0.10 - 1900	24 - 55	2 - 1300	70

Table 8: **Range (ppm) of Heavy Metals Reported for Correctives used in Cement raw Mixes**

Element	Class	Iron Ore	Earth's Crust
Cd	1	0.2 - 15	0.2
Hg	1		0.08
Tl	1	0.07 - 400	0.5
As	2	4 - 680	1.8
Co	2		25
Ni	2	10 - 340	75
Se	2		0.05
Te	2		0.01
Cr	3	9 - 1400	100
Cu	3		55
Pb	3	9 - 8700	13
Sb	3		0.2
Sn	3		2
V	3		135
Zn	n.c.	6900 - 9400	70

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4.3 Secondary Raw Meal Components

To improve the economics of clinker production or improve cement quality many plants are today using so called secondary components in their raw mixes. These materials are mainly waste products from other processes and are often available in large quantities. Examples of such materials are given in Table 9 with fly ash and foundry sand in a few cases being the main alumino-siliceous raw mix component. Of these the purest material is probably the man made FCC catalyst but, depending on the source of the oil being processed in the refinery, this material can contain up to 1860 ppm of Ni, 1265 ppm of Sb and over 5000 ppm of V. As can be seen a natural material such as fly ash can contain up to 530 ppm of As, 600 ppm of Ni, 650 ppm of Cu, 800 ppm of Pb, 500 ppm of V and 910 ppm of Zn. In cases in which coal is used as the main fuel and fly ash as a major mix component, appreciably levels of heavy metals may be introduced into the kiln. Foundry sand is a component that is just starting to be used but as seen in Table 9 may contain appreciable maximum levels of Ni (1520 ppm), Cu (1500 ppm) and Pb (2300 ppm).

Table 9: Range of Heavy Metals (ppm) in Secondary Components used in Cement Raw Mixes

Element	Class	Foundry Sand	FCC Catalyst	Fly Ash	Earth's Crust
Cd	1	300	< 0.6	0.2 - 8	0.2
Hg	1	2	0.04 - 2	0.2 - 0.8	0.08
Tl	1		< 1.5 - 145	2 - 15	0.5
As	2	2	2 - 6	40 - 530	1.8
Co	2		13 - 93	90 - 250	25
Ni	2	1,520	265 - 1860	150 - 600	75
Se	2	1	< 11 - 128	1 - 35	0.05
Te	2				0.01
Cr	3	10	58 - 142	60 - 330	100
Cu	3	1,500	6 - 93	140 - 650	55
Pb	3	2,300	12 - 174	125 - 800	13
Sb	3		<25 - 1265	0.5 - 90	0.2
Sn	3	100	7 - 23	(38)	2
V	3	5	1050 - 5515	230 - 500	135
Zn	n.c.	130	32 - 115	350 - 910	70

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4.4 Toxic Elements in Raw Meal

The levels of the heavy toxic elements previously given for the mix components, major, minor and secondary give the maximum range encountered on a world-wide basis. Of more importance is the levels of the classified elements that are actually encountered in practice and these are presented in Table 10. As can be seen the concentrations of many elements are less than the average for the crust. The only elements which are present in concentrations above the average for the Earth's crust are limited to Tl (6 ppm) and As (28 ppm). These levels can in no way be considered as high.

Table 10: Heavy Toxic Metal Levels (ppm) in Raw Meal

Element	Class	Raw Meal	Earth's Crust
Cd	1	0.04 - 1	0.2
Hg	1	< 0.02 - 0.6	0.08
Tl	1	0.1 - 6	0.5
As	2	3 - 28	1.8
Co	2	11	25
Ni	2	18 - 30	75
Se	2	< 0.2	0.05
Te	2		0.01
Cr	3	23 - 39	100
Cu	3	17	55
Pb	3	4 - 15	13
Sb	3		0.2
Sn	3		2
V	3	32 - 102	135
Zn	n.c.	15 -90	70

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4.5 Natural Fuels

Because the normally used fuels coal and oil have been formed over millions of years by naturally occuring processes in the Earth's crust, it will not now be surprising that these also contain heavy, toxic elements in trace quantities. The quantities reported by various authors are given in Table 11.

As can be seen brown coal contains the smallest quantities of classified heavy metals with only Zn at up to 70 ppm being above the 10 ppm level. For hard coal the maximum quantities of the major heavy metals present are Pb (270 ppm), Zn (220 ppm), V (180 ppm), Ni (96 ppm) and Cr (80 ppm). For bunker oil the heavy metals present in the maximum concentrations are V (117 ppm), Zn (85 ppm) and Ni (43 ppm). For petcoke, a residue of oil distillation, the levels of the heavy metals are more concentrated with the maximum levls reported being Ni (355 ppm) and Cr and Pb at both just over 100 ppm.

Table 11: Range of Heavy Metals (ppm) in Natural Fuels Used in Clinker Production

Element	Class	Hard Coal	Brown Coal	Petroleum Coke	Bunker Oil	Hard Coal Fly Ash	Earth's Crust
Cd	1	0.1 - 10	0.1 - 2.4	0.1 - 4.0	0.02 - 0.4	0.2 - 8	0.2
Hg	1	0.1 - 1.0		0.01 - 0.09		0.2 - 0.8	0.08
Tl	1	0.1 - 1.2	0.07 - 0.3	0.04 - 3.1	< 0.02 - 0.12	2 - 15	0.5
As	2	2 - 50	0.3 - 9		< 0.01 - 0.1	40 - 530	1.8
Co	2	5 - 34				90 - 250	25
Ni	2	15 - 96	0.6 - 1.9	24 - 355	5 - 43	150 - 600	75
Se	2	0.6 - 3.4				1 - 35	0.05
Te	2	0.2 - 0.6					0.01
Cr	3	5 - 80	0.9 - 8	3.6 - 104	2 - 4	60 - 330	100
Cu	3	10 - 60				140 - 650	55
Pb	3	10 - 270	0.9 - 8	6 - 102	1 - 34	125 - 800	13
Se	3	0.4 - 2.0				0.5 - 90	0.2
Sn	3					(38)	2
V	3	30 - 180	2 - 7		2 - 117	230 - 500	135
Zn	n.c.	6 - 220	1 - 70	(7.6)	5 - 85	350 - 910	70

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4.6 Secondary Fuels

Table 12 lists the range of heavy metals used in the most commonly used secondary fuels. As can be seen, the levels of many elements are below the average for the Earth's crust but, when these latter values are exceeded, this is by a considerable margin and in elements specific for the individual fuel concerned.

For example waste automobile oil is enriched in those elements which are introduced by the wear of bearing materials. Thus elevated values of Pb (21'000 ppm) and Zn (3000 ppm) are to be found in waste oil. Automobile tyres are especially enriched in Zn for which levels between 0.93 % and 2.05 % are reported. The impurities in used ion exchange resins will depend on the material stream which they have been used to clean up, with especially high levels being reported for Pb (6400 ppm) and Zn (3900 ppm). In contrast the levels of the heavy metals in used wood are considerably lower but with maximum values of Cd (16 ppm), Se (5ppm), Cu (77 ppm), Pb (70 ppm), Sb (10 ppm) and Zn (560 ppm) which are definitely above the average level in the crust.

Table 12: Range of Heavy Metals (ppm) in Secondary Fuels Used in Clinker Production

Element	Class	Waste Oil	Fullers Earth	Ion Exchange Resins	Automobile Tyres	Used Wood	Earth's Crust
Cd	1	4	0.01 - 2	9 - 50	5 - 10	1.5 - 16	0.2
Hg	1					< 0.05	0.08
Tl	1	0.02	0.2	0.03 - 0.07	0.2 - 0.3	0.5	
As	2						1.8
Co	2						25
Ni	2	3 - 30	0.001 - 30	8 - 87	77	11 - 50	75
Se	2					5	0.05
Te	2						0.01
Cr	3	5 - 50	2 - 11	20 - 330	97	16 - 25	100
Co	3					40 - 77	55
Pb	3	10 - 21000	2 - 2500	150 - 6400	60 - 760	38 - 70	13
Se	3					10	0.2
Sn	3						2
V	3						135
Zn	n.c.	240 - 3000	10 - 480	56 - 3900	9300 - 20500	130 - 560	70

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5. BEHAVIOUR OF TOXIC METALS IN CEMENT KILNS

Now that we know the concentrations of the heavy toxic metals in the raw material components and fuels let us now turn our attention to the behaviour of these metals in the cement kiln.

5.1 Volatility of Toxic Element Compounds

To gain an impression of the volatility behaviour of the toxic metals listed in the German and Swiss Clean Air Acts, the vapour pressures of the most likely compounds to be present in a cement kiln, i.e. when the elements are not incorporated into the lattice of the clinker minerals, have been reviewed by Haegermann. Those having a partial pressure of at least 1 mm Hg at 300°C are listed in Table 13. As can be seen in addition to the high volatility of Hg compounds, which are essentially completely emitted from cement kilns, other elements listed in the German clean air act (Be, As, Se and V) can all form compounds with vapour pressures of 1 mm Hg at temperatures of lower than 300°C and are hence capable of being emitted from cement kilns.

Table 13: Heavy Metal Compounds of Class 1, 2 and 3 Elements which have a Partial Pressure of over 1 mm Hg at or below 300 °C

Element	Class	Fluoride	Chloride	Bromide	Iodide	Oxide	Sulphate	Hydride	Metal
Be	1 carcin		*	*	*				
Cd	1								
Hg	1		*	*	*				*
Tl	1								
As	2	*	*	*	*	*		*	
Co	2								
Ni	2								
Se	2	*	*			*			
Te	2								
Cr	3								
Cu	3								
Pb	3								
Sb	3								
Sn	3								
V	3								

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5.2 Distribution of Toxic Heavy Elements

For a comprehensive assessment of the distribution volatilization and emission of an element, the exact balance of its intake and output must be known. To this end it is necessary to analyse all material and gas streams, such as raw mix and fuel, in order to determine the intake and the amount present in clinker and discarded dust.

Normally these calculations of the balances indicate that the total intake of an element from raw material (Ir) and from fuel (If) minus discharges via clinker (Dc) and discarded dust (Dd) equals the emission (Ex) plus that portion (Rk) retained inside the kiln system:

$$Ir + If - Dc - Dd = Ex + Rk \quad (1)$$

According to the behaviour of the elements in cement kilns, three separate cases can be distinguished, one for non-volatile and two for volatile compounds:

- Case 1: For non-volatile compounds: Ir + If = Dc, i.e. there is no emission and no accumulation in the kiln system: Ex, Rk = 0
- Case 2: For volatile compounds: Ir + If > Dc
  - (a): surplus intake remains totally in the "kiln" system with no emission, i.e. Ex = 0
  - (b): surplus intake is wholly emitted with no retention in the kiln: Rk = 0

The transposition of equation (1) results in:

$$Dd + Rk = Ir + If - Ac - Ex \quad (2)$$

To a first approximation the terms on the right side of the equation (2) can be considered as constant:

$$Dd + Rk = \text{constant} \quad (3)$$

By means of equation (2) it is possible to calculate that a certain portion of a compound will remain in the kiln system if no dust is rejected and also the amount of dust required to be rejected in order to keep the retained portion Rk below a certain limit. For this purpose it is necessary to know the emission Ex. Only limited data are available for the latter as this is sensitive information which cement plants do not make readily available.

The quantity emitted Ex can, however, be estimated if it is known which of the three cases best describes the compound in question.

According to HMC/TC's own investigations and the findings of the Research Institute of the German Cement Industry (VDZ), the Class 2 and 3 elements are not likely to exceed the emission limits of 1 resp. 5 mg/Nm^3, respectively, in normal operation and with normal raw materials. Even if sewage sludge is used as combustible, as partial substitute for coal, the emission of all Class 2 and 3 elements is well below the limits. The elements of Class 1, especially Hg and Tl show quite a different behaviour. This Class, characterized by its very low emission limit of 0.2 mg/Nm^3, represents a permanent danger, a fact which also becomes evident from the concentration of the elements in the Earth's crust shown in Tables 7 to 12. Therefore, it is worthwhile to give a more detailed description of the findings of these investigations.

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5.3 Distribution of Cadmium

According to HMC/TC's measurements, the degree of binding of cadmium in clinker shows very large variations from 41 to 100 %. This corresponds to the findings of Sprung (VDZ). The reasons for the variations are not clear. Relatively small temperature fluctuations and also the burning conditions will have an effect. Due to the fact that the clinker absorbs only part of the cadmium and the emission is insignificant, the rest will remain in the kiln system causing recirculations with cadmium enrichment in different materials. The cadmium concentration increases from raw meal to electrostatic precipitator dust to the hot meal. The dust shows large differences in concentration between direct and compound operation. These findings show that cadmium as a partly volatile element belongs to case 2(a) with no emission (Ex = 0) but with accumulation in the kiln system.

5.4 Distribution of Mercury

The investigation of different materials has shown that the mercury contents vary widely and are relatively low in raw materials and coal compared to the average concentration in the lithosphere.

Mercury was not detected in clinker, however, in some cases an accumulation was observed in kiln dust, although these concentrations (max. 190 mg/t) are insignificant compared to the other elements, e.g. thallium. Furthermore, it should be mentioned that the mercury emission in compound operation is appreciably lower than that in the direct mode. In compound operation the emitted quantities are even lower than the intake. Obviously, much of the mercury vapour is retained in the raw mill perhaps by adsorption on the surface of the meal particles. In direct operation, the retained mercury will be released again and is emitted.

Summarising, it can be generally stated that the mercury introduced by raw materials and fuels is completely emitted in gaseous form (in the dry process mainly in direct operation) so that its behaviour is that of a volatile compound without retention in the kiln (Rk = 0, i.e. case 2b). These findings were also confirmed by the stack gas analyses available.

5.5 Distribution of Thallium

According to German investigations (Sprung, VDZ) and HMC/TC findings, cement raw mixes contain about 100 to 800 mg thallium per ton corresponding to 150 to 1200 mg thallium per t clinker. The concentration can be much higher in kiln feed and coal dust because large quantities of thallium are accumulated from the kiln gases during drying and grinding. Furthermore, the kiln feed shows significant differences between direct and compound operation: The thallium concentration increases considerably during direct operation and may ultimately be up to 8 times as high as in the compound mode. The measurements reveal that neither the clinker nor the stack gas discharge the kiln system of thallium. Only very high exhaust gas temperatures above 200°C (long dry and wet kilns) may allow some thallium vapour to escape. Nearly the entire intake remains in the system. Therefore, thallium is to be considered as a volatile compound with no emission (Ex = O) and no output by the clinker (Ac = O), i.e. all Tl is accumulated in the kiln system. Hence, as will be appreciated in light of the "Lengerich incident" it is particularly important to relieve the kiln/grinding system e.g. by creating a "valve" in order to avoid excessive accumulations of thallium which ultimately lead to kiln dust with extremely high Tl concentrations. Especially in cases of disturbed operations e.g. insufficient or lacking precipitator function could the emission of Tl become considerable.

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5.6 Typical Balance of Toxic Elements In Kilns

5.6.1 Wet Kiln

A simple balance of the toxic elements in a large wet kiln fired with coal is displayed in Table 14 in which the percentage input as coal and slurry as well as the output via the clinker, dust and stack gas are given. Here a simplified case is presented in that the totals for the inputs and outputs are all 100 %. As can be seen the major input always comes from the slurry but for As and Be almost 25 % is delivered from the coal. At the other end of the scale less than 10 % of the elements Pb, Tl and Zn comes from the fuel.

Table 14: Balance (% total) of Toxic Elements in a Large Wet Kiln

Element	class	Input	Input	Output	Output	Output
		slurry	coal	clinker	E.S.P.dust	Stack
Be	1 carcin	76	24	96	4	0
Cd	1	82	18	18	81	1
Tl	1	95	5	22	14	42
Ni	2	82	18	97	3	0
As	3	77	23	92	8	0
Pb	3	93	7	22	77	1
Zn	n.c.	93	7	88	12	0

Regarding the output of Hg, although not mentioned, this essentially leaves the kiln system completely via the stack gas. The next most volatile element was found to be Tl of which 42 % of the total input left the kiln in the stack gas whilst for Cd and Pb only 1 % left by this way.

Of the elements which did not leave the kiln system, with the exception of Tl these were mainly found in either the filter dust or in the clinker. Those less volatile elements found predominantly in the clinker were As (92 %), Be (96 %), Ni (97 %) and Zn (88 %). In contrast, for the more volatile species, 81 % of the Cd and 77 % of the Pb were found in the kiln dust.

5.6.2 Lepol Kiln

An example from Krcmar et.al. of a heavy metal balance for a Lepol kiln fired with petroleum coke as fuel is presented in Table 15. This presentation is more comprehensive than that previously given in that the actual concentrations of the individual elements are given for the various types of inputs and outputs. Those elements leaving the kiln in the form of a vapour were not measured and the fact that the output can be either higher or lower than the input is acknoweledged, these indicating a gain resp. loss within the kiln system. A missing output therefore does not automatically mean the emission of the element concerned as a vapour. This is certainly the case for Tl where additional measurements showed that the missing Tl was not being emitted.

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On the basis of the present balance the volatility of the classified heavy elements can be classified by the volatility factor f, defined as:

f = concentration in clinker / concentration in raw meal

The following classes have been defined

- volatile (f = 0 to 0.35): Cd, Tl, Se, Pb
- semi-volatile (f = 0.7 to 1.1):
- on-volatile (f = 1.2 to 2.7): As, Co, Ni, Cr, Cu, Sb, Zn

Table 15: Heavy Metal Balance for a Lepol Kiln Fired with Petroleum Coke (Concentrations marked * given in ppb)

TOXIC ELEMENTS IN KILN DUST

Element	Class	Input						Output							
		r.meal (ppm)	r.meal (%)	petcoke (ppm)	petcoke (%)	missing (ppm)	missing (%)	clinker (ppm)	clinker (%)	E.S.P. dust (ppm)	E.S.P. dust (%)	Stack (ppm)	Stack (%)	missing (ppm)	missing (%)
Cd	1	*367.8	98.80	*4.5	1.20			*83.9	22.50	*138.1	37.10	*8.2	2.20	*142.1	38.20
Tl	1	*567.4	99.20	*4.8	0.80			*53.3	9.30	*413.6	75.40	*29.1	5.10	*58.2	10.20
As	2	18.20	99.40	0.10	0.60			17.90	97.80	0.20	1.10	<<0.1	<0.1	0.20	1.10
Co	2	17.40	98.40	0.30	1.60			17.90	99.50	0.10	0.30	<<0.1	< 0.1	0.10	0.20
Ni	2	32.10	56.00	25.20	44.00			53.20	92.80	2.60	4.50	0.10	0.20	1.40	2.40
Se	2	2.80	100.00							< 0.1	0.40	< 0.1	0.40	2.80	99.20
Te	2														
Cr	3	45.60	98.90	0.50	1.10	0.20	0.40	45.80	99.30	0.50	1.10	<0.1	< 0.1	1.40	2.40
Cu	3	31.40	99.10	0.30	0.90	0.50	1.60	32.00	100.90	0.20	0.60	0.10	0.10		
Pb	3														
Pd	3														
Pt	3														
Rh	3														
Sb	3	1.40	95.20	<0.1	4.80	0.10	3.40	1.50	102.70	<0.1	0.70	<0.1	<0.1		
Sn	3	3.30	99.10	0.10	0.90			2.90	86.90	< 0.1	0.60	<< 0.1	0.10	0.40	12.50
V	3	76.70	47.50	84.90	52.50			135.80	84.00	18.40	11.40	0.90	0.60	6.50	4.00
Zn	n.c.	48.10	96.60	1.70	3.40	3.20	6.40	50.20	100.80	2.70	5.40	0.10	0.20		

The concentrations of toxic elements in cement kiln dusts have been measured for the wet, semi wet/semi dry and dry types of cement kiln and also for rotary lime kilns. The results obtained are listed in Table 16 in which it can be seen that no values are given for mercury (Hg) due to the fact that it is essentially wholly emitted and not retained in the dust or in the clinker. The values given refer to a period prior to the introduction of the German (TA Luft) and Swiss (LRV) Clean Air Acts. In some cases the plants were using secondary materials (combustible and non combustible) and also iron pyrites as a corrective. In those cases where the filter dust was sampled the values given in Table 16 are indicative of the range of these toxic elements and, as far as is known, are in no way associated with the burning conditions. It is not possible to adjust conditions to ensure that the toxic elements are steered into the clinker minerals and not into the dust. The values given for the stack gas dust are again indicative of the raw materials used but now for their interpretation the efficiency of the filter system must also be taken into consideration.

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Table 16: Concentration (in ppm) of Toxic Elements in Kiln Dusts

Class	Kiln	Wet	Semi	Dry	Lime
1	Cd	1-21	3-120	1-15	1-2
1	Tl	1-237	43-586	1-617	1-56-69
2	As	63-225	1-103	37-222	12-33
2	Co	1-2	1-11	1-169	41-765
2	Ni	11-75	20-87	22-4145	61-90
3	Cr	37-65	61-282	31-4107	
3	Cu	45-1180	128	44-69	27-77
3	Pb	31-2830	522-7000	12-321	
3	V	77-79	12-321	1-33	
n.c.	Zn	80-4500	27-77	107-147	

5.7 Class 1 Elements (Cd and Tl)

As seen in Table 16 the values for these elements can be as low as 1 ppm to as high as 120 ppm in the case of Cd and 617 ppm (0.06 %) for Tl. It is interesting that both the wet and dry kiln dusts had all Cd values essentially under 20 ppm whereas for the semi processes the lowest value was 31 ppm and the highest 120 ppm. When Tl is considered, which is mainly to be found in the dust and not in the clinker, no obvious trend could be seen but again the lowest value for semi processes was higher than for the wet and dry types. For the two lime kiln dusts studied, for which limestone was the only component of the kiln charge, the levels of Cd and Tl were both under 2 ppm, an indication that the limestone is usually low in toxic metals.

5.8 Class 2 Elements (As, Co and Ni)

Of the five Class 2 elements only As, Co and Ni were determined as the elements Se and Te are not often encountered. The absence of these elements should not be taken for granted as they are in the same column of the periodic table as S, and their absence should be first be confirmed by taking an XRF-spectrum. In one secondary fuel tested at HMC/TC the level of Se was found to be 12'000 ppm i.e. 1.2 %.

For As the values found in the dusts ranged essentially from 37 to 225 ppm, for Co from 1 to 169 ppm, with all values except one being under 37 ppm, and for Ni from 11 up to 4145 ppm (0.41 %) in an extreme case. For Ni the values were either under 200 ppm or well above with the extreme values all being for dry process kilns. With regard to the lime kiln dusts the maximum values were 69 ppm for As, 33 ppm for Co and 765 ppm for Ni.

5.9 Class 3 Elements (Cr, Cu, Pb, V)

The very exotic elements in this class (Pd, Pt and Rh) as well as Mn, which is always present in cement raw materials in small percentages, were not analysed. Up to date none of the very expensive elements have ever been found in XRF-spectra performed at HMC/CS.

Regarding the toxic metal Cr, whose presence is associated with chrome eczema, this has been found in quantities between 31 and 4107 ppm (0.41 %) in kiln dusts. Most values lie under 300 ppm and the appreciably higher values were found in only three cases. With regard to Cu most values lie between 40 to 70 ppm and only in two cases (128 ppm and 1180 ppm) were higher values determined.

The toxic element present in the highest levels in kiln dust was found to be Pb which ranged in value between 12 and 7000 ppm (0.7 %). For this element the level in the dusts of the

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semi process plants were definitely the highest and with only two exceptions the values lie between 1274 and 7000 ppm. For the dry process plants no values were higher than 321 ppm and for wet process kiln dusts most were lower than 150 ppm but with two extreme values of over 2220 ppm. The levels determined for V lie between 1 and 79 ppm and so are under the average level of this element (110 ppm) present in the Earth's crust.

5.10 Non-Classified Elements

Although Zn is not specified in the German and Swiss Clean Air Acts it can be found in kiln dusts. The levels found ranged mainly between 80 and 230 ppm with one extreme value of 4500 ppm (0.45 %) also being found. This highest value was determined on a stack gas dust.

5.11 Maximum Allowed Levels in Stack Gas Dust

Although the levels of the toxic elements present in the emitted stack gas dusts are high, their significance should be considered by taking not their concentrations but the quantities actually emitted into consideration. If it is considered that the maximum allowed emissions or Classes 1, 2 and 3 are 0.2, 1.0 and 5.0 mg/Nm^3 respectively, then for a stack gas stream with 50 mg/Nm^3dust load the maximum allowed concentrations in the dust are 0.4 %, 2.0 % and 10.0 %. In none of the cases investigated did the emissions come near to these values and even with filters of poorer efficiency is it unlikely that the limits would be exceeded.

6. TOXIC ELEMENTS IN CLINKER

6.1 Levels of Toxic Elements

The levels of toxic elements in clinkers, reported in the literature are given in Table 17. As can be seen for the elements of Class 1 i.e. Cd, Hg and Tl, the levels found were at the highest 1.5 ppm in the case of Cd. For the elements of Classes 2 and 3 the values reported, with the exception of Se, varied over a larger range. The lowest values could be low for As (2ppm), Pb (1 ppm) but could be much higher such as in the case of Zn (530 ppm), Ni (400 ppm) and Cr (320 ppm)

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Table 17: Range of Heavy Metals (ppm) in Clinker

Element	Class	Clinker	Earth's Crust
Cd	1	0.01 -1.5	0.2
Hg	1	< 0.01 -0.12	0.08
Tl	1	<0.01 -1.2	5
As	2	2 -87	1.8
Co	2	10 -21	25
Ni	2	10 -397	75
Se	2	<0.2 -<1.0	0.05
Te	2		0.01
Cr	3	10 -319	100
Cu	3	5 -136	55
Pb	3	1 -105	13
Sb	3	0.1 -1.5	0.2
Sn	3	(2.9)	2
V	3	10 -135	135
Zn	n.c.	29 -531	70

6.2 Influence of Toxic Metals on Cement Properties

As just seen in Table 17 the highest concentration of a toxic element found in clinker was 531 ppm (0.05 %) in the case of Zn and 397 ppm (0.04 %) for Ni. What effect will these have on cement properties?

From the studies of C. Tashiro the influence of additions of Cr, Cu, Zn, Cd, Hg and Pb, in both soluble and poorly soluble forms has been well documented. At an addition level of 0.5 % on a clinker basis no appreciable changes in hardening properties were detected and thus with the much lower levels of these metals,and then already incorporated within the clinker, definitely no change in cement properties is expected. The results of studies in which toxic metal dosages to cement of appreciably more than 0.5 % and which showed retarding and accelerating properties are not relevant to the properties of clinker. Statistical studies by Blaine on normally produced cements also showed that at the levels of the toxic metals present in clinker no changes are to be expected.

6.3 Ability of Clinker Compounds to Immobilize Toxic Metals

At the present time much research is being carried out into the use of cements to immobilize the toxic metals present in ashes, slags and other industrial waste materials so that they are not removed by leaching after dumping in an approved site. From the studies of Bhatty it is known that the toxic elements can be incorporated into the structure of the CSH-gel during its formation but can also enter the structure after its formation with the displacement of Ca ions. In certain cases new unknown compounds containing a toxic metal may also be formed.

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From the studies of the VDZ (Sprung) it is known that even the water soluble toxic elements will be completely immobilized on hydrating. This has been shown to be the case from studies in which larger quantities of toxic metals than are possible in clinker as found in wastes have been completely bonded by the use of cement pastes. The bonding is chemical and adsorbtive together with the formation of a dense structure.

From the data shown in Table 18 it can be seen that the solubility of the toxic metals in a cement suspension is between 10^{-1} and 10^{-6} % of the quantities present.

Table 19: **(1) Solubilities of Heavy metal Compounds at pH Values of 12.6 to 13.0: (2) Leaching Rates of Compacted Mortars (11 % Cement) after 28 Days Hardening**

Element	Solubility in $Ca(OH)_2$ / KOH Solution	Solubility in Cement Suspension (%)	Degree of Solubility (%) from Mortar
As	8×10^{-3}	4×10^{-5}	
Pb	93	7×10^{-3}	
Zn	1×10^{-4}	2×10^{-6}	
Cd	1×10^{-4}	$< 1 \times 10^{-6}$	2×10^{-4}
Tl	90	1×10^{-1}	1×10^{-4}

With regard to the leaching out of the elements from 28-day mortars prepared with 11 % by weight of cement, the quantity of the elements removed by the eluate was maximum 2×10^{-4} % of the quantity added. With regard to the small quantities normally present in clinker these will be certainly immobilized when the cement is present in hydrated form in concrete.

7. INFLUENCE OF KILN TYPE ON HEAVY METALS EMISSIONS

7.1 Experience from Literature and Plants

7.1.1 Non-Volatile Metals

Emission of low or non-volatile heavy metals are not critical with regard to actual emission limits. They can be kept under control by an efficient dedusting.

According to comparatively secured information [1, [18 the emissions of As, Be, Co, Cr, Cu, Mu, Mo, Ni, P, Pb, Sb, Se, Te, V, Zn are extremely low on suspension preheater kilns with efficient dedusting. As a general rule the emission of non- or low-volatile elements is always much below 0.1% of the corresponding metal input into the suspension preheater kiln.

Thallium has a different behavior. It is known to form an outer cycle (not inside kiln) which can create a high accumulation in the filter dust (a few thousand times) if no countermeasures are taken. This can also increase the normal solid emission or become a potential danger as at Lengerich.

On grate preheater kilns not only Tl but also additional elements as Cd, Pb and Zn can become highly enriched in the filter dust. The emissions of all these elements are still mainly in solid form and therefore related to the efficiency of the filter. Due to the enrichment effects the emission may reach the order of 1% (instead of 0.1% as before).

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All elements mentioned so far and under the conditions of suspension or grate preheater kilns are known to have a very low or insignificant vaporous proportion (equilibrium concentration). This is certainly applicable for exhaust gas temperatures <150°C. It should be kept in mind that the volatility depends not only on the temperature but also on the other accompanying components occurring on cement kilns (e.g. Cl).

7.1.2 Volatile Metals

Volatile elements cannot be controlled by efficient dust filters and are also classified as most toxic (class 1). Therefore, they need special attention.

The most prominent example is the mercury. Hg is suspected to be almost totally emitted [18, 24, 25]. At the same time the emission limit is only 0.2 mg/m^3, therefore, it may soon become critical if the Hg input is significant. Hg emissions on cement kilns are almost completely in form of vaporous Hg. Therefore, many older emission measurements (based on pure solid sampling) either omitted Hg or did show unrealistically low figures.

Although Hg is very volatile, a certain retention effect in the cement kiln system is also possible. This may e.g. be the case under the influence of some sulphur compounds or in a raw mill during compound operation [7]. However, the retention during mill operation creates a Hg accumulation which can evaporate when switching to direct operation thus creating a high peak emission and a less favorable long term retention effect. This was e.g. observed in [18] when some 0.2 mg/m^3 were found during direct operation. However, this high emission value seems to be very exceptional in general because normal raw materials and fuels do not create critical Hg inputs. The problem of the highly volatile Hg is therefore limited to very special cases and is not a general problem under the present emission limits.

It is not quite clear whether also other elements could become as volatile as Hg under certain conditions. According to [14] Selenium on grate preheater kilns would be highly volatile but this is not confirmed.

7.1.3 Long Wet and Long Dry Kilns

Unlike suspension and grate preheater kilns long wet and long dry kilns are less explored in their heavy metal emission behavior. From the basic principle and from the higher stack gas temperatures (high vaporous equilibrium concentrations) one could expect a less favorable situation but the practical evidence is incomplete.

The most complete and extensive information basis so far are heavy metal balances from the wet kiln at Obourg [16]. The results are partly surprisingly favorable. The emissions for all heavy metals investigated such as Be, Cd, Hg, Tl, As, Co, Ni, Se, Cr, Cu, Mn, Pb, Sn, V, Zn are very low and the calculated relative emissions are below 0.5% even for Cd, Tl, Hg. In other words: none of the heavy metals seems to be volatile. The elements of class 1 and 2 are typically < 0.01 mg/m^3 and class 3 elements are < 0.1 mg/m^3.

7.2 Specification of Missing or Inaccurate Data

When reviewing the corresponding literature one is often impressed by the very low heavy metal emission or the excellent metal retention effect found for certain elements. When going more into details it becomes evident that such impressive results are usually based on pure solid sampling on the clean gas. The vaporous proportions are also verified in some cases but the measuring methods on cement kiln exhaust gases do not allow for extremely low detection limits (< 1 mg/m^3) under practical conditions (method in [20] is not accepted for practical emission measurements).

Measuring methods for vaporous proportions of metals also appear as possible weak point in some cases. Measurements do often consider only solid sampling and the assumption is made that vaporous proportions are negligible. The analytical confirmation is often missing

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due to the lack of sufficiently sensitive measuring methods. And if the dedusting performs very well, very low emissions and impressive retention effects for heavy metals of 99.999% or even better can be calculated. Although there seems to be no problem with respecting emission limits there could be a problem with the impressive number of nines in the retention effect. In view of vision towards a future "zero emission" sampling methods and analytical procedures need to be improved.

8. STANDARD TECHNOLOGY FOR REDUCTION OF HEAVY METAL EMISSION

8.1 Reduction of Heavy Metals Input into Kiln System

The input of heavy metals into the kiln system from raw materials and fuels can be controlled and limited in order to avoid undesired effects. The justification can be:

- Limits set for heavy metals in clinker (usually "political" limits, concentrations are often too low to produce clear negative effects in clinker, except Cr)
- Cycles of semi-volatile heavy metals and their influence on (CKD) kiln dust or bypass dust
- Emission of volatile heavy metals (e.g. Hg)

With regard to emissions, it is most important that the heavy metal input is limited, especially if conditions are present which promote emissions such as:

- high stack gas temperature
- low metal retention capacity of kiln system
- presence of compounds which can volatilize the metal, such as chlorine

For example, if sewage sludge is burnt in a cement kiln the measuring input has to be limited [18, 19].

The emission of a metal only becomes important if the metal cycle can actually reach the filter and if there is a problem with the function of the filter (EP shut down or poor performance), e.g.Thallium cycle on SP kiln

Besides these special cases most of the known heavy metals behave as low volatile and are virtually not emitted. The question is rather which metal concentrations in clinker and kiln dusts are acceptable from a political and environmental point of view. Therefore, input reduction will always maintain its importance, even if metal emission values are very low.

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8.2 Modification of Process

Break the Cycle of Enrichments

If semi-volatile metals as e.g. Tl cannot be avoided in the raw mix there is a possibility to break the cycle of Tl enrichment. This will then limit the maximum concentration in the filter dust and thus reduce the potential risk during any malfunction of the filter.

The principle is often used on systems which originally worked with complete reintroduction of the kiln dust (closed systems), as e.g. suspension preheater kilns or grate preheater kilns without any bypass. Enriched filter dust is extracted from the system and usually introduced into the cement grinding process, e.g. on suspension preheater kilns in combination with a raw mill the kiln dust during direct operation can be extracted. The costs are very moderate (< 0.2 Mio CHF). The effects in terms of normal emission is not the main argument, it is rather a reduction of a potential risk.

8.3 Secondary Reduction Measures

Improve Dedusting

In case of non or semi-volatile heavy metals the emission can be reduced by improving or replacing the dedusting. The effect of an improved emission is an almost linear decrease of the heavy metal emission according to the reduction of the dust emission. However, the heavy metals are usually not the main justification for such modifications.

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9. BAT (BEST AVAILABLE TECHNOLOGY) FOR REDUCTION OF HEAVY METAL EMISSION

9.1 Non-volatile metals

For most of the heavy metals which are non-volatile or semi-volatile the emission reduction can be achieved by reducing the dust emission (see paper PT 95/4391/E, dust).

9.2 Volatile metals

Special technology in form of secondary reduction measures is only justified for volatile metals as e.g. Hg. The following technologies are possible:

1) Dry scrubbing by circulating fluidized bed is also effective for Hg reduction [26. The use of other absorbers (activated coke) is also possible and would improve the effect.
2) Activated coke filter The Polvitec at "HCB Siggenthal removes Hg at an efficiency of some 99%. As a pure Hg removal system it would be extremely expensive for this particular purpose (Investment of approx. 30 Mio CHF). Since the activated coke is re-used as kiln fuel the mercury would form a cycle. The cycle is broken up by extracting filter dust during direct operation and using it in cement grinding. Treatment of the dust (thermal Hg-expulsion) has been tested but not realized on a large scale due to low Hg concentrations.
3) Wet scrubbing This is not yet known on cement kilns but rather on waste incinerators. Wet scrubbing of mercury is inefficient if Hg is present in metallic form (insoluble) and may not be automatically the appropriate method [29. Oxidyzing the Hg would be necessary. Again, this method would be extremely expensive only for the purpose to remove volatile heavy metals.
4) Removal of critical heavy metal from specific fuels or material components

   Example: Hg can be thermally expelled from sewage sludge before feeding it into a cement kiln [27].

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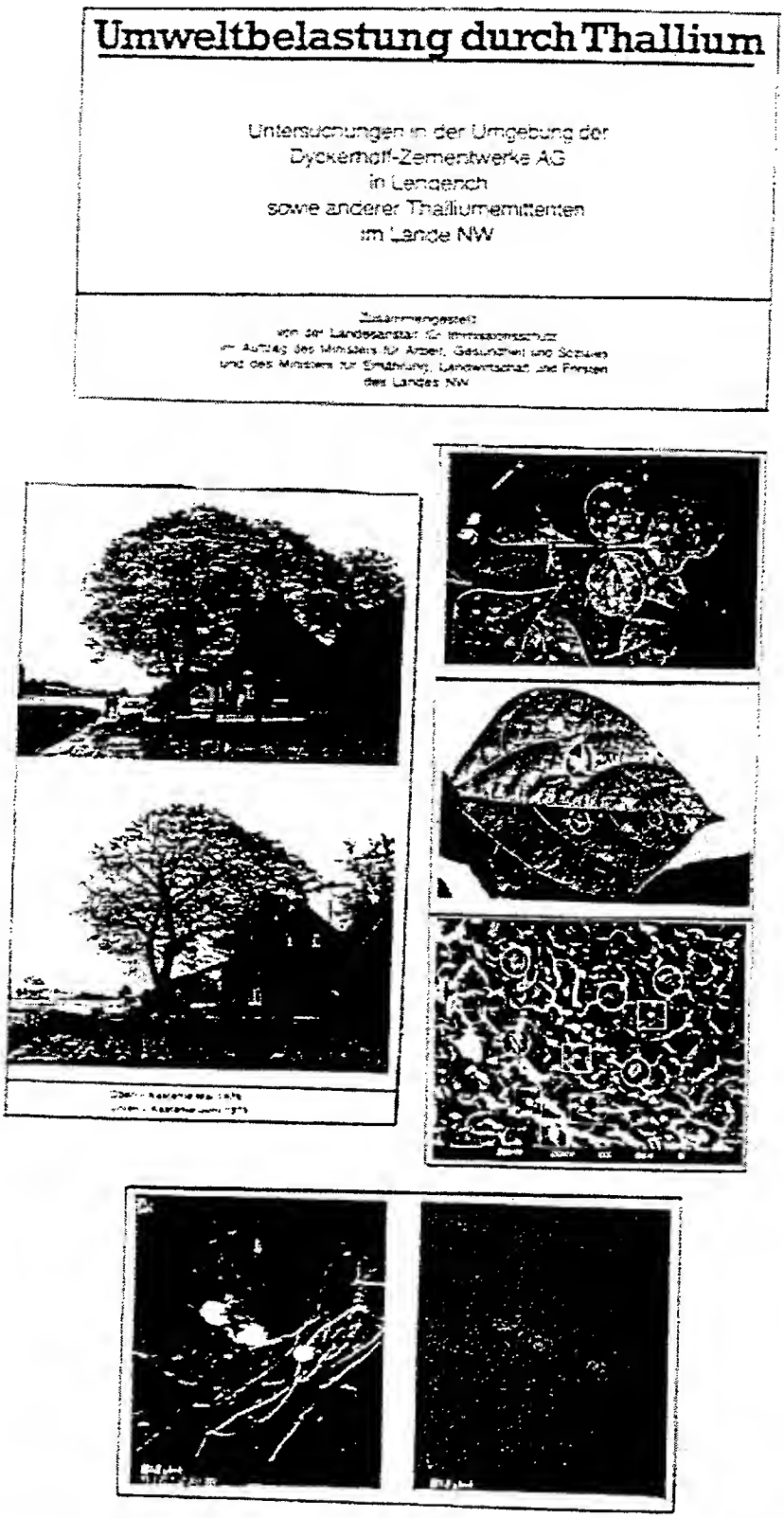
11. APPENDIX

11.1 Thalium Documentation Dyckerhoff-Zementwerke AG, Lengerich



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Characteristics of Toxic Elements quoted in TA Luft and LRV

Element	Class	Critical Dose		Uses	Source	Natural concentrations
		WHO max. dose	Lethal dose			
Cd	I	0.5 mg / week		Coating for metals (hot dip) Alloying material (wires, bearings, batteries) Pigment (CaS in glass, paints, etc.) Insecticide (CdO, $Cd(OH)_2$)	Zn / Pb Ores	1 ppm in plant and animal tissues
Hg	I			Scientific instruments Mercury vapour lamps Solders Mercury arc rectifiers Anti-fouling paints Pharmaceuticals Seed dressing	HgS (Cinnabar)	Soils, dust, water, 0.005 - 0.25 ppm in nearly all foods
Tl	I		100 ppm poss. toxic to soils 12 mg/kg body weight	Rodenticide / insecticide (Thallous sulphate) Lenses, IR instruments Alloying material Fireworks Dyes, pigments Low temperature thermometer (with Hg) Catalyst Hair remover	Pyrites	Not naturally found in plants or soils
Be	I (cancer.)			Cu / Be alloys X-Rax tubes Fluorescence light tubes Nuclear reactor parts	Beryl ore	
As	II	MAC of 0.5 mg/m^3		Insecticides, Weed killers, fungicide (as K-arsenate), Wood preservative Glass Alloying material Therapeutic (organic As compounds)	By-product Cu, Pb, Zn, Sn, Au-ores	In all soils from <10 to 500 ppm
Co	II		10-20 mg per 100 g body weight	Alloying material Magnet alloys Cutting tools Binder for tungsten carbide Blue pigment in china glass	In Cu / Co ores in Zaire	
Ni	II	Ni and soluble Ni-salts are not considered as being toxic to humans. The most important industrial aspect is their trend to cause dermatitis.		Alloying material Special steels Electroplating Catalyst for hydrogenation of oils Coinage Paints and enamels Ni-Cd batteries	S-ores in Ontario	0,0016% of the Earth's crust and is widely found in soils and plants
Se	II			Se rectifiers Anti-corrosive coating on copper, steel etc. Decolouriser in glasses Paints and dyes Photoelectric cells Insecticides Catalyst	Anode slime of Cu-refineries	II soils from trace to 30 ppm, at which level animals suffer from alkali disease
Te	II			In cast iron to improve properties In various Pb, Cu and Fe alloys In rubber industry As catalyst Glass colouring agent In bacteriological tests In thermocouples for low temperature	Bi-ore residues	Traces in seleniferous soils but not in crops

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Element	Class	Critical Dose		Uses	Source	Natural concentrations
		WHO max. dose	Lethal dose			
Sb	II			Alloying material Battery grids Pewter Printers type Lead electrodes Bearing metals Rubber compounding Flameproof clothing Paint pigment In ceramic / glass	Sb_2S_3 (Stibnite)	
Pb	III			Pipes, Roof coverings Tanks for acids Lead shot, Bullets, Linotype metal Battery plates Alloying material In paints and pigments In glasses and enamels Litharge rubber Insecticides Noise barriers	PbS (Galena)	In soils and plants in very low levels. Clay soils make Pb insoluble but acid soils increase solubility and toxicity.
Cr	III			Alloying material in steels Chromium plating Cutting tools (as carbide) Paint primer (zinc chromate) Leather tanning agent Wood preservative (zinc, chloride, chromate) Pigment (paint, inks, dyes, etc.) In photo industry as sensitiser	$FeO.Cr_2O_3$ (Chromite)	In small quantities in all plants and soils. Plants grown on soils with 0.2-0.4% are infertile
CN	III					
F	III					
Cu	III		780 mg as $CuSO_4$	Alloying material As electrical conductor Gas / water pipes, Roof sheeting Bronze paint Anti-fouling paint Insecticide	As S-ores in several areas	
Mn	III			Alloying material Potassium permanganate Colour in glasses / ceramics Paints, inks Matches, Fireworks Enrich manganese deficient soils	As oxide sulphide and carbonate	up to 20 ppm in some nuts
Pd	III			Alloying material Caralyst Dental alloys Jewellery	In certain coals as aresenide or selenide	
Pt	III			Contact material in electrical circuits Electrode material Catalysts High temperature electric furnaces Dies for glass fibre production	As native metal in Cu / Ni ores	
SiO_2	III					
Rh	III			Pt / Rh alloys Electroplating Electrical contacts surfaces Reflector in high power lamps	In wastes from Cu, Ni and Pt extraction	

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Element	Class	Critical Dose		Uses	Source	Natural concentrations
		WHO max. dose	Lethal dose			
V	III			In special steels and alloys Industrial catalyst Pigment in paints, dyes, inks In insecticides In photography	Oil refinery residues (Carnotite)	To 470 ppm in soil; to 10 ppm in plants
Sn	III					

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Chapter 7

Kiln Dust

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Disposal of Wastes from Cement Production

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1. INTRODUCTION

Whereas natural processes generally occur in cycles, in which the waste products created do not harm the biosphere, the same cannot be said for human activities at today's high population densities. The cement industry is no exception in that it generates wastes whose properties range from being an unsightly nuisance to being definitely toxic. Table 1 lists these principal solid, liquid and gaseous wastes. Whereas the gaseous wastes go into the atmosphere and rapidly lose their identity by dilution, the same is not true for solid types. The latter remain in possession of the plant, which has the responsibility for their safe disposal

Table 1: Wastes generated in Cement Plants

Solid Wastes	♦ overburden, earth
	♦ kiln dust, bypass dust
	♦ mill dust
	♦ sulphur absorbent
	♦ reject production
	♦ used refractories
	♦ cleaning dust
	♦ scrap iron
	♦ rubber wastes (tyres, belts)
	♦ test prisms / cubes
Liquid Wastes	♦ rainwater / wash water
	♦ sewage water
	♦ oils / greases / emulsions
	♦ laboratory chemicals
	♦ solvents
Gaseous Wastes	♦ CO_2, NO_x, SO_2, Hydrocarbons, etc.

In the present article the main emphasis will be placed on the management of the large tonnage wastes namely refractory tear out and the various types of kiln dusts. No mention will be given to that waste generated in the largest quantities, namely CO_2 emissions, of which the cement industry is a significant but not the major contributor.

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2. REGULATORY ASPECTS OF DUMPING SOLID WASTE MATERIALS

In the past most manufacturing and processing industries simply dumped their waste materials, euphemistically calling them "landfill". They did this because it made economic sense and an awareness of the accompanying burden to the environment existed in only rare cases, if at all.

Today the situation has changed. Environmental awareness is only all too present and those harmlessly dumped wastes of yester year are now becoming the potential environmental time-bombs of today and tomorrow. Many a cement plant is going to be confronted with a bill for cleaning up its dumped materials, costs which will certainly be a multiple of the sum originally "saved". No cement company in a developed country is today going to be allowed to wind up its operations until it has made certain that all the materials it dumped during its operational lifetime are in a form which poses no threat to the environment.

Some examples of the regulations applicable to the dumping of materials will now be given.

2.1 Swiss TVA 95

The Swiss TVA regulation of December 1990 is a typical example applicable to the dumping of all types of material. It defines three categories of dumping operations, namely of the 'inert material", "residual material" and of the "reactor" types. Details of the requirements for end storage in inert and residual dumps are given in Tables 2, 3 and 4. The use of reactor dumps is presently reserved for slag from rubbish incinerating plants and for sewage sludge (< 65 % H2O), inert/building waste materials, etc. when no capacity is available in inert or residual material dumps. Dumps for highly toxic materials do not exist and such materials have to be eliminated by other means.

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Table 2: Swiss Dumping Regulations (TVA-1990) Compositional Requirements

MT-Mat. No / Material / Condition	TVA Requirements for End Storage	
	Inert	Residual
Composition known (%)		> 95
Stone like Material (%)	> 95	
H_2O-soluble substances (%)	0,5	5
Organic material (% C)		5
Pb (ppm)	500	
Cd (ppm)	10	
Cu (ppm)	500	
Ni (ppm)	500	
Hg (ppm)	2	
Zn (ppm)	1000	
Lipo. org. Cl compods. (ppm)		10
Gas formation with H_2O		none
Reaction with self, H_2O,air		none
Neutral, Capacity (Mol/kg)		1

The following building material can be included in an inert substance deposit: concrete, tiles, asbestos cement, class masonry and road construction wastes.

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Table 3: Elutriation Test (No. 1)
Requirements in Swiss Dumping Regulations (TVA 1990)
(Elutriation in CO_2 Saturated Solution)

MT-Mat. No / Material / Condition	TVA Requirements (mg/l) for End Storage	
	Inert	Residual
Al	1.000	10.00
As	0.001	0.10
Ba	0.50	5.00
Pb	0.10	1.00
Cd	0.01	0.10
Cr-III	0.05	2.00
Cr-VI	0.01	0.10
Co	0.05	0.05
Cu	0.20	0.05
Ni	0.20	2.00
Hg	0.005	0.01
Zn	0.50	5.00
Sn	0.20	2.00

Materials which may be deposited in inert substance dumps can either be unreactive substances as defined in the mentioned tables or "building wastes" such as concrete, tiles, asbestos cement products, glass and masonry/road construction wastes. Materials such as metal, paper, wood, and textiles must be removed from "building wastes" as much as is economically practical.

As can be seen in the Tables 4 to 6 there are limits set down on the chemical composition (toxic metals, organic substances) of the materials themselves and on their leaching behaviour in two elutriation tests. In the first test CO_2 is bubbled through the solution to keep the pH value low and thus ensure that the toxic cations are brought into solution and not precipitated. In the second elutriation test, in which the anions and organic substances present are determined, distilled water without CO_2 is employed.

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Table 4: Elutriation Test (No. 2)
Requirements in Swiss Dumping Regulations (TVA 1990)
(Elutriation in Distilled Water)

MT-Mat. No Material	TVA Requirements (mg/l) for End Storage	
Condition	Inert	Residual
NH^{4+} / NH_3	0.50	5.00
Tl	not specified	not specified
Cyanide	0.01	0.10
Fluoride	1.00	10.00
Nitrite	0.10	1.00
Sulphite	0.10	1.00
Sulphide	0.01	0.10
Phosphate	1.00	10.00
DOC *	20.00	50.00
HC's	0.50	5.00
LOC *	0.01	0.05
Cl Sol *	0.01	0.10
pH Value	6 - 12	6 - 12

DOC: dissolved organic carbon

HC: hydrocarbons

LOC: lypophilic organic chlorine compounds

Cl Sol: chlorinated solvents

As a result of these regulations the following restrictions prevent (in Switzerland) the dumping of waste materials such as kiln dust and refractories:

- H_2O soluble substances < 0.5 %, resp < 5.0 % exceeded due to presence of soluble alkali chlorides and sulphates
- total Pb and Zn levels over 500 ppm due to use of alternative fuels and raw material components
- soluble Pb and Zn limits exceeded due to use of alternative fuels and raw material components.
- soluble Cr^{6+} limit exceeded in the case of spent magnesia-chrome refractories

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2.2 E.P.A.'s TCLP Leaching Test

In the USA the disposal of kiln dust and of spent kiln refractories is mainly by dumping the refractories, often in plant external sites. This disposal is regulated by the Resources Conservation and Recovery Act (RCRA) and hence the class of dump to which they can be sent, is decided by their behaviour according to the Toxicity Concentration Leaching Procedure (TCLP) test. Details of the latter and the limits defined for the various elements in the leachate are given in Table 5.

Table 5: Details of USEPA's TCLP Leaching Test

Leaching Medium	Demineralised H_2O
	+ acetic acid (pH=5)
Liquid / Solid ratio	20 : 1
Max. Particle Size (mm)	9.5
Nr. of Extractions	1 (agitation)
Extraction Time (hr)	18 ± 2

Element	Extract Limit (mg/l)
Ag	5
As	5
B	
Ba	100
Be	0.007
Cd	1
Cr	5
Cu	
CrHg	0.2
Ni	70
Pb	5
Se	1
Sb	1
Tl	7
Zn	

As can be seen the elements defined are (with one exception) trace toxic elements and as such only present in natural raw materials and in new, unused refractories in trace quantities, if at all. Trace quantities can, however, be picked up during their presence in the kiln. The main oxides from which kiln refractories are produced i.e. SiO_2, Al_2O_3, CaO, MgO and the trace oxides Fe_2O_3, K_2O, Na_2O etc. are not stipulated and are anyway the same oxides as found in PC clinker.

The one exception is Cr which, as Cr_2O_3, is a major oxide component of magnesia chrome refractories, with some refractories containing up to almost 18 %. Although it is wholly present as insoluble Cr_2O_3 in the new, unused bricks this is partially converted to a soluble alkali sulphate chromate during kiln operations and therefore contains soluble Cr^{6+}. On the basis of tests carried out at HMC Corporate Services up to 0.4 % of soluble Cr^{6+} has been determined in spent magnesia chrome bricks which originally contained 17 % Cr_2O_3.

Spent magnesia-chrome bricks are therefore not likely to pass the TCLP test for disposal in a municipal garbage dump and hence will have to be treated before such a disposal with ensuing costs. A change to refractories of the chrome-ore free magnesia-spinel or doloma types avoids this disposal problem completely.

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2.3 European DIN 38 405 D24 Leaching Test

According to this regulation the following concentrations of Cr^{6+} dictate the disposal of all wastes, but especially those containing chromite ore (magnesia chrome bricks) as follows:

- < 0.1mg/l: in household garbage dump
- > 0.1 mg/l < 0.5 mg/l: in toxic material dump
- >0.5 mg/l: special treatment or must be dumped underground

3. DISPOSAL OF GENERAL CEMENT PLANT WASTES

Of the general wastes generated in cement plants examples of their management at the Rekingen and Ciminas plants are listed in Tables 6 and 7. Whereas the procedures for dealing with wastes are already established in developed countries, plants in developing countries invariably have to set up their own infrastructure. It is important that clearly marked collection centres are located near to the places the wastes are generated and that their use is promoted.

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Table 6: Waste Concept at Rekingen

Material	Quantity per Month	Internal Disposal	External Disposal
Office, stores			
Paper carton		Kiln	
Workshop			
Ferrous	4t		Scap metal
Non-ferrous	2t		Scap metal
Rags			Garbage incinerator
Waste Wood Plant			
Ferrous	40t		Scrap metal
Non-ferrous	3t		Scrap metal
Masonry	60t	Mix bed	
Paper, styropor	0.5t		Garbage incinerator
Water		Mix bed	
Refractories			
Cuttings	10	Mix bed	
Tear out	500 t/a	Mid bed	
Garage			
Tyres	0.5t	Siggenthal	
Oil / Air filters	50 kg	Siggenthal	
Waste Oil	1000l		Waste oil dealer
Laboratory			
Liquid waste	10l		Special collection
Solid waste	10l		Special collection
Radioactive source			Supplier
Various			
Glass			Local collection
Light tubes	100		Special Collectio

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Table 7: Waste to be Disposed at Ciminas Plant

Material	Means of Disposal
	Internal & External use
Absorbed Diesel Spillage	Used as combustable
Filter bags	Reused, sold for reuse
	Internal use
Asbestos components	Crushed, added to raw mill
Cement ex spillage's	Returned ro process
Clinical wastes	Incinerated in containers
Glass	Crushed, added to raw mill
Lamps	Added to raw mill
Rags / cloths	Incinerated in containers
Refractories (Cr containing)	Crushed, added to raw mill
Wastes from charcoal	Added to raw mill via clay
Sax ex sand blasting	Added to raw mill
	External use
Askarel oil	Incinerated externally
Clothes / shoes / helmets	Sold for recycling
Concrete cubes	Given to personnel
Fibre glass	Sold for recycling
Filter elements	Pressed and sold
Iron ex slag	Sold to foundries
Metal / electrical scrap	Sold for recycling
Oils / greases	Sold for recycling
Paper ex office	Sold for recycling
Plastics	Sold for recycling
Refractories	Sold for reuse
Rubber / car batteries	Sold for recycling

It is the disposal of the larger quantities of cement plant specific wastes kiln dust (up to 300'000 tpa at one Group plant) and spent refractories (up 800 tons at some Group plants) that must receive the greatest attention. Whereas refractories are consumable items and it is understandable that there will be remnants to be disposed of, this is not the case with kiln dusts. In fact most Group plants do not have to discard dust and those actually doing so are forced to do so for several reasons:

- production in wet kilns of low(er) alkali cements from raw materials which do not have sufficiently low alkali levels to produce such cements without the discarding of dust
- use of a bypass in dry process kilns when the raw materials have a too high Cl content
- to prevent the formation of rings in wet process kilns caused by a molar excess of SO_3 over alkalis.

Most of the dust discarded in the Group is generated at wet plants in the USA and Canada for the production of low alkali cements and thus in countries whose standard specifications still do not allow the incorporation of kiln dust into cement.

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4. DISPOSAL OF KILN DUSTS

4.1 Discarded Dust, its Potential and Drawbacks

The total quantity of dust discarded annually in the "Holderbank" Group amounts to some 1.8 mio. tpa, with the quantity per plant ranging from zero (most cases) to over 200'000 tpa at a few N. American plants. In these latter cases the kiln feed / clinker factor can be as high as 1.91 which means having to process excessive quantities of kiln feed and a faster depletion of the reserves.

The chemical composition of the discarded kiln and bypass dust varies with kiln type as well as from from plant to plant. Details main oxide composition of kiln dust discarded at Group Plants is given in Table 8, with details of the heavy metal levels in American kiln dusts being listed in Table 9.

Table 8: Analyses of Dusts from Various Kiln Types

Kiln type	Suspension preheater				Grate preheater		Wet long		Dry long	
	Filter		By-pass		(Lepol)					
Abbreviation	DS-F		DS-BP		DG		WL		DL	
Number of samples	19		6		13		36		2	
	from-to	average	from-to	average	from-to	average	from-to	average	from-to	average
L.o.I	31.3-37.9	34.5	4.3-13.3	8.9	5.2-28.6	15.6	3.8-33.8	22.6	21.8-27.0	24.4
SiO_2	9.0-16.8	13.1	13.1-16.4	14.4	4.9-21.0	14.0	3.5-18.3	13.3	7.6-11.3	9.5
Al_2O_3	2.2-6.6	4.1	3.0-4.5	3.7	1.1-9.0	3.7	1.4-6.9	3.9	1.6-3.1	2.9
Fe_2O_3	1.2-3.3	2.2	1.2-2.5	1.7	0.4-3.7	1.5	0.6-4.3	1.9	1.1-1.7	1.4
CaO	38.6-45.7	42.2	44.4-60.8	52.3	14.9-52.7	36.1	10.6-53.8	41.7	31.7-49.6	40.2
MgO	0.70-2.4	1.4	0.80-2.7	1.7	0.4-2.6	1.0	0.3-4.5	1.3	1.3-1.3	1.3
SO_3	0.04-2.4	0.75	2.6-4.8	3.6	1.3-32.2	121.1	0.68-36.7	6.3	6.0-14.8	10.4
K_2O	0.35-2.7	1.07	4.1-12.6	8.8	3.4-35.2	13.2	0.79-38.1	6.0	0.88-15.8	8.3
Na_2O	0.07-0.30	0.17	0.17-1.6	0.62	0.18-2.6	1.1	0.12-14.1	1.3	0.07-1.3	0.69
Cl	0.04-0.68	0.25	2.3-7.1	4.2	0.10-2.0	0.81	0.07-10.8	1.3	0.88-1.3	0.69
Free lime	0.00-0.54	0.12	2.6-31.1	4.2	0.71-7.7	6.32	0-17.9	6.33	1.3-8.8	5.1

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Table 9: Total (acid soluble) metals in cement and cement kiln dust

Volatile		Total in Cement, mg/kg				Total in CKD, mg/kg				Ratio of Avg.
		Min.	Avg.	Max.	No.	Min.	Avg.	Max.	No.	Cement /CKD
▲	Mercury	<0.001	0.014	0.039	21	0.004	0.660	25.500	68	2%
	Selenium	0.62	NM	2.23	6	2.68	28.14	307.00	54	NM
	Thallium	0.01	1.08	2.68	32	1.40	43.24	776.00	89	2%
	Cadmium	0.03	0.34	1.12	42	0.1	10.3	59.6	81	3%
	Lead	1	12	75	82	34	434	7390	95	3%
	Antimony	0.7	NM	4.0	2	0.3	NM	3.4	9	NM
	Silver	6.75	9.20	19.90	93	4.8	10.53	40.70	92	87%
	Arsenic	5	19	71	54	2	18	159	50	106%
	Nickel	10	31	129	85	1	22	60	65	141%
	Barium	91	280	1402	94	35	172	767	95	163%
	Berrylium	0.32	1.13	305	93	0.13	0.65	3.54	94	174%
▼	Chromium	25	76	422	94	8	41	293	95	185%
Refractory										

Notes:

1) Exclude non-detectable values
2) NM indicates average *not meaningful* because of large number of non-detectable values
3) No. indicates number of detected values

Kiln dusts, although discarded by a few plants can, in specific cases, possess characteristics that make them of interest to other uses. The most important properties are:

- low price
- the very fine particle size / high absorptive capacity
- neutralising properties due to the presence of $CaCO_3$ and free CaO
- enhanced levels of K_2O and SO_3
- content of hydraulically active components

These properties have lead to proposals being made for using cement kiln dust in the following applications:

- absorber material
- filler component in cement and asphalt
- neutralizating agent
- fertilizer/agricultural lime
- stabilisation of mud from oil drilling
- substitute for lime in
  - sewage sludge dewatering
  - waste stabilization
  - exhaust gas scrubbing

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- glass manufacture
- rearing cattle
- production of soil replacement
- production of low grade binder
- as cat litter

Here it must be emphatically stated, that only in exceptional cases has cement kiln dust actually been used for any of the above applications in large quanties and over many years. No universally acceptable uses have yet been located for kiln dusts outside of the cement plant and their disposal problem has yet to be solved.

4.2 Methods of In-Plant Disposal of Untreated Kiln Dust

4.2.1 Addition of Dust to Cement

The most cost-effective method of using kiln dust is by incorporating it into cement. This practice was allowed early on in a few national standard specifications in European countries, which in 1989 resulted in 7 Group plants employing kiln dust in this manner. The quantity involved at single plants varied from a minimum of 0.4 wt.% to a maximum value of 8.5 wt.% of the OPC produced, with a total of 170'000 t being used in this way. In Germany kiln dust could also be added to blast furnace slag cements, where the addition resulted in improved early strength characteristics.

This use of kiln dust has since been incorporated into the European cement standard (ENV 197-1) which, as seen in Table 10,allows the addition of to 5 % of "minor additional constituents" which may include kiln dust. As can be seen this addition is allowed in all cement classes.

Table 10: ENV 197-1 Cement Types and their Compositions (wt.%)

Cement Type	Clinker (K)	B.F. Slag (S)	Pozzolana Natural (Zn)	Possolana Indust. (Zi)	Fly Ash Siliceous (Cs)	Fly Ash Calcareous (Cc)	Burnt Shale (B)	Limestone (L)	Silica Fume (D)	Minor Additions (F)
I	95-100	-	-		-		-	-	-	0-5
II/A	80-100	S+Z+C+B+L+D+F = 6 - 20								-
II/B	65-79	S+Z+C+B+L+D+F =21 - 35								-
III/A	35-64	35-65	-		-		-	-	-	0-5
III/B	20-34	60-80	-		-		-	-	-	0-5
III/C	5-19	81-95	-		-		-	-	-	0-5
IV	60-90	-	0-40			-	-	-	0-10	0-5
V/A	40-64	18-30	18-30			-	-	-	-	0-5
V/B	20-39	30-50	30-50			-	-	-	-	0-5

1) Values in the table refer to cement nucleus, excluding calcium sulphate and additives
2) Minor additional constituents may be filler or may be one or more of the main constituents unless inclueded as main constituents
3) Portion of silica dust is limited to 10% in types III/A and III/B
4) Portio of filler is limited to 10% in types III/A and III/B

For those countries in which the local standard specifications do not allow the addition of modest additions of kiln dust as in ENV 197-1, cement companies should, by active participation in standards committees, attempt to bring about the necessary changes in the national standard.

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4.2.2 Returning to Kiln after Granulation or Compaction

Essentially this method prepares kiln dust for re-entry into the kiln by simply increasing the particle size, either by granulation (1-2 mm) or by briquetting it into long thin rods.

As far as is known the granulation method only found one practical application, on kiln 8 at Obourg, but was not needed for the larger kilns 9 and 10 as the introduction of standard specifications, which allowed the addition of dust to cement, made its use irrevelant. The layout of the kiln 8 granulator (Schugi) is shown in Figures 1 and 2.

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Figure 1: Granulater for production of fine granulated powder (1 - 2 mm)

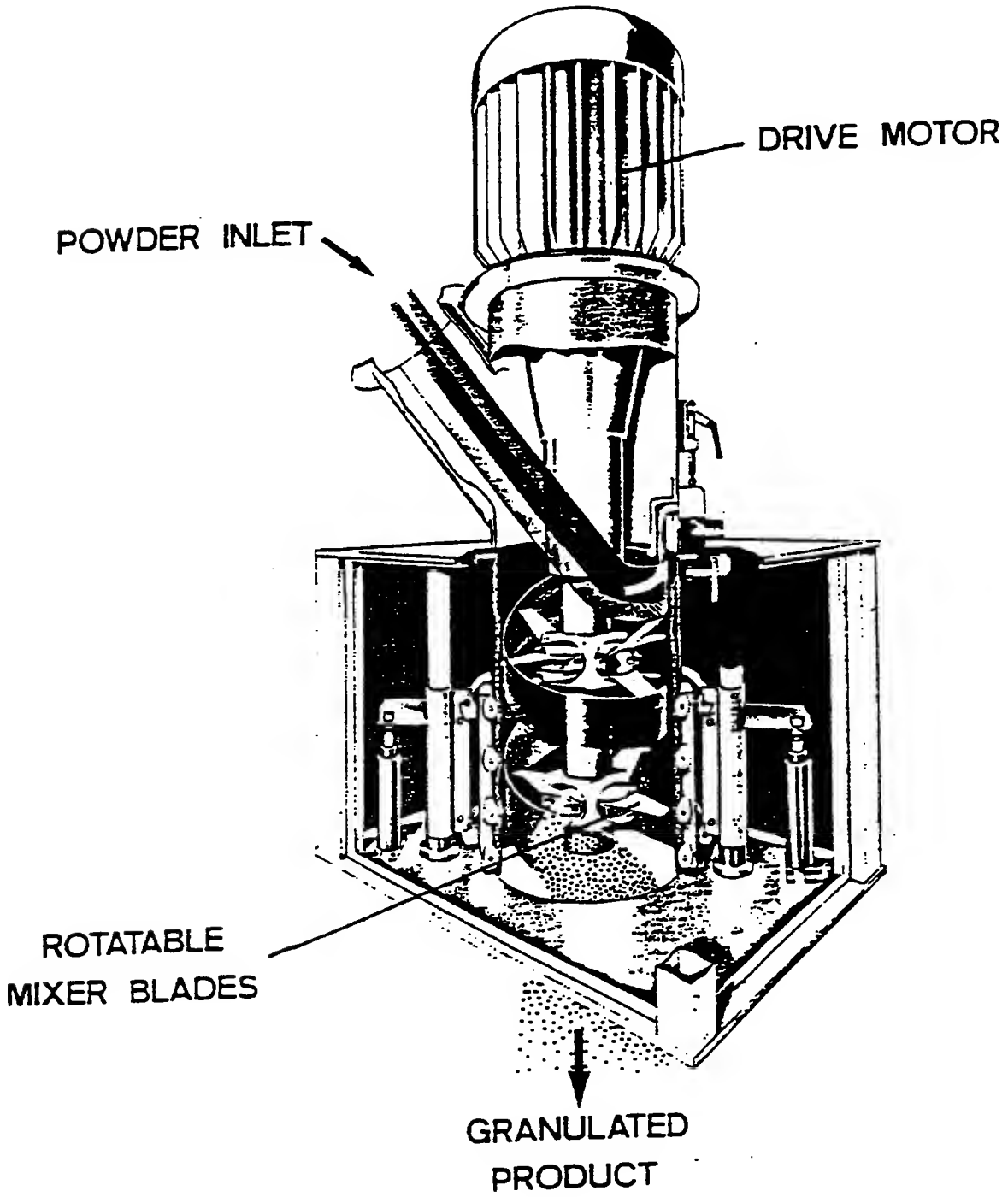

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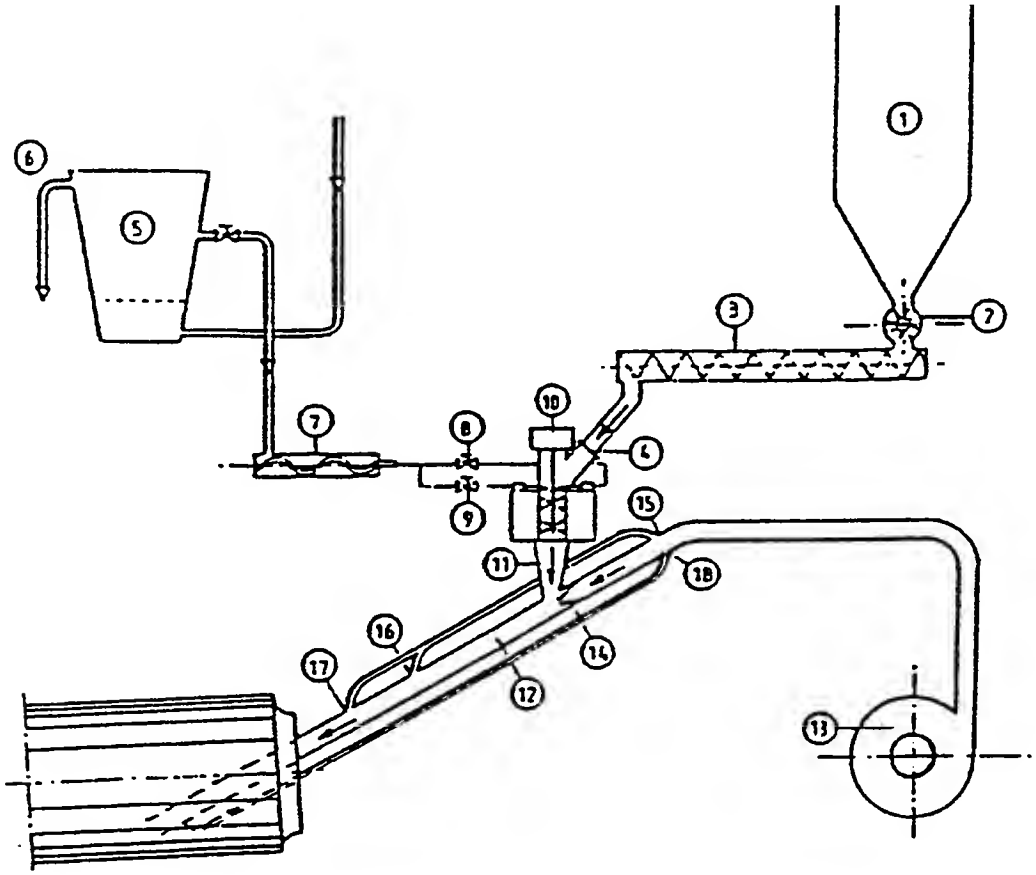
Figure 2: Dust circuit, Slurry circuit, Mixture and compressed air circuit



Dust circuit	Slurry circuit	Mixture and compressed air circuit
1 Silo	5 Surge tank	11 Mixture outlet
2 Rotary valve	6 Overflow	12 Injection pipe
3 Feed screw	7 Slurry pump	13 Fan
4 Stainless steel chute	8 Slurry injectors	14 Primary injector
	9 Slurry injectors	16/17 Secondary injectors
	10 Schugi blender	15/18 Branches to secondary injectors

As can be seen, the granulator consists of a vertical cylinder in which a concentric shaft is placed, rotating at a speed between 1'000 and 3'000 rpm. Attached to the shaft are several groups of knives whose angle to the shaft is continuously variable. To prevent the granules sticking permanently to the cylinder wall the latter is lined with a deformable neoprene liner which can be deformed from the outside by using pneumatically powered rollers, so causing any deposits to fall off. Kiln dust and a binding liquid are introduced into the granulator, which by the action of the rotating knives are formed into small granules, a residence time of 0.5 to 1 sec. being sufficient for granules of 0.5 mm to 1.5mm, max. 2 mm in diameter. Typically for a granulator of 10 t/h capacity only several kg of dust are present in the granulator at any one time, thus requiring an accurate and reliable dosage of both kiln dust and binder. Normally the kiln dust is itself sticky enough so that water alone can be used as granulating fluid.

At the Obourg plant the use of equal quantities of dust and slurry allowed the whole of the dust produced in kiln 8 to be granulated and fed back to the kiln. The return of the dust in this way allowed the water content of the slurry fed to the kiln to be reduced by some 2 to 3 % and increased the quantity of the dust generated about 2 to 4 %. To prevent the granules from sticking when being fed into the kiln via the 12 m tube, they were held in suspension in a carrier gas stream. The velocity of the gas suspension was about 9 to 10 m/s and the extra quantity of air introduced some 2 %. The tube was held about 40 cm above the level of the slurry on which floated a compact bed of granules, of which about 97 % were captured by the slurry. The granulator was shut down when kiln 8 went out of production.

4.3 Methods for In-Plant Use of Kiln Dust after Removal of Alkalis

4.3.1 Removal by Simple Leaching

An early method (1963) for the treatment of wet kiln dust, so as to allow its return to kiln, is the leaching system developed and used at Holnam's Dundee plant in the USA. As far as is known this is the only leaching plant of this type in operation. Here some 350 tpd dust are collected daily from both of the plant's 1'500 tpd wet kilns. Before the installation of the bag filter in the 1990's, the coarser dust taken from the smoke chamber and the first two compartments of the electrofilter was returned to the kiln by insufflation at a rate of about 10 tph through the burner pipe. The remaining fine dust with higher levels of volatiles (from chambers 3 and 4) was processed in the leaching plant, (see Fig. 3) in which over 70 % of the alkalis were removed using quarry water. This leaching plant is still operating today but the electro filter has since been replaced by a bag type.

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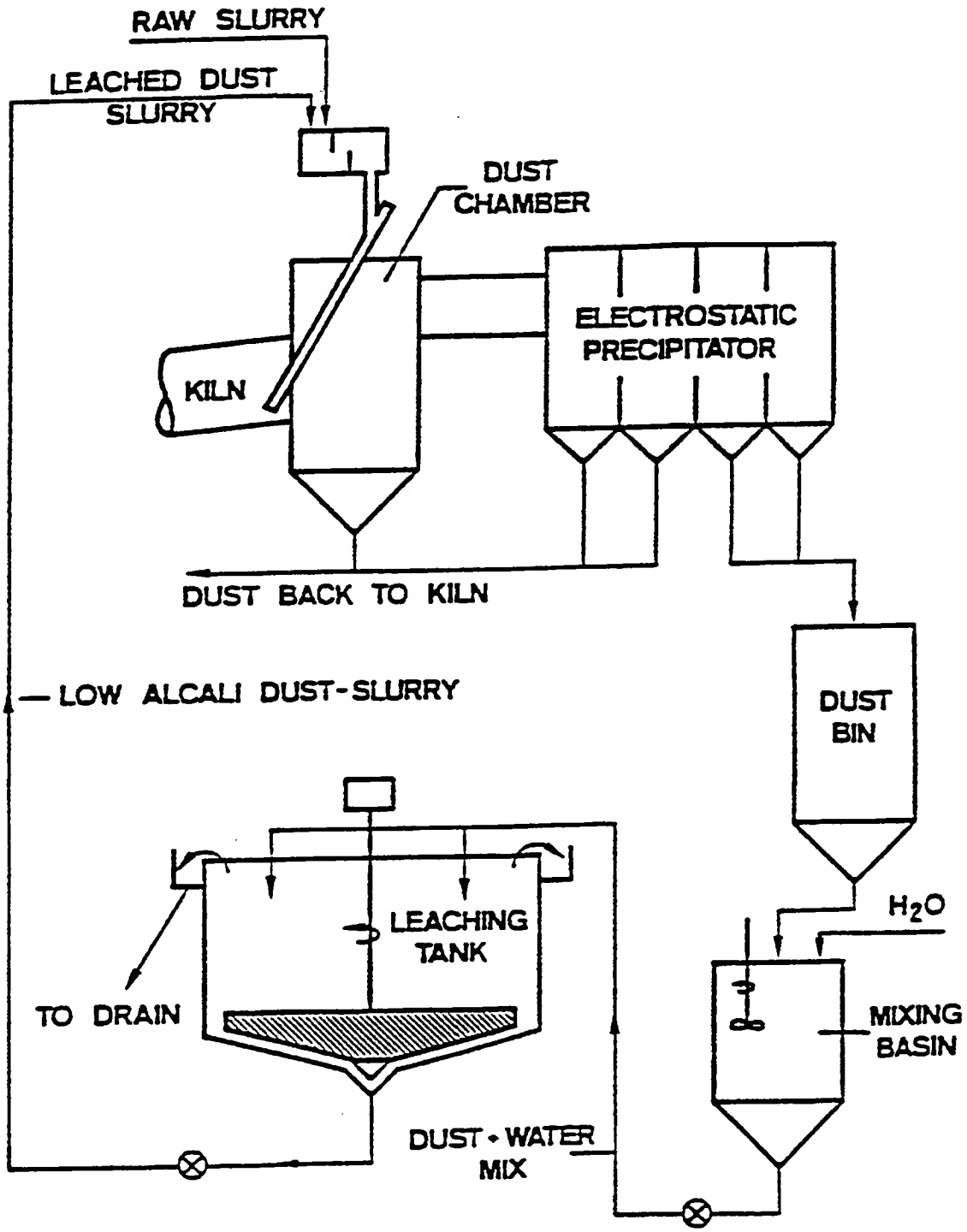
Figure 3: Dust Leaching System



The kiln dust is slurried batch-wise in a mixing basin using a water to dust ratio of 9:1 and then fed to the leaching tank from which the thickened slurry containing some 50 % water is then returned to the feed end of the kiln. The use of the leaching plant drops the K_2O value of the clinker from 1.1 to 0.5 % and the SO_3 from 1.0 to 0.4 %.

An important factor for the continuing operation of the leaching plant is ensuring that the effluent complies with the water pollution regulations. In the situation at Dundee the waste water disposal problem was two-fold, namely the high pH value of the leaching plant effluent and the high H_2S levels of the quarry seepage water. The two problems were reduced to one by using the quarry water for the leaching operations.

The effluent from the leaching plant has initially a pH value of 12.5 and contains S, SO_4, K, Na and Ca ions. It is neutralized by using steel mill waste pickling acid and ferrous sulphate producing $CaSO_4$ and $Fe(OH)_2$, with the H_2S being converted into insoluble sulfides. This treated effluent is then discharged into a system of three lagoons in the mined-out quarry where the solids settle out and the water is then aerated and clarified. Part of this treated water is used for clay slip preparation, the rest being discharged into the river under state regulations. As can be seen in Table 11 the operating permit of 1963 does not mention limits for the heavy toxic metals. Because of their low solubility at high pH values the concentration of the heavy toxic metals in the effluent will be anyway low.

Table 11: National Pollutant Discharge Elimination System (Permit No. MI. 0002020)

	Effluent Characteristics	Discharge Limitations	Monitoring Requirements
1.	Outfall 001		
	Flow	17'500 m^3/d	Daily
	Total suspended solids	20 - 30 ppm	Daily - 24 hr. comp.
	Total dissolved solids	5'000 - 6'000 ppm	Daily - 24 hr. comp.
	Oil and grease	No visible film	Daily visual obs.
	Fecal coliform bacteria	200 / 100 m/l	Weekly grab
	Total Iron-Potassium	--	Monthly 24 hr. comp.
	Hydrogen sulfide	0.1 ppm	Weekly grab.
	pH	6.5 - 9.5	Bihourly grab
2.	Outfall 001		
	Flow	4'500 m^3/d	Daily
	Total suspended solids	--	Monthly grab
	Total dissolved solids	--	Monthly grab
	Oil and grease	No visible film	Weekly visual obs.
	Temperature	--	Weekly regarding
	pH	6.5 - 9.5	Monthly grab

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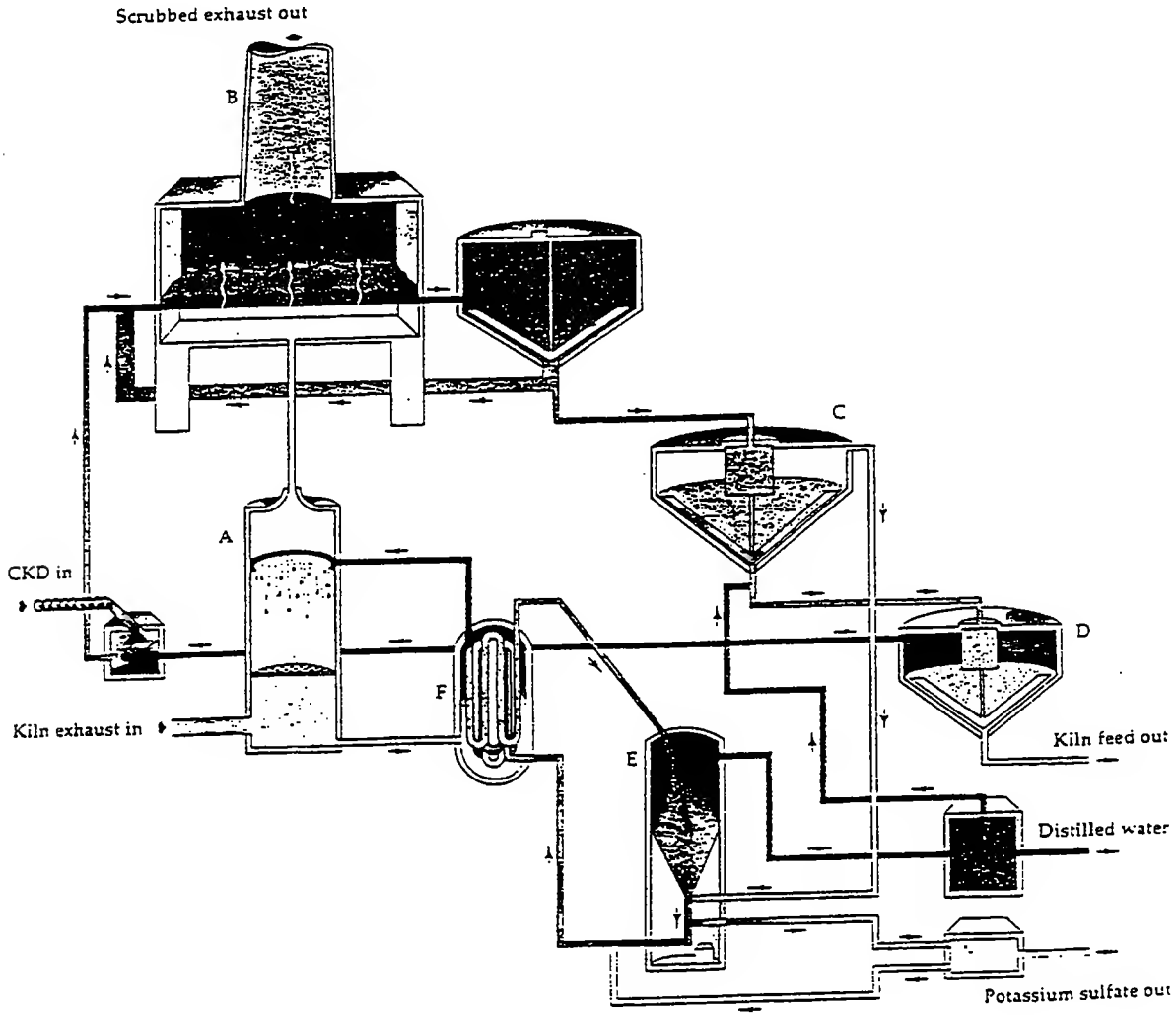
4.3.2 Removal by Passamaquoddy Technology Recovery Scrubber

This is essentially a modern development of the previously described leaching system and first went into operation in 1990 at the Thomaston plant in the USA. In spite of world-wide sales efforts this technology has only been installed at the above mentioned plant.

The layout of the equipment is shown in Fig. 4. An improvement over the Dundee leaching plant is that there is no discharge of alkali salt effluent into the waterways, these being converted into solid alkali sulphate/chloride to be sold as fertilizer, plus distilled water. The system also includes a scrubbing system which removes 90 % of the SO_2 from the kiln exhaust gases as well as reducing particulate emissions. It further allows the reprocessing of already dumped kiln dust.

Figure 4: Layout of the Passamaquoddy Scrubber



In the Passamaquoddy system kiln dust is mixed with water to form slurry just like at the Dundee leaching plant. Cooled kiln exhaust gas is bubbled through a slurry bedwhich reacts with lime and gypsum to form limestone and so avoids setting problems. At the same time the SO_2 anc Cl in the kiln exhaust gas converts the alkalis in the slurry into soluble sulphates and chlorides. The alkaline leachate is then fed to the crystallizer, which, using waste heat from the recuperator and reduced pressure, turns it into the desired products, namely alkali salts (mainly K_2SO_4) for sale as fertiliser and distilled water. It was also said that the scrubber would be able to remove dioxins and furans and convert the heavy toxic metals into insoluble carbonates.

The reasons for the non-preliferation of this system are believed to be that the costs were very high, in the order of 30 mio USD for a 3000 tpd kiln, and that a revenue from the sale of alkali sulphate fertilizer and of distilled water was never demonstrated.

4.3.3 Removal by Auxiliary Kiln or by Fuller Fluidised Bed Reactor

In very recent times Holnam Inc.,working with Fuller have carried out initial small, pilot scale trials to evaluate the possibility of producing low alkali clinker (Na_2O eqvt < 0.60%) from kiln dust using the Fuller's Fluid Bed CKD Recovery Process. Discarded dust from Holnam's Clarksville plant, both fresh and landfilled, was used for the test which demonstrated an alkali volatilisation rate < 65 % utilising a maximum permissible fluid bed operating temperature of 1310'C and oxidising conditions. The use, however, of a mildly reducing reactor atmosphere (0.8 % to 1 % CO) increased the alkali volatilisation rates to > 90 % utilising a slightly lower fluid bed temperature of 1300-1305 °C. Details of the Fuller reactor are presented in Fig. 5.

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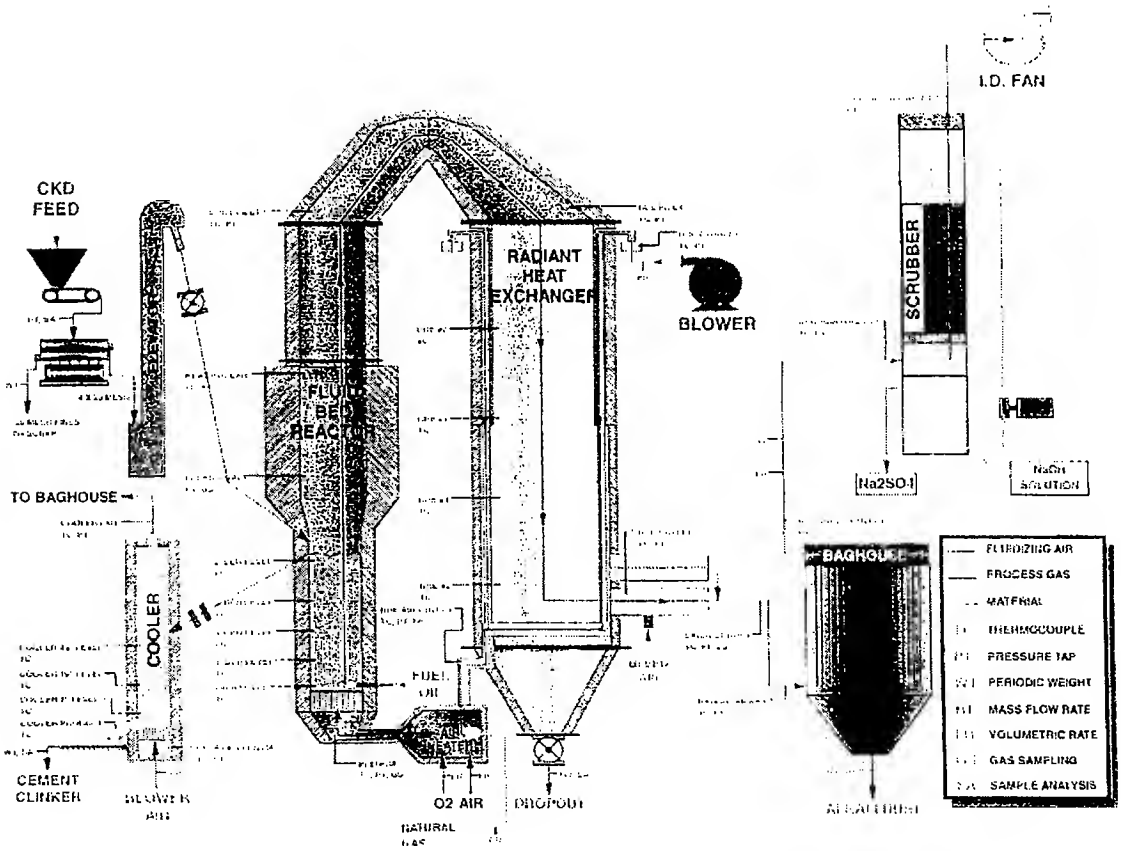
Figure 5: 18" Fluid Bed CKD Recovery System



From the results of the trials it can be said that:

- a clinker with 0.33 % free CaO was obtained after a 3 hr retention time
- the fluid bed system has a low rate of fines generation
- this fine dust, with typical composition 51.5% K_2O, 1.33% Na_2O, 33.29% SO_3 and 11.32% Cl, might be suitable as fertiliser
- the TCLP heavy metals in the dust appears to meet the limits of the RCRA leaching test.

At one plant now being built in Egypt, Polysius have foreseen the use of an auxiliary rotary kiln to concentrate the bypass dust prior to its disposal or further use.

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4.4 Reported Plant External Uses of Kiln Dust

4.4.1 As Fertilizer

The features of kiln dust which make it potentially useful as a fertilizer are its potassium content, its lime content and its acid neutralizing capacity.

When compared to potassium-lime fertilizers produced by the fertilizer industry, the disadvantages of kiln dust include its variable chemical composition, its lower K_2O content, its high fineness and associated nodulizing problems.

In general a potassium fertilizer should fulfill the following requirements which may differ somewhat from country to country:

- should be in nodule form
- only slight variations in chemical composition
- should have a minimum content of soluble K_2O and CaO (In Germany the K_2O content of potassium fertilizers should be between 40 to 60 % for the chloride types and between 25 to 50 % for sulphate formulations)
- for some crops a low Cl content is required

Lime fertilizers should have a CaO content of between 40 to 90 % depending on whether they consist of calcium carbonate, burnt lime or hydrated lime.

Only two plants were known in the past to successfully market their kiln dust. These were the ENCI plant at Maastricht and Alsen-Breitenburg's Itzehoe plant, which sold their dusts as fertilizers under the "KENCICA" and "Alsen" brand names. None of these plants today sells kiln dust as fertilizer; the Itzehoe plant was closed in the early 1980's and at ENCI the kiln dust is being legally added to the cement.

With regard to their chemical compositions the dusts from these plants had quite different characteristics as shown in Table 12. The fertilizer from Alsen was of the potassium type, that from ENCI of the lime-potassium type, and as can be seen these compositions are typical of the dusts from many wet and semi-wet plants.

Table 12: Chemical Composition (wt%) of the ENCI and Itzehoe Dusts

	ENCI	ITZEHOE
L.O.I. (500°C)	1.5	1 - 2
CO_2	18	2 - 6
SiO_2	18	2 - 6
Al_2O_3	3	0.5 - 1.5
Fe_2O_3	3	0.2 - 0.9
CaO	46	5 - 15
MgO	-	0.1 - 0.3
SO_3	5	32 - 39
K_2O	5	32 - 42
Na_2O	-	2 - 3
Cl	-	1

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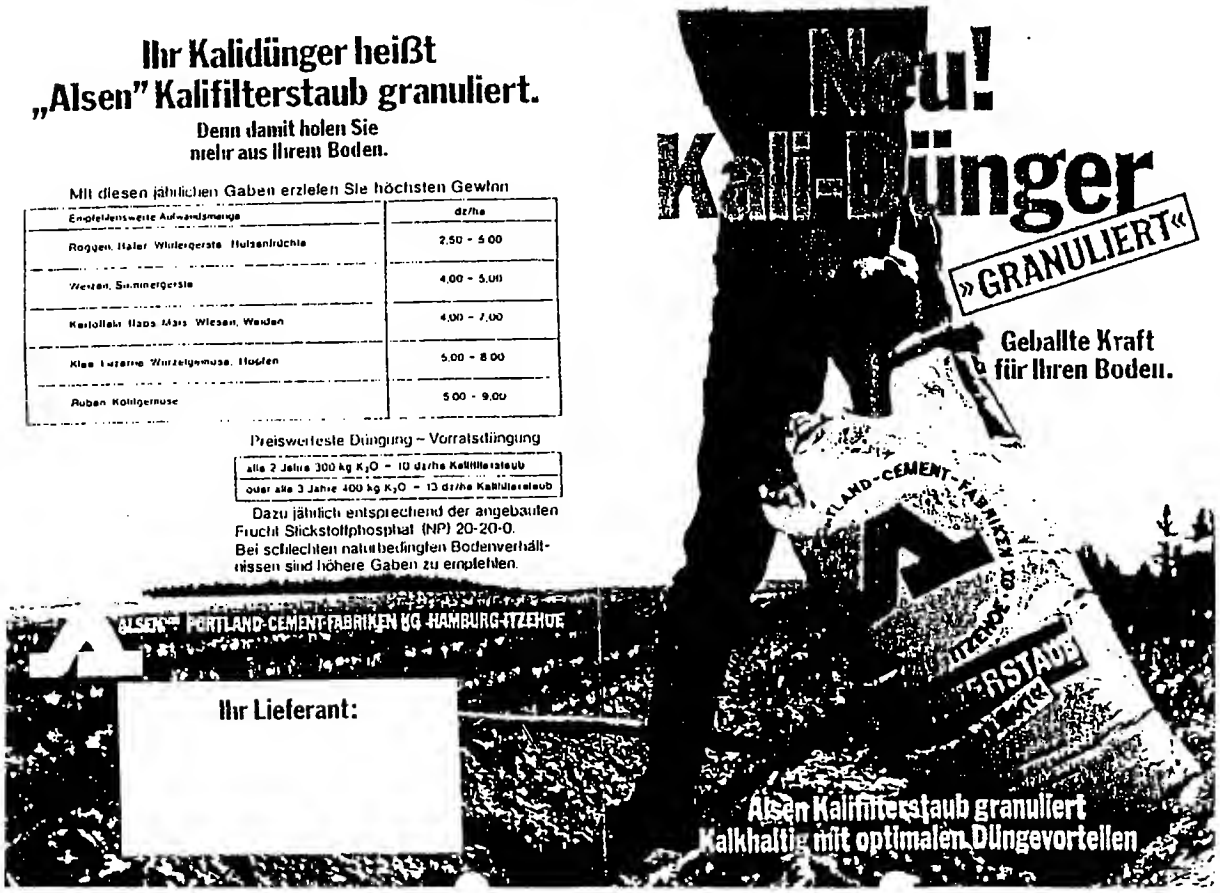
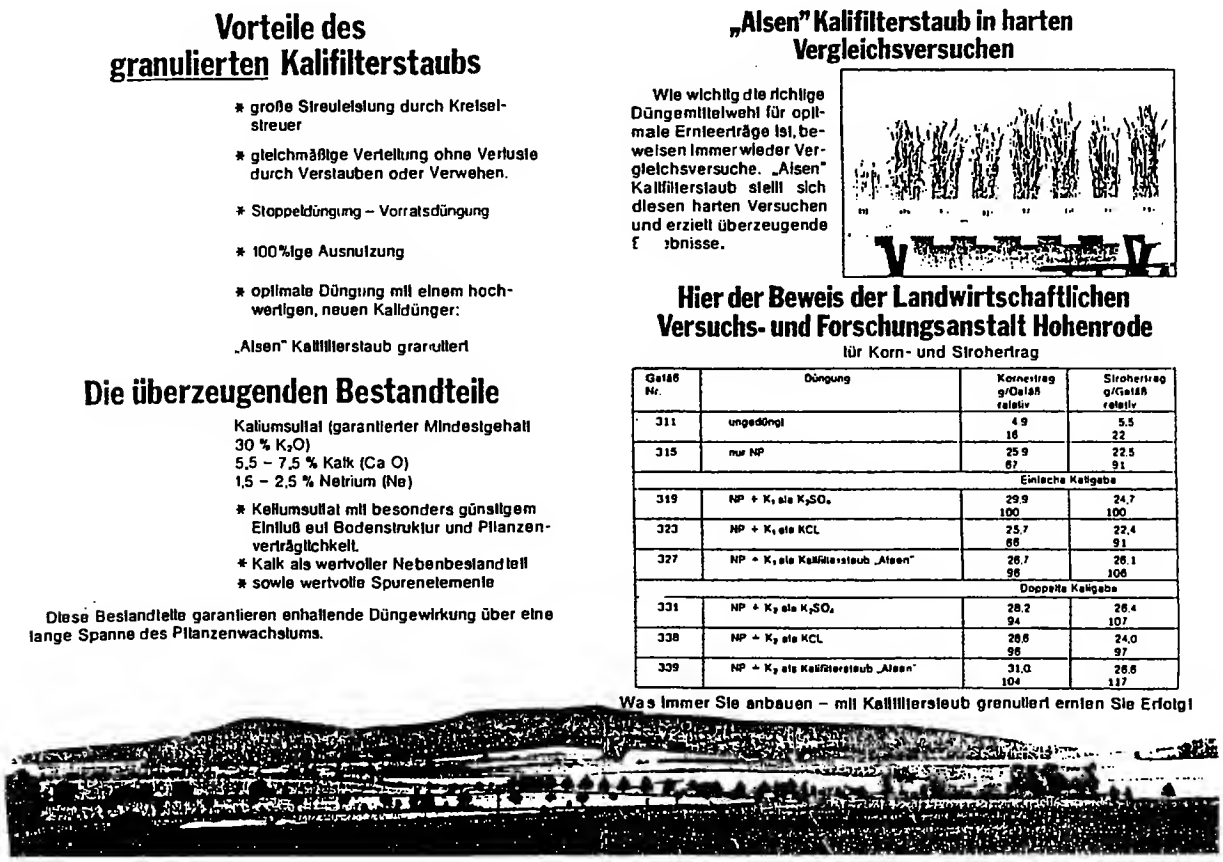
For the successful marketing of kiln dust a very large input may not be required. ENCI for instance did not market directly under their own name but did so through a separate company having a long experience in the business of providing farmers with fertilizers. They were able to claim on the basis of crop tests that barley, turnips, potatoes and sugar beet produced the same yields as when conventional lime or potassium sulphate fertilizers were employed. It was even claimed that sugar beet contained up to 30 % more sugar when "KENCICA" was used. ENCI claimed that the CaO to K_2O ratio of 9:1 and the presence of calcium silicates made their product especially valuable as a fertilizer.

In the case of "Alsen" the company did its own marketing, an example of the promotional literature being shown (in German) in Figures 6 and 7.

Figure 6: Promotional Literature for "Alsen" Fertilizer





Gefäß Nr.	Düngung	Kornertrag g/Gefäß relativ	Strohertrag g/Gefäß relativ
311	ungedüngt	4.9 16	5.5 22
315	nur NP	25.9 87	22.5 91
	Einfache Kaligabe		
319	NP + K_1 als K_2SO_4	29.9 100	24.7 100
323	NP + K_1 als KCL	25.7 86	22.4 91
327	NP + K_1 als Kalifilterstaub „Alsen"	28.7 96	26.1 106
	Doppelte Kaligabe		
331	NP + K_2 als K_2SO_4	28.2 94	26.4 107
338	NP + K_2 als KCL	28.6 96	24.0 97
339	NP + K_2 als Kalifilterstaub „Alsen"	31.0 104	28.8 117

4.4.2 As Raw Material for Artificial (N-Viro) Soil

An interesting use of kiln dust, presently practised in the USA is in the production of artificial soil. The kiln dust and bypass dust from a few US plants is being sold to a specialist company - N-VIRO - which mixes the dust with partially dried sewage sludge to produce an artificial soil, marketed as N-VIRO SOIL. The kiln dust, especially bypass dust, behaves as lime and when mixed with the dried sludge, causes an increase in temperature which kills most pathogens and greatly reduces the sludge odour, all this happening within only 7 days. The production of this artificial soil is approved by the EPA as a PFRP (Process to Further Reduce Pathogens) treatment. Uses to which N-VIRO soil is put include:

- daily landfill cover material
- land and mining reclamation
- land remineralization
- highway embankment construction
- landscaping and turf material
- agricultural lime/soil conditioner

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Details from N-VIRO's promotional literature are shown in Fig. 8 and the composition of the dust in Table 13.

Table 13: Chemical Composition of B

ANALYSIS	EPA-LIMIT	BYPASS DUST (%)
SiO2		17.55
Al2O3		4.87
Fe2O3		1.84
CaO		61.41
MgO		0.72
SO3		4.23
Na2O		0.51
K2O		1.53
TiO2		0.39
LOI		6.97
Total		**100.02**
E.P. TOXICITY PROCEDURE		
	Max. Limit ppm	**ppm**
As	5.0	0.003
Ba	100.0	2.44
Cd	1.0	0.05
Cr	5.0	0.21
Pb	5.0	0.55
Hg	0.2	< 0.001
Se	1.0	0.006
Ag	5.0	0.07
TOXICITY CHARACTERISTIC LEACHING PROCEDURE		
	Max. Limit ppm	**ppm**
	5.0	0.004
Ba	100.0	1.7
Cd	1.0	0.05
Cr	5.0	0.25
Pb	5.0	0.6
Hg	0.2	0.003
Se	1.0	0.002
Ag	5.0	0.04

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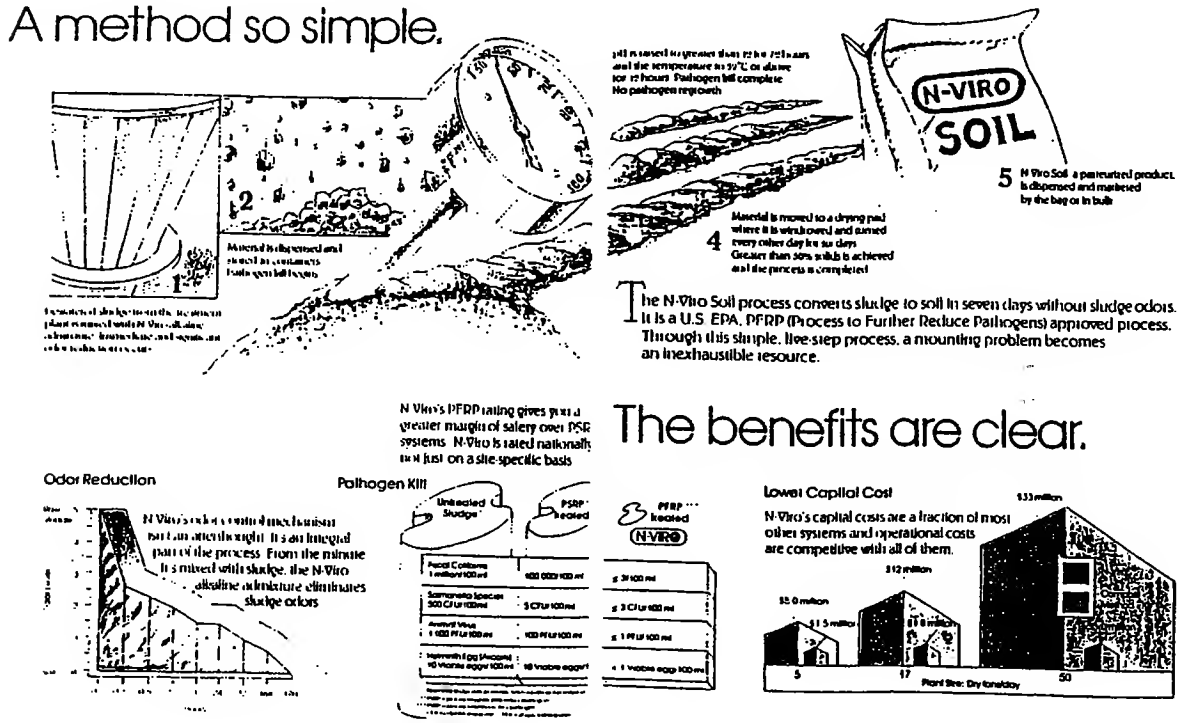
Figure 8: Promotional Literature for "N-Viro Soil"



4.4.3 As Asphalt Filler

Only one plant in the "Holderbank" Group has, as far as is known, been able to sell some of its kiln dust (approx. 200 t/a) for use as a filler in asphalt, this amounting to only 10 % of the approx. 2'000 t of dust sold each year.

On the basis of laboratory trials it was ascertained that only the dust from the first chamber of the electrofilter was promising as a filler. the other dusts having the following drawbacks:

- smoke chamber: too coarse
- chambers 2, 4 and 4: too low density
  too high solubility

Chamber 1 dust could be mixed with a normal filler (Vaulx) and used in the preparation of asphalt according to the Dutch "Zwacke" class. The production of "Zeer Zwacke" and "Middlesoort" classes was not possible. The proportion of natural filler that could be replaced with chamber 1 dust in the "Zwacke" grade is about 50 %. The performance of the various proportions of filler and chamber 1 dust are given in Table 14.

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Table 14: Comparison of Performance of Different Proportions of Filler and Kiln Dust as Filler Component in Asphalt

Characteristics	Belgian Standards	Dutch Standard			Filler (Vaulx)	Proportion Filler: Cust (Filed 1)				
		Z.Zwakke	Zwakke	Middlesoort		20:80	30:70	40:60	50:50	60:40
Residue on:										
1 mm		0	0	0	0	0	0	0	0	0
0.09 mm		0 - 15	0 - 15	0 - 15	5 - 10	5 - 10	5 - 10	5 - 10	5 - 10	5 - 10
0.63 mm		5 - 25	5 - 25	5 - 25	13 - 19	14.5 - 20.0	14.5 - 20.0	14.5 - 20.0	14.5 - 20.0	14.5 - 20.0
Apparent density in toluene	0.5 - 0.8				0.5 - 0.6	0.43	0.45	0.43	0.46	0.48
Porosity	30 - 50	> 29	> 36	> 44	30 - 34	46.5	46.0	43.7	43.9	39.5
Sensitive to water	< 10	< 10	< 10	< 10	< 10	0	0	0	0	0
Swelling	< 3	< 3	< 3	< 3	0.35 - 1.0	3.0	2.8	2.7	2.7	2.5
L.O.I (150°C)		< 1.5	< 1.5	< 1.5	0.1 - 0.21	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
Solubility in water		< 10	< 10	< 10	0.38 - 1.90	12.5	11.6	9.6	8.5	7.1
Bitumen (Getal)		28 - 30	40 - 50	54 - 60	32 - 37	58	56	55	50	44

5. DISPOSAL OF BYPASS DUST

5.1 In Plant Use

Because the original purpose of a bypass is to reduce the levels of Cl or alkalis in the kiln system, it is just not possible to return the dust so extracted to the kiln. At one Group plant (Höver) some 16 tpd of bypass dust (at times up to 30 tpd) is generated, equivalent to some 4'400 t/a. The composition of the dust is given in Tables 15 and 16. As can be seen the enrichment in Pb, Cd and Hg is high, the value for Pb being higher when tyres are being added as secondary fuel at the kiln inlet. Although it was possible in the past to dump bypass dust in the quarry, this is no longer the case and so uses have had to be found for this material.

Table 15: Bypass dust chemical composition

main components %	raw meal	clinker	bypass dust	enrichment factor raw meal bypass dust
ignition loss	35.8	0.3	6.5	
SiO_2	13.5	21.7	12.5	
Al_2O_3	3.6	6.1	2.2	
Fe_2O_3	1.3	2.0	2.5	
CaO	43.2	66.5	31.0	
MgO	0.6	1.1	0.4	
SO_3	0.6	1.0	11.0	18.3
Na_2O	0.3	0.3	2.6	8.6
K_2O	0.6	0.9	18.4	30.1
Cl	0.06	0.007	11.5	191.6

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Table 16: Bypass dust chemical composition

trace elements mg/kg/10^{-4}%	raw meal	clinker (not analyzed)	bypass	enrichment factor raw meal bypass dust
Sb	2.1	--	10.0	5.0
As	10.4	--	64.0	6.1
Be	1.0	--	1.6	1.6
Pb	21.0	--	947.0 (450)*	45.0 (21)*
Cd	1.0	--	46.0	46.0
Cr	16.0	--	90.0	5.6
Co	2.0	--	11.0	5.5
Cu	5.0	--	28.0	5.6
Mo	2.0	--	6.0	3.0
Ni	12.0	--	38.0	3.2
Hg	0.05	--	2.0	40.0
Se	2.0	--	56.0	28.0
Tl	4.0	--	27.0	7.0
V	41.0	--	80.0	2.0

*without burning tyres

Several plant internal uses have been found for the bypass dust including:

- addition at raw material preparation
- feeding to the raw meal
- addition to OPC and slag cements as allowed by ENV 197-1

The bypass dust is added to the raw meal only when the Cl content in the kiln feed falls to 0.04% to 0.05 % and sufficient dust is added to give a constant level of 0.08 % Cl. This ensures a smooth bypass operation. When added to cement the quantities are limited by the requirement that the maximum Cl content should be less than 0.1 %, and is in fact held constant at 0.05 %. The quantity of dust added on a cement basis varies between 0.8 and 2.25.%. The maximum quantity of bypass dust that can be used within the plant by this method amounts to some 2'400 t/a. As an instantaneous recycling is not always possible, some dust must be temporary stored until it can be used as previously described. Other plant internal potential uses which have been evaluated but did not lead to success were in the production of:

- low quality hydraulic binders
- special binding agents for salt mining

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6. DISPOSAL OF REFRACTORIES

6.1 Background

Prior to the 1980's the disposal of spent refractories invariably involved dumping the broken out material in the quarry. Thus only monetary aspects were taken into consideration and any potential burden on the environment was ignored. The introduction of strict dumping regulations has also altered this practice but in some countries an equally undesirable situation now exists. There, the same refractory materials which are continuously being worn down inside the kiln during normal operations cannot be disposed of by slow addition to the raw materials after their removal.

What quantities of spent refractories have to be disposed of? For say an average wet kiln, with a total consumption of 750 g/t basic refractories and 450 g/t of acidic materials this will mean that about 190 g/t of basic and 110 g/t of acidic refractories, i.e. 300 g/t in total actually have to be disposed of. This represents only some 0.03 % of the annual clinker production (0.019 % on a raw material basis). Assuming the spent refractories are disposed of over a period of 1 month this would mean the incorporation of 0.23 % spent refractories into the raw materials. This quantity is very small and is less than the quantity of most waste materials from other industries being disposed of in cement industry kilns.

HMC / CS is of the opinion that the plants of the "Holderbank" Group should aim at being able to dispose of all their spent refractories by incorporation into the raw materials.

6.2 Materials Involved and their Chemical Compositions

6.2.1 Materials Involved

In the widest sense the waste materials associated with cement kiln refractories must also be include the packing materials in which they are delivered. This includes wooden pallets, paper and plastic packing, steel and plastic drums etc. Such materials are disposed of within the framework of a plant's general refuse concept (see Chapt. 2) and will not be further considered.

The spent refractory materials fall mainly into the following categories:

- basic bricks of the magnesia and doloma types
- high alumina bricks
- fireclay and lightweight insulating bricks
- dense refractory concrete
- insulating boards
- steel plates, steel anchors

Most is in the form of bricks.

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6.2.2 Chemical Composition

The chemical compositions of typical cement kiln refractories as supplied are given in Table 17. Here it can be seen that, apart from the elevated MgO and Cr_2O_3 contents of some types, their compositions are similar to the oxide components used in clinker manufacture. During their operational lifetime the original chemical composition changes due to various reactions which include:

- reaction with non volatile CaO and SiO_2 from kiln charge
- reaction with CO_2 and SO_2 kiln atmosphere (doloma types)
- reaction with with alkalis and sulphates to form Cr^{6+} compounds (magnesia chrome types)
- pick up of alkalis, sulphates and chlorides into open pores

Table 17: Typical Cemical Cmposition of Various Classes of Refractories (Unused)

Element	Magnesia Spinel	Magnesia Chrome	High Alumina	Dolomite	Fireclay	Refractory Concrete
L.o.I.	0.15	0.22	(2.7)	0.29	0.12	0.98
SiO_2	63	2.3	11.5	0.61	55	49.8
Al_2O_3	10.5	8.5	77.5	0.27	39.1	24.2
Fe_2O_3	0.57	4.9	1.6	0.88	1.8	2.6
CaO	0.91	1.2	0.13	58	0.32	13.9
MgO	87	71.5	0.23	37.6	0.11	1.5
SO_3	0.01	0.01		0.02	0.01	0.14
K_2O	0.02	0.02	0.32	0.02	1.1	0.94
Na_2O	< 0.01	< 0.01	0.02	0.01	0.12	3.7
TiO_2	0.04	0.11	3.1	0.02	1.5	1.3
Cr_2O_3	0.15	12.3				
Mn_2O_3	0.01	0.08	0.01	0.08	0.02	0.05
P_2O_5	0.07	0.04	2.8	0.02	0.15	0.05
Cl	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01

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6.3 Regulatory Aspects / Manufacturers Safety Data Sheets

Within the European Union and the USA various regulations are now in operation which regulate the disposal of spent kiln refractories. Details have already been given in Chapt. 2.

Of importance to cement plant refractories specialists is the safety data sheets that most European refractory manufacturers are now beginning to draw up. An example of such a safety sheet, issued to comply with EU Guideline 91/155/EG, is given in Appendix I. As can be seen in page 3 of this guideline, comments are made regarding the disposal of the refractory materials (cut pieces and breakout) and also of the packing materials.

These guidelines are for consumers in general and do not take into account the cement kilns capability of disposing of its worn refractories. Normally mention is made to the presence of Cr^{6+}.

6.4 In-plant refractories disposal

In many Group plants, especially in Europe, spent refractories are cost effectively and safely disposed of via the kiln, into which they are introduced at a controlled dosage rate by:

- milling in the raw mill or by
- milling in the coal mill

In both cases the spent refractories first require to be crushed and any steel plates, shims or anchors removed by a magnet (or in some cases by hand) before grinding

The crushing process can be either carried out in:

- existing plant crushers (raw materials or clinker) or using
- a mobile crusher from an external company (disposal via coal mill)

The route via the coal mill is of special importance for the case of spent doloma refractories in wet process plants as their inclusion into the raw materials would have an adverse effect on slurry viscosity and pumpability.

In the following examples are given of several methods of in plant disposal.

6.4.1 Plant "A"

In this semi-wet process plant fitted with a filter press, the spent bricks leave the kiln via the grate cooler and are crushed via the clinker crusher . They are then transported on a band, where a magnet removes any metal, to a small silo from which the crushed refractories are weighed into the raw feed at a slow controlled rate. The waste refractories are thus kept within the plant, require no additional handling and are effectively out of sight.

6.4.2 Plant "B"

In this large wet process plant the spent refractories are removed from the kiln via a man hole and are taken by truck to a covered storage. At regular intervals an external contractor crushes the spent material with a mobile crusher and removes the steel present. The crushed refractories are fed slowly into the coal mill and introduced into the kiln as an additional "fuel ash". It is thus homogeneously distributed.

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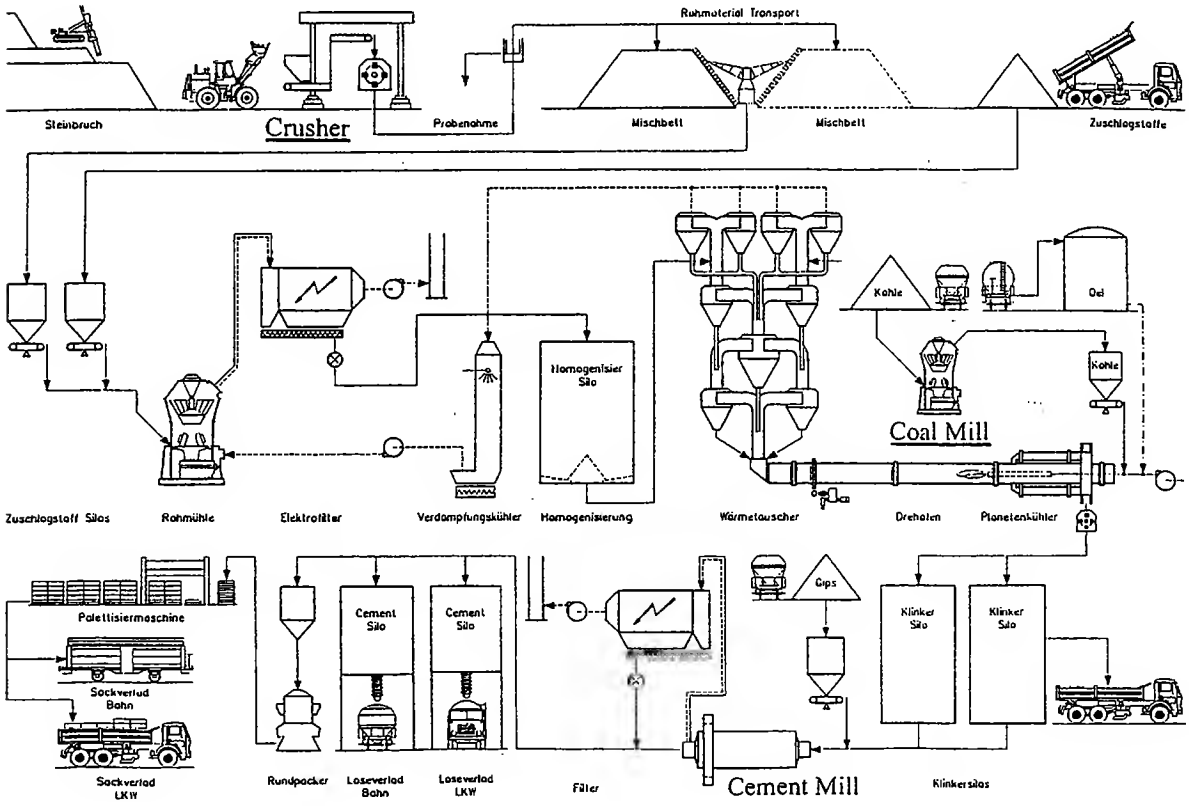
Figure 9: In-Plant disposal of Worn Refractories



7. RECYCLING OF SPENT CEMENT KILN REFRACTORIES

The philosophy of:

- avoiding / reducing waste,
- recycling or down cycling it,
- generating energy by its disposal and
- finally disposing of the "ultimate waste"

is now becoming established in many developed countries. When applied to cement kiln refractories all these possibilities, with the exception of energy generation, are in principle thinkable.

Under the motto, "nothing will be recycled until it is economic", the recycling of spent cement kiln refractories does not appear to offer much potential. The only materials that could in theory be completely recycled are the small volumes of cuttings from unused bricks bricks and rest bricks which were never installed. Once used refractories from cement kilns are normally not suitable for recycling.

When viewed from a purely financial viewpoint low cost firebricks will never be worth recycling and only high cost qualities such as magnesia spinel might have some potential. From a technical viewpoint used cement kiln refractories are always contaminated with a host of substances which substantially lower their melting temperature and so prevent their re-use in cement kilns. At best recycling means use in a lower or non refractory context and certainly not in the cement industry.

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From a practical point of view compound materials such as monolithics will probably never be able to be recycled. The recycling of dolomite qualities would necessitate the resintering of the material because of deterioration of the sinter due to hydration.

In Japan used magnesia chrome bricks from the cement and glass industries are classified as hazardous wastes and are returned to the supplier. This magnesia chrome waste is recycled as gunning mixes for application in BOF steel works.

However, even if landfilling costs were to become astronomical the fact that the possibility remains (or should remain) for a cost effective disposal via the clinkering process, means that it is very unlikely that cement industry refractories will ever need to be recycled on a regular basis.

As far as the production of cement is concerned the remnants of kiln refractories cannot be returned and recycled by the refractory manufacturer. The techniques required for this are not available and the quantities involved are small. Kiln refractories are and will stay consumable items. Only in a few developing countries are worn kiln refractories used outside of the plant as aggregate for the production of low quality refractory concrete. This form of "disposal" could not be practised in developed countries.

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8. Appendixes

8.1 Appendix Ia

MATERIAL SAFETY DATA SHEET following 91/155/EEC

Code: SD.304.1-0
State: Sept. 25/95
page: 1 of 4

Last revision: September 25/95

Tradename: **PERILEX® 80 / PERILEX® 83**

1 Identification of the substance/preparation and of the company/undertaking

1.1	Supplier:	REFRATECHNIK GmbH
1.2	Address:	D-37079 Göttingen, Rudolf-Winkel-Straße 1
1.3	Telephone:	(Germany) 0551 6941 0
1.4	Telefax:	(Germany) 0551 6941 104
1.5	Telex:	968 63, 968 11
1.6	Information:	R&D-Department (0551 6941 208)
1.7	Emergency Telephone No.:	R&D-Department (0551 6941 208)

2 Composition/Information on ingredients

2.1 Chemical Characterisation: **Magnesia chrome brick**

2.1.1

CAS-No.	Chemical name	Concentr. in %	Symbol	R-Phrases
1309-48-4	Magnesia	50 - 100	n.k.	n.k.
1308-31-2	Chrome ore	2.5 - 10	n.k.	n.k.

3 Hazards identification

As supplied the product is not dangerous. During handling (cutting, grinding, wrecking) non-siliceous dusts may be formed (see also chapter 8.). Under oxidising alkaline operating conditions the chromium^{3+} compounds may form minor amounts of chromium^{6+} compounds. Chromium^{6+} compounds are known to cause injury to the skin and respiratory tract which may lead to cancer. Chromium^{6+} compounds are water soluble and may lead to aquatic toxicity.

4 First Aid Measures

4.1	General Information:	none
4.2	Inhalation:	Remove to fresh air
4.3	Skin contact:	Wash gently with soap
4.4	Eyes:	Flush eyes with plenty of water, seek medical care
4.5	Ingestion:	Drink plenty of water, seek medical care
4.6	Information for the Doctor:	none

5 Fire-fighting Measures

5.1	Suitable Extinguishing Media:	no restrictions. As per surrounding area.
5.2	Extinguishing Media which must not be used for Safety Reasons:	none
5.3	Special Exposure Hazards arizing from the Substance or Preparation itself, Combustion Products, Resulting Gases:	none
5.4	Special Protective Equipment for Firefighters:	not necessary

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8.2 Appendix Ib

MATERIAL SAFETY DATA SHEET following 91/155/EEC

Code: SD.304.1-0
State: Sept. 25/95
page: 2 of 4

Last revision: September 25/95

Tradename: **PERILEX® 80 / PERILEX® 83**

==========

6 Accidental Release Measures

Avoid dry sweeping. Wet material, sweep up and bag. Minimize dust generation.

7 Handling and Storage

7.1 Handling: **A small amount of dust may be generated during handling.**
7.2 Storage: **Store dry and cool away from fierce heat that may cause damage to packaging material.**

8 Exposure Controls/Personal Protection

8.1 Engineering measures: **Ensure sufficient ventilation during handling.**
8.2 Specific Control Parameters:

CAS-Nr.	Substance	Type	Limit
	general limit for dust (< 5 µm)	**MAK[1)]**	**6 mg/m³**
	chromium^{6+} compounds	**MAK[1)]**	**0,1 mg/m³**

[1)] Maximum concentration at working place

8.3 Personal Protective Equipments
8.3.1 General Protective and Hygienic Measures: **Keep away from food. Before breaks and at the end of work wash the hands.**
8.3.2 Respiratory Protection: **Approved dust respirator recommended**
8.3.3 Hand Protection: **Gloves recommended**
8.3.4 Eye Protection: **Approved eye protection recommended**
8.3.5 Clothing: **Safety boots recommended**

9 Physical and Chemical Properties

9.1.1 Appearance: **solid**
9.1.2 Colour: **grey-brown**
9.1.3 Odour: **n.a.**

			Value/Range	Unit	Testing Method
9.2.1	Melting Point/Melting Range:		**1980**	°C	
9.2.2	Boiling Point/Boiling Range:		**n.a.**	°C	
9.3	Flashpoint:		**n.a.**	°C	
9.4	Flammability (solid, gas):		**not flammable**		
9.5	Ignition Temperature:		**n.a.**	°C	
9.6	Autoflammability:		**n.a.**		
9.7	Explosive Properties:		**not explosive**		
9.8.1	Lower Explosion Limit:		**n.a.**	Vol.-%	
9.8.2	Upper Explosion Limit:		**n.a.**	Vol.-%	
9.9	Vapour Pressure:	at 20 °C	**n.a.**	mbar	
9.10	Bulk density:	at 20 °C	**2.90 - 3.05**	g/cm³	
9.11.1	Solubility in Water:	at °C	**not soluble**		
9.11.2	Solubility in:	at °C	**n.a.**	mg/L	

8.3 Appendix Ic

MATERIAL SAFETY DATA SHEET following 91/155/EEC

Code: SD.304.1-0
State: Sept. 25/95
page: 3 of 4

Last revision: September 25/95

Tradename: **PERILEX® 80 / PERILEX® 83**

9	**Physical and Chemical Properties (Cont.)**			
9.12	pH-Value in:	at 20 °C	approx. 10	
9.13	Partition Coefficient. n-Octanol/Water		n.a.	log POW
9.14	Viscosity:	at °C	n.a.	
9.15	Solvent Separation Test:	at °C	n.a.	%
9.16	Solvent Content		n.a.	weight.-%
9.17	Other Data:		none	

10	**Stability and Reactivity**	
10.1	Conditions to avoid:	**Magnesia containing bricks can react with water to form hydrates which are non-hazardous, but destroy the bricks mechanically. Minor amounts of chromium^{6+} may be formed from chromium^{3+} compounds under oxidising and alkaline operating conditions.**
10.2	Materials to avoid:	**Water**
10.3	Hazardous Decomposition Products:	**none**
10.4	Further Data:	**As supplied the product ist even under elevated temperatures stable.**
11	**Toxicological Information**	
	none	
12	**Ecological Information**	
12.1	Persistence and Degradability:	**persistent.**
12.2	Mobility:	**none**
12.3	Bioaccumulative Potential:	**none**
12.4	Further Ecological Informations:	**Before undertaking the wrecking of used magnesia-chrome refractory linings, customer are advised to evaluate any changes to the product that may be induced by the introduction of substances and operating conditions not under the control of vendors. The product in the supplied or unused form can be disposed as solid non-hazardous waste according to national laws and subject to the absence of chromium^{6+} compounds.**
13	**Disposal Considerations**	
13.1	Product:	**Before disposing examine that amounts of chromium^{6+} and dispose the product according to the result of this examination.**
13.2	Contamined Packing Material:	**Combust or dispose**
14	**Transport Information**	

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8.4 Appendix Ib

MATERIAL SAFETY DATA SHEET following 91/155/EEC

Code: SD.304.1-0
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10.3	Hazardous Decomposition Products:	none
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11	**Toxicological Information**	
	none	
12	**Ecological Information**	
12.1	Persistence and Degradability:	persistent.
12.2	Mobility:	none
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Chapter 8

Quality Assurance

1. Quality Assurance 277
2. Quality Systems in the Cement Industry according to ISO Standards 295

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Quality Assurance

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1. INTRODUCTION

Every industry producing goods or services is confronted with the subject of quality. Their products are in the true sense of the word qualified by the customer with respect to the extent they meet his expectations. Included in his expectations is a conception of the price; thus, quality cannot be dealt with apart from cost aspects: on one hand, a product able to fulfill high and special requirements usually fetches a higher price than a "middle of the road" product; on the other hand, its production costs are as a rule also higher. Furthermore and that is not of less importance for the producer the market for a highly specialized product might be limited and the profitability questionable in spite of the high specific selling price.

It is the fundamental task of the quality assurance of a company to deal with the product quality and its relation to production costs, selling price and market potential, with the objective to manufacture a profitable product.

The establishment of an efficient quality assurance and its integration within all levels of company organisation, has for many companies been the basis for their success the brilliant advance of various sectors of the Japanese industry is an impressioning example.

In view of this experience, a generally applicable concept of quality assurance has been systematically developed, and has reached now the stage where quality assurance systems are specified in national or international standards.

Also in the cement industry, the needs for a well conceived quality assurance have significantly increased over the last twenty years, mainly due to the following developments:

- more stringent requirements on concrete and cement
- growing competition due to over-capacities
- escalating energy costs
- more efficient - though also more delicate - production facilities
- increasingly tighter environmental regulations
- involvement in active environmental protection

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2. DEFINITION OF QUALITY AND ITS CEMENT RELATED INTERPRETATION

Quality is defined as follows:
Quality is the totality of features and characteristics of a product that bear on its ability to satisfy a given need or fulfill certain requirements [1].

It is obvious that cement producers and consumers have a different attitude towards cement quality.

- The cement manufacturer is above all interested in low investment and production costs. Quality, therefore, is primarily associated with costs for materials, energy, equipment and labour. His major concern usually was - and often still is - concentrated on the suitability and uniformity of raw materials and intermediate products to allow optimum operation of the expensive installations with respect to quantity and costs.
- On the other hand, the consumer expects a product with favourable characteristics for its many applications. The changed market conditions - these days most countries envisage a very pronounced buyer's market - put him in a very strong position with regard to quality requirements. He need not be satisfied by the mere fulfilment of the standard specifications of the purchased cement, but may well demand more stringent requirements. Often, these additive requirements mean higher strengths and excellent uniformity - permitting to minimize this, according to the user, expensive ingredient in concrete - but they can also refer to other properties such as rheology (concrete workability) or colour. Furthermore, the consumer may urge the producer for detailed information not only about the final product, but even on all stages of cement manufacture. This may ultimately result in periodic inspection visits to the plant by external officials to so-called protect the consumers interests. Quality assurance by the cement manufacturer, therefore, is interpreted by the consumer as quality insurance in the sense of product liability.

Hence, the quality requirements of producers and consumers are by far not congruent; while a certain limitation of the product variability is generally in line with the manufacturer's attitude towards quality, his product costs can be seriously affected by the user's request for pushed quality levels, excessive uniformity and special requirements (Fig. 1). Depending on the suitability of the available raw materials, correctives, additives etc., it will be more or less costly for the manufacturer, if he has to comply with all the specific requirements. Moreover, the requirements of the two parties change significantly with time, and this change does usually not follow along the same lines.

[1] ISO 8402 - 1986: Quality : Vocabulary

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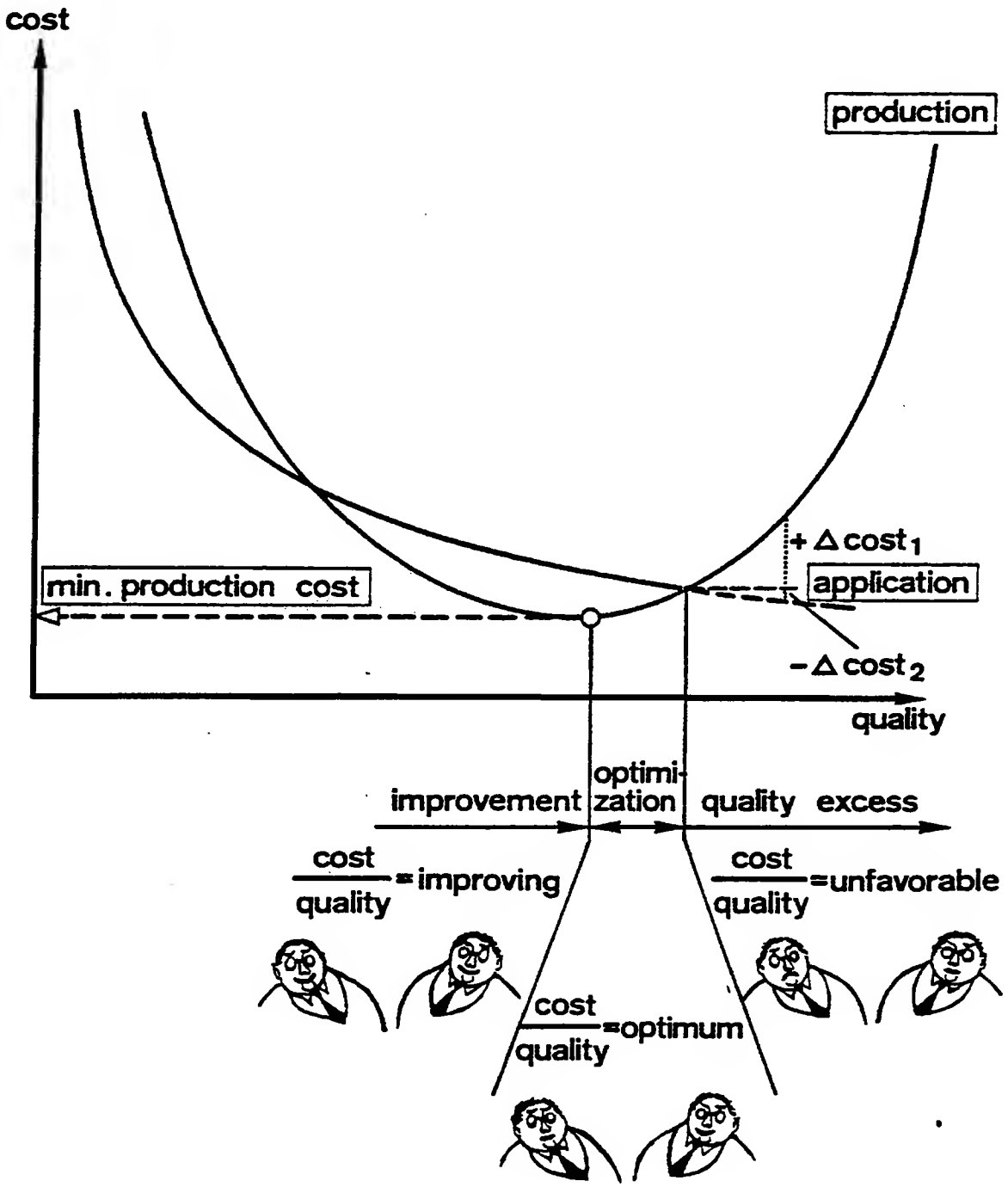
Fig 1 Cost/Quality Relation



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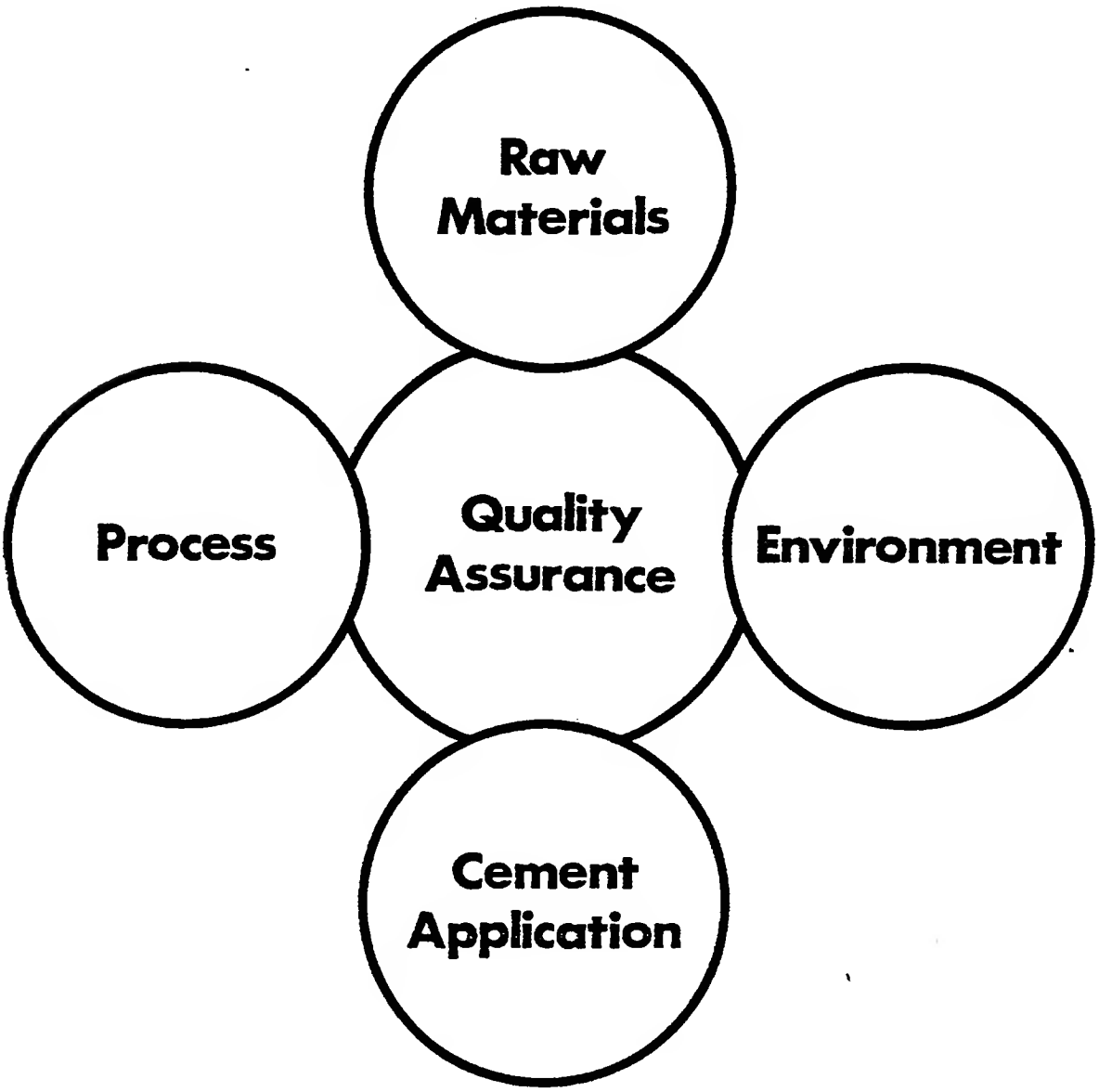
3. OBJECTIVES OF QUALITY ASSURANCE

It is the task of quality assurance to find an appropriate compromise between the divergent interests of producer and consumer.

More specifically, quality assurance has to ensure (Fig. 2):

- the manufacturing of market-conform products
- a smooth process by an appropriate design of the raw mix and other intermediates
- an economic utilisation of raw materials
- the compliance with environmental regulations

Fig 2 Objectives of Quality Assurance



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The prevailing trends within these four subjects can be summarised as follows:

1) cement application:
Product oriented aspects gained in importance over the past years. For many plants it is no longer sufficient to produce cement meeting the standard specifications or any nationally accepted quality level, but they are obliged to adapt their products and product ranges to specific customer requirements, be it for the inland or export market.
The implementation of technical consulting services for cement application provides an important marketing support for many plants in competitive markets.

2) process:
Induced by world-wide cement over-capacities and rising energy costs, plants are under considerable pressure to produce more cost-efficiently. A detailed optimisation of the raw mix and other intermediates with respect to process requirements can significantly contribute to this aim. Rationalisation by means of automatisation of process and quality control are also potential measures. However, they require a careful return-on-investment analysis.

3) raw materials:
Optimal utilisation of raw material deposits, both long - and short-term, has become an economic necessity and is nowadays facilitated by powerful computer programs.
Waste materials - both combustibles and non-combustibles - are successfully utilised in cement manufacturing. Special measures and investigations are necessary prior to and during their application.

4) environmental aspects:
Just as any other industry, the cement industry has to comply with increasingly stringent regulations on emission and immission. Different to most other industries, it offers, however, the possibility to actively contribute to a safe disposal of waste materials by means of their utilisation for cement manufacturing.

4. ACTIVITIES IN QUALITY ASSURANCE

By definition (ISO 8402), quality assurance comprises all those planned and systematic actions necessary to provide adequate confidence that a product or service will satisfy given requirements for quality.

Within the concept of quality assurance, three principle functions are distinguished:

Quality Assurance		
Quality Planning	Quality Control	Quality Audit

4.1 Quality Planning

The basic objective of quality planning is to determine the company's strategy and tactics in quality assurance. This begins with defining the relative significance of quality assurance and its main areas of emphasis, taking the particular situation of a company, e.g. with respect to market or to raw materials resources, into account.

The individual subjects that have to be dealt with are compiled Fig. 3, and can be summarised as follows:

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Outwards activities have to include the decision making in co-operation with marketing on number and types of products to be manufactured on the basis of criteria such as: demand of domestic and export market, raw materials situation, plant characteristics, etc.; evaluation of the demand for an extended support of the marketing department, e.g. in form of a technical customer service, and, if required, the realisation of such a service; furthermore, to represent the company's or cement industry's interest in the design of cement and concrete standard specifications or in the environmental legislation, by participation in the respective committees or bodies.

Plant internal planning activities comprise the design of the raw mix(es) as a direct follow-up to product planning and, in the same context, the establishment of a concept for short-and long-term raw material exploitation and securing. Of a very practical relevance is the set-up of a concept for quality control; questions regarding the personnel structure, the equipment, the degree of automation as well as data-processing and communication are of primary importance. Last but not least it is also incumbent on quality planning to increase the knowledge on the quality criteria and to develop a quality consciousness of the personnel by adequate training. Addressants are on the one hand the management of both the production and marketing sectors, and, on the other hand, the subordinate staff and unskilled labour members, since it will ultimately depend on them whether quality assurance is properly observed in all sectors of the plant activities.

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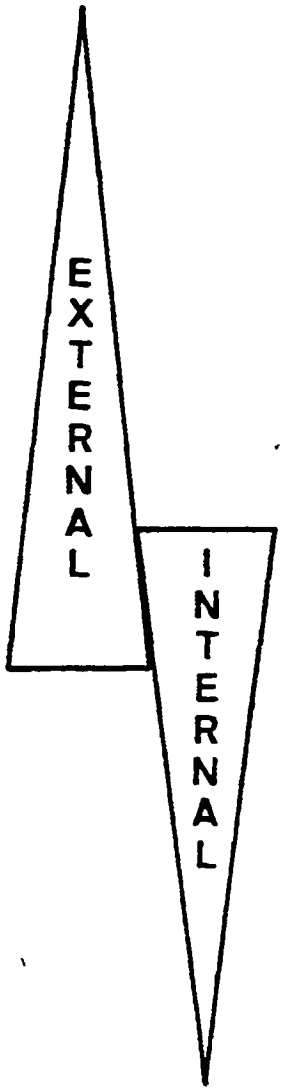
Fig 3 Quality Planning

QUALITY PLANNING

STATEGY AND TACTICS

- PROFESSIONAL ORGANISATIONS
- ENVIRONMENTAL LEGISLATION
- STANDARD COMMITTEES
- TECHNICAL CUSTOMER SERVICE
- MARKET
- MATERIAL RESOURCES
- PRODUCT DESIGN
- MIX DESIGN
- EXPLOITATION CONCEPT
- CONTROL CONCEPT
- EDUCATION/TRAINING



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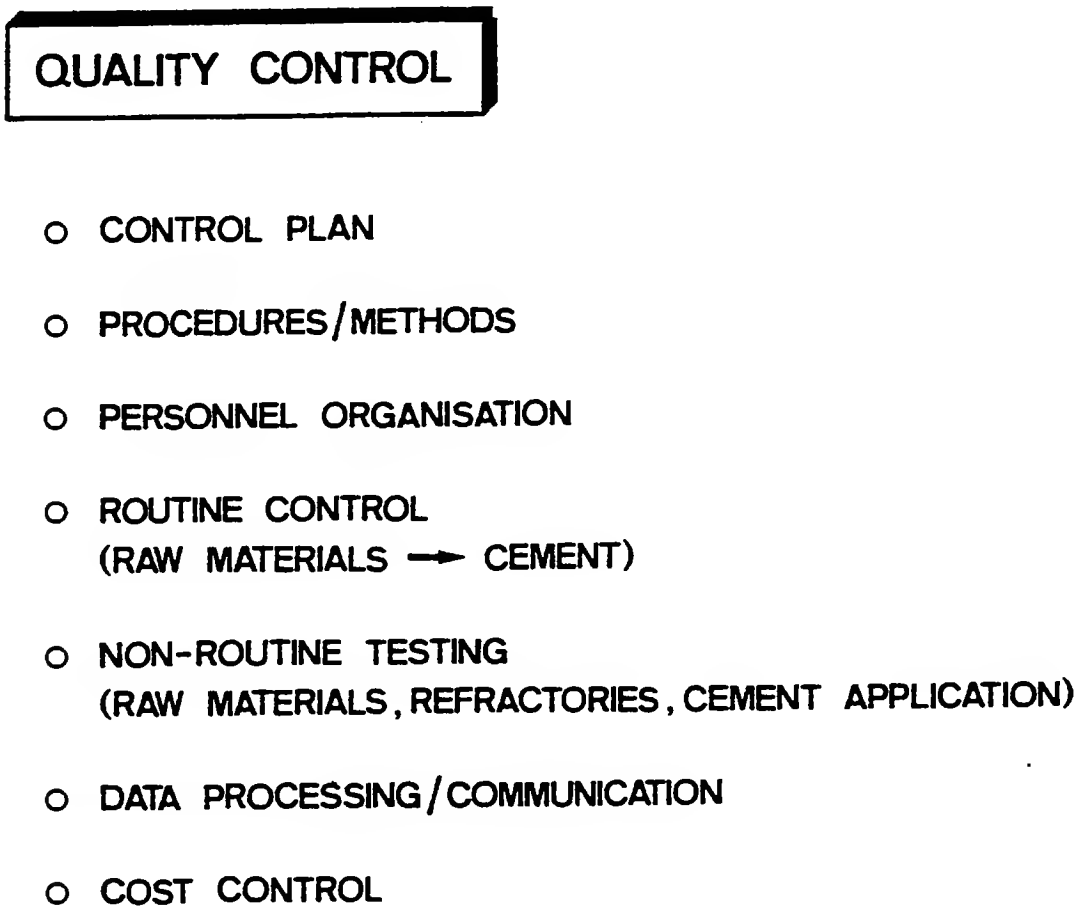
4.2 Quality Control

The task of the quality control (Fig. 4) is the routine monitoring of material properties throughout the process and the initiation of necessary adjustments in the process controls. The pre-requisite are an elaborate control plan and well - established methods and procedures for sampling and testing. It needs no particular mention that the infrastructure of quality control can also be used for non-routine purposes.

Fig 4 Quality Control



In view of new technologies, special attention has to be paid nowadays to data storage, processing, presentation as well as to data communication to the plant management and to process and sales departments. Data processing and communication not only serve to initiate process control measures - which can, in certain cases, be done directly by computer control - but also as a "quality certificate" for produced or dispatched products and as basic information for quality planning.

4.3 Quality Audit

Quality audit - as an integral part of the QA system - has the objective to critically scrutinise from time to time the procedures and efficiency of quality planning and quality control and to suggest adjustments if deemed necessary. Particular reasons for performing a quality audit are certainly significant changes, e.g. in the market situation, in the raw material situation or in the plant itself (extension, shut-down of production lines). It might be sensible to have a quality audit carried out by a third party in order to ensure an unbiased approach.

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5. QUALITY ASSURANCE SYSTEM

5.1 The Elements of a Quality Assurance System

In slight deviation from the official definitions, quality assurance system may be defined as the organisational structure, responsibilities, procedures, processes and resources for implementing quality assurance.[1]

In very general terms, the elements of a quality assurance system comprise:

- Organisation of personnel, dedication of tasks and responsibilities
- Lines of communication of information and decisions in routine and non-routine situations
- Quality control proper, i.e. sampling, testing, initiation or direct performance of reactions (automatic and manual)
- Training scheme
- Technical customer service
- Co-operation with external official bodies
- Quality cost control

It is evident that a quality assurance system has to be individually tailored to the specific needs of a company, taking regard of the complexity of the plant situation, of the prevailing market situation, and of the relationship with official institutions.

Detailed lists and descriptions of the required elements of a quality assurance system are contained in national or international standards, particularly in the ISO 9000 standard series. Companies meeting the standard requirements can have their quality assurance systems certified by recognised certification organisations. The ISO 9000 standard series or quality assurance systems find increasing application in the cement industry, in respond to corresponding demands from the market or from product conformity certification schemes (see report MA 91/3866/E).

5.2 Organisation of Quality Assurance

It is the responsibility of the plant managers to supervise, on the basis of operating reports and statistics, the suitability and reserves of the raw materials, the quality of intermediate and final products as well as the conformance of the latter with valid and relevant standards. For these tasks, the plant manager requires an organisation for quality assurance to establish the required information and data, and thus to prepare proposals in the context of quality assurance on which the plant manager can base his decisions.

There are several ways of organising quality assurance: again, the right choice is strongly dependent on the significance of quality assurance for the company.

[1] The term quality assurance system is not defined in ISO 8402. The term quality system is defined in ISO 8402 as the organisational structure, responsibilities, procedure processes and resources for implementing quality management.

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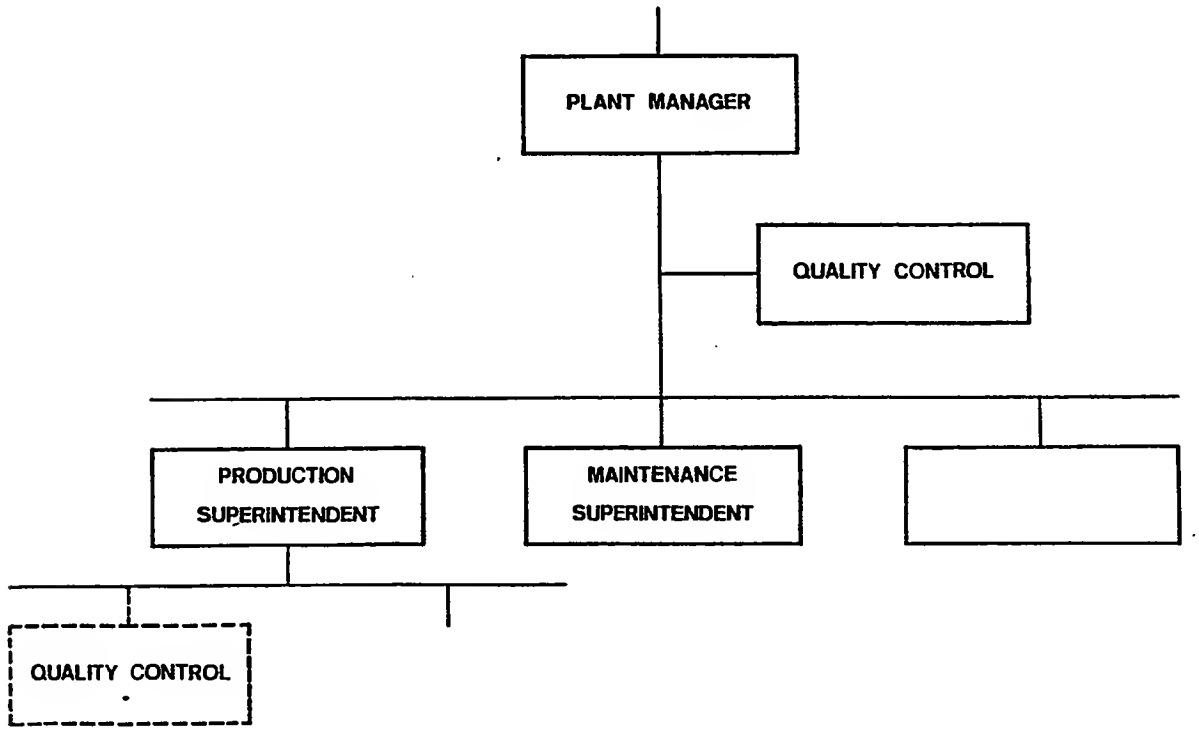
The personnel from the chemical and physical laboratory may be placed in an advisory position (staff function) to the plant manager and is merely responsible for the technical execution of quality control (Fig. 5). Where the aspect of quality control is mainly connected with the process of cement manufacture, a form of organisation is often established in which the production superintendent is directly responsible for the laboratory. These kinds of organisation have, however, the consequence that practically no direct decisions can be taken by the organisation which carried out the actual quality control. All decisions and instructions have to come from the plant manager and this bears the danger of an overburdened and inflated plant management. Moreover, it hinders communication and direct co-operation of the quality control department with other departments which have a strong relation with quality assurance. Consequently, this kind of organisation may serve the needs in a situation where material problems in production are not too complicated and when only a limited product range is supplied to a relatively uncritical market.

Fig 5 "Low Profile" Quality Assurance



Otherwise, the situation implies usually that the scope of the company's quality assurance is split so that various parts of the organisation are actively involved (Fig. 6). The plant laboratory is only concerned with the aspects of quality control in production; in addition to this, a separate group is established to cope with the quality problems directly associated in the application of cement. Such a group for technical customer services is often helpful when a company runs several plants - it is usually part of a company's technical planning and marketing divisions respectively. This form of organisation certainly offers the advantage that in the individual fields of clinker/cement production and of technical consultation of customers, a high level of expertise may be achieved with a limited number of specialists. On the other hand, the following problematic aspects have to be considered:

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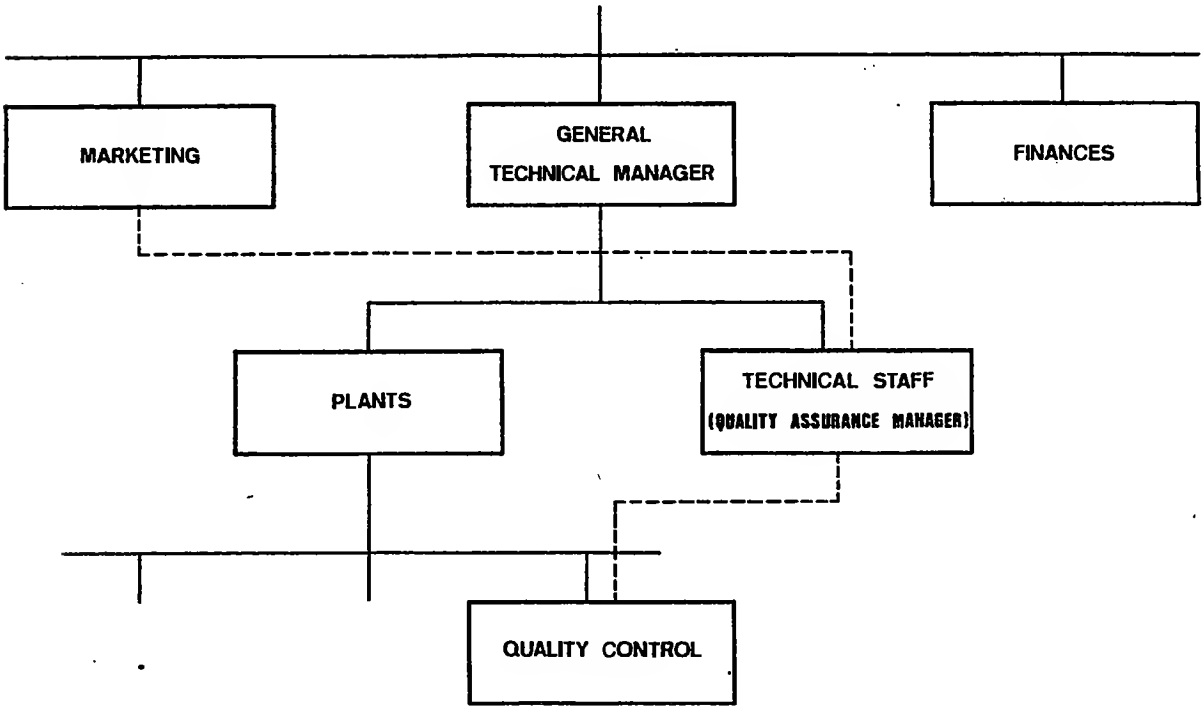
Fig 6 Quality Assurance Organisation in Multi-Plant Company



- An optimum reflection of the entire system of quality assurance from raw materials through the process to cement application is lost. As a consequence of such a partial consideration and the exclusive activity as a service organisation, the interest and comprehension and thus motivation and initiative of the people responsible for actual quality control may be strongly reduced; it thus remains questionable if a continuous optimisation of production and product will still take place.
- Without an organisation which is in a position to consider all relevant aspects of cement manufacture and cement application, there is an inherent danger that the producer's interests are not sufficiently protected against the users' requests on product specifications (e.g. when testing and product specifications are established).

The aim of delegating responsibilities and competence to those who are fully dedicated to quality assurance leads to the assignment of a superintendent of quality assurance within the line organisation of the plant (Fig. 7). Today, this type of organisation is widely applied, most functions of quality assurance are assigned to the quality superintendent who is also responsible for the laboratories.

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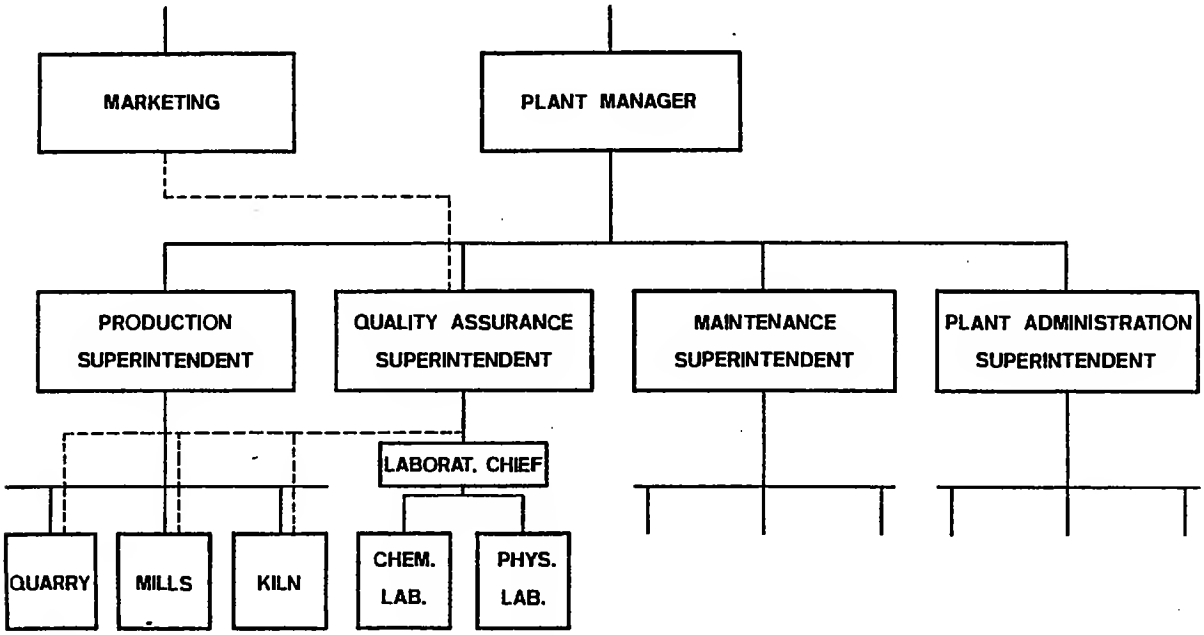
Fig 7 Typical Quality Assurance Organisation



He reports directly to the plant manager and assists the latter in all matters concerning quality and quantity of raw materials. His further tasks can comprise:

- to advise the production superintendent
- to examine the final products
- to handle technical problems
- to advise plant management and production superintendent on environmental aspects
- to participate in standardisation committees

Before deciding on the one or the other form of organisation for quality assurance, the situation should always be thoroughly analysed. A selection of the organisational form purely on the basis of tradition ("because it always had been this way") is definitely not adequate. Also preferences and dislikes of available personnel should not become decisive factors when organising a company's quality assurance. To achieve an organisation with an optimum efficiency not only in the short but also in the long run, it is essential

- to define clearly the company's tasks of quality assurance, considering prevailing and possible future conditions of raw materials, process and market,
- to formulate duties and responsibilities as well as authority for the personnel in charge of quality assurance (job description, performance standards).

From this, clearly defined requirements as to the qualification of the personnel for quality assurance and hence to the training scheme according to their tasks, can be derived.

Depending on the level of responsibility for quality assurance, this training scheme should not only be limited to techniques of quality control (sampling, testing, data processing), but should also include sound experience in the entire field of cement manufacture and application. Moreover, in addition to the professional skill, basic knowledge and understanding of economic principles and relations is required for the superintendent of quality assurance to achieve the primary goal, namely to make a profit on quality. Being a leader of an organisational entity, he also requires training in the field of leading personnel.

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5.3 Quality Assurance Handbook

It is strongly recommended that for every company the quality assurance system and generally all features related to quality assurance are compiled and documented in a "Quality Assurance Handbook". This handbook should be of a binding nature and serves to ensure transparency and efficiency in quality assurance. Although primarily intended for plant internal purposes, the awareness of its existence by customers or official bodies can only be beneficial to the company's quality image.

In order to meet the requirements of national or international standards on quality assurance systems, a comprehensive quality documentation needs to be established. It usually comprises two or three document levels, i.e. a top level quality manual, mid-level quality procedures and lower level work instructions (see Report MA 91/3866/E).

6. QUALITY COSTS

6.1 Introduction

"Quality assurance: from outlay to profit"

It is increasingly recognised that quality assurance is no longer primarily - though inevitably - a cost generating nuisance but represents a valuable instrument to increase profit.

Quality assurance no doubt creates costs - as does any other activity within an enterprise - but, on the other hand, it also allows to reduce unnecessary costs which may arise in case of inadequate quality of services or goods produced.

To ensure the economic benefits of quality assurance, the costs referred to above should be assessed and presented in a transparent way. This is ultimately the most effective way to convince the top management of the benefits of quality assurance, and to obtain their support in establishing and maintaining an integral company policy for quality assurance.

The purpose of assessing the quality costs of a company can be summarised as follows:

a) the quality cost assessment allows to recognise and to rank weak points. It makes evident what kind of deficiencies generate the highest (additional) costs at what stage of production. Measures to improve quality and quality control can be implemented accordingly. Also, sectors apparently causing very high control costs can be re-assessed as to whether the risks justify the costs.

b) quality costs serve as basis for a cost/benefit calculation in connection with planned investments for production installations and for the instruments of quality control. Also the consequences of changes in personnel structure can be estimated on this basis.

c) quality costs can represent a motivation at all levels to increase the efficiency of quality assurance

d) quality costs may permit a comparison with similar enterprises. This, however, requires due precaution because either the assessment of quality costs or the means of production or both may be different.

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6.2 Elements of Quality Costs

Quality costs are in general terms defined as the difference between the real expenditures and revenues and those which would arise, if no quality related failures during product development, manufacture and marketing would and could occur.

It is generally accepted that quality costs should be subdivided into three elements:

QUALITY COSTS
Prevention Costs + Appraisal Costs + Failure Costs

Applied to the cement industry, these cost categories can be characterised in the following way:

6.2.1 Prevention Costs

These basically comprise the costs arising in connection with quality planning as outlined in chapter 4.1.

Cost compilation may be unproblematic as far as it concerns the QA-internal personnel costs. For the investigations involving other departments - which is often the case - the fraction of "quality costs" is difficult to determine and would have to be estimated.

6.2.2 Appraisal Costs

These comprise all expenditures of quality control:

- personnel costs
- costs of accommodation and infrastructure
- purchase and amortisation of equipment
- consumables

These costs can easily be ascertained.

6.2.3 Failure Costs

Failure costs are additional costs or non-realised revenues arising as a result of inadequate quality ("non-quality") of intermediate or final products. They can have most different origins as documented by the following list which is by no means exhaustive:

- undue consumption of expensive corrective material on account of non-optimal mix design
- elevated heat consumption, reduced production rate, or excessive refractory wear due to inappropriate raw mix quality
- increased cement grinding energy (higher fineness) due to inadequate clinker quality
- separate storage and blending of under-burnt clinker - . addition of costly "quality improvers" upon cement grinding when using inferior (e.g. outdoor stored) clinker
- use of off-specification special clinker as ordinary clinker
- treatment of customer complaints; payments for damage in cases of failure in cement application
- sales losses owing to manufacture of not market-conform products or due to loss of confidence after a case of damage.

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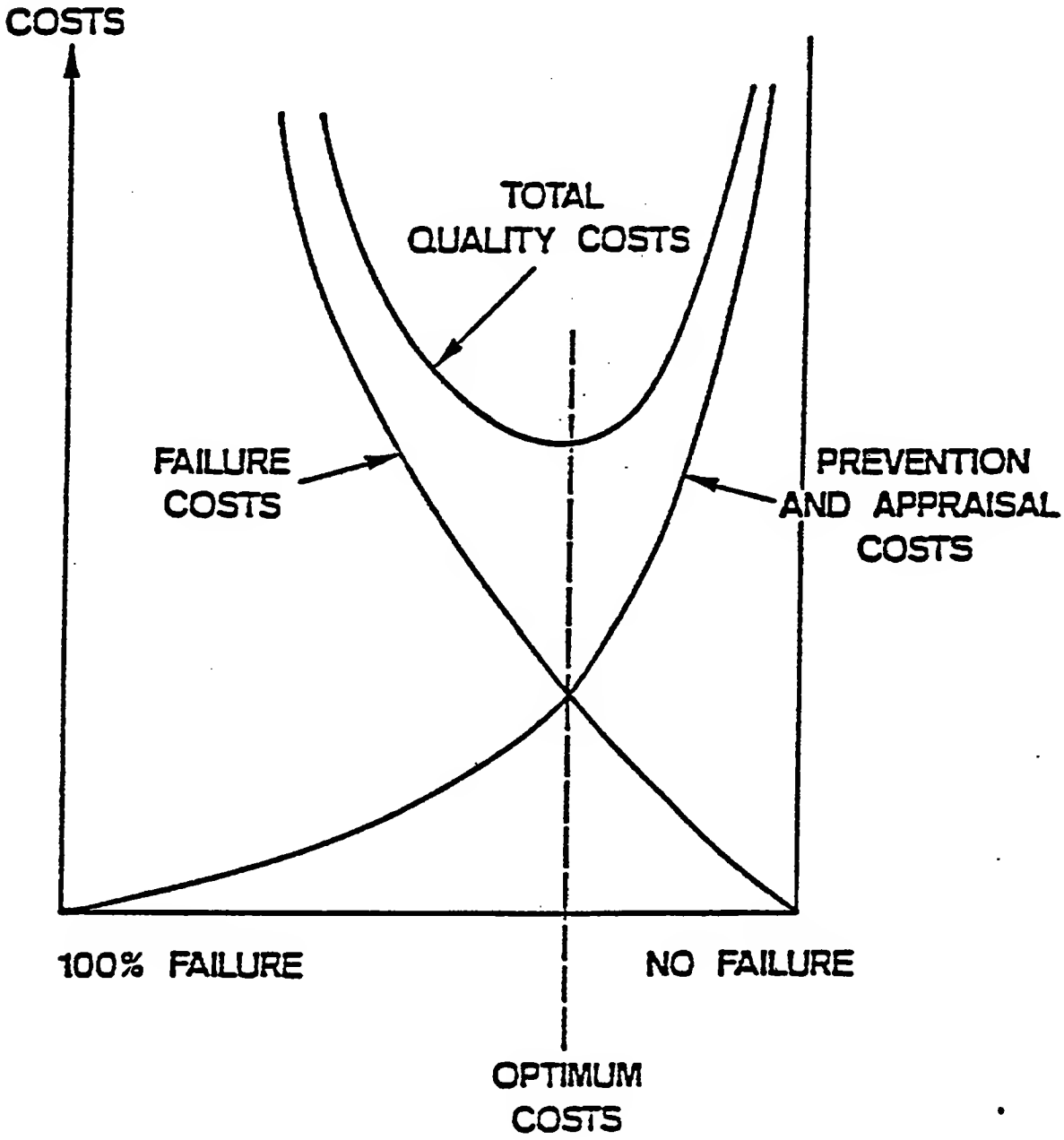
It is evident that a comprehensive quantitative assessment of these failure costs is very difficult.

The relation between these three cost elements is shown in figure 8.

Fig 8 Relationship between Quality Cost Elements



6.3 Relationship of Quality Costs to Sales

Reliable figures for quality costs in the cement industry are not available, mainly due to the lack of a quantitative assessment of failure costs. Reasonable estimates of the expenditure for personnel, equipment and infrastructure associated with quality assurance, which may be taken as a rough approximation of the prevention and appraisal costs, reveal figures in the order of 1 to 2% of the cement net sales price. Sound estimates of the failure costs ("non-quality" costs) are not available; ad-hoc estimates range between 5 and 20 % of net sales price

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For comparison, some figures related to other industries for quality costs and improvements achieved after optimisation are given in figure 9.

Fig. 9 Quality costs in different industries before and after optimization, expressed as a percentage of turnover.

Industry	before	after
motor cars	6- 9	4-6
chemicals - raw materials	5- 8	3-5
food	7-12	3-6
chemicals - complex products	7-12	4-7
ceramics	15	10
overall average	11	6

The different types of quality costs before, during an after successful optimisation, expressed as a percentage of turnover.

	Before	during	after
prevention	0.5	1.5	1.0
appraisal	3.5	3.0	2.5
consequential	7.0	4.0	2.5
total	11.0	8.5	6.0

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Quality Systems in the Cement Industry according to ISO Standards

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P. Bürki, HMC/TC/Materials Division

Formalized and documented quality systems are increasingly applied in the cement industry, often applying the models given in the ISO 9000 standard series. The article addresses the internal and external reasons for this trend and outlines the basic features of ISO 9000 and the criteria for an appropriate documentation. Examples of application and documentation's from 'Holderbank'. Group plants are discussed.

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1. INTRODUCTION

The cement industry has been traditionally subjected to an extensive product standardization. The compliance with these standards formed the only generally valid quality liability towards the consumer. The way in which the products are produced and in which the quality is controlled has been entirely up to the producer.

In many other industries, another method of quality commitment in the supplier/customer relationship has been adopted: in place of product standardisation - which is practically impossible for products undergoing rapid developments - the manufacturer has to provide evidence that he disposes of o quality system that enables him to plan and realize a state-of-the-art quality level throughout the production cycle of the product. Efforts to find an internationally agreed upon concept of quality systems and a generally accepted terminology eventually resulted in the ISO 9000 standard series which were approved in 1987, and which were adopted in many national standards as well as in the European standards (EN 29 000). It is a fad that today many of the supplier/ customer relationship in Europe are based on these standards.

2. REASONS FOR IMPLEMENTATION OF DOCUMENTED QUALITY SYSTEMS

Various reasons taken individually or in combination, can lead a company to implement a quality system along with a corresponding documentation.

2.1 Internal reason

2.1.1 Confidence, transparency efficiency.

A well documented quality system provides transparency and confidence with regard to all quality related activities, both to those directly involved in production, marketing, and quality assurance, and to the company management

2.2 External reasons

2.2.1 Market.

Customers can require the existence of a (certified) quality system as a prerequisite for a delivery contract. This particularly applies for large customers or for customers in the public sector.

2.2.2 Product liability

For the protection of the individual private consumer, fault independent regulations for damages caused by defective products are increasingly being introduced. While such consumer oriented product liability legislation has been established in the USA for quite some time, similar laws have been, or are being introduced in Europe following the guidelines issued by the EC - Commission - as well as in other parts of the world.

A documented quality system and the availability of quality data records can be considered as effective preventive measures for avoiding the release of defective products and are moreover valuable evidence for the defence of a producer in case he is involved in a product liability litigation.

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2.2.3 Certification of products conforming to standards.

There is a trend largely promoted by consumer-oriented organisations - that the conformance to standards of a given product should be certified by official authorities (certification body, approved body). The scheme for certification may not only involve external product control, but also the requirement of a documented quality system. This aspect may become the <<driving force>> for implementation of standard quality systems based on national or international standards (e.g. ISO 9000) in the European cement industry, as further outlined in the subsequent chapter.

3. IMPLEMENTATION OF DOCUMENTED QUALITY SYSTEMS IN THE CEMENT INDUSTRY

3.1 General

Until recently, there was little external pressure for establishing documented quality systems in the cement industry. Nevertheless, a number of plants have established a company-wide quality system and documentation on a volunteer basis, with a view to improve internal transparency and efficiency and/or to use it for promotional purposes to gain a competitive advantage.

Increasingly, however, external factors are pushing the cement industry, at least in certain regions, to adopt documented quality systems, practically exclusively according to ISO 9002:

The British Cement Industry has subjected itself to a certification scheme by the British Standard Institute (BST). Labels for <<quality assured cement>> are granted to the products of cement manufacturers certified on the basis of a ISO 9002 quality system.

In Australia, the cement and construction industries are subject to the generally valid state regulation that <<all firms supplying manufactured goods to (governmental) agencies, must have adequate quality assurance systems...>>. In practice, again, the ISO 9002 system is adopted.

Very recently (1991), API—the American Petroleum Institute—has introduced the demand for a documented quality system (Q1) similar—though not identical—to ISO 9002 for granting their monogram on well cement.

3.2 Specific European Situation

In the context of the European cement standardization, (EN 196; ENV 197) a scheme for the certification of conformity of cement complying with the requirements of ENV 197 has been proposed (document entitled <<European Cement Certification>>). The proposed scheme takes account of, and refers to two EC regulations, namely the Construction Products Directive" and the <<Global Approach to Certification and Testing>>, which attempt to harmonize within Europe the regulations, the standards and the means for verifying compliance with standards.

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The European Cement Certification scheme defines the tasks of the manufacturer and of approved bodies, in order to obtain product certification. Among the tasks of the manufacturer belongs the implementation of a quality system based on EN 29002 (ISO 9002), to be documented in form of a quality manual. It is foreseen in the scheme, that approved inspection bodies regularly inspect the production facilities, quality control procedures and testing equipment with a view to approve and verify the procedures defined in the quality manual. This implies a considerable intensification of external control for assuring conformity to standards, which has so far been largely based on auto-control and external control of finished products. It has to be pointed out that this certification scheme is still much debated and an approval in the majority of the member countries not yet in sight.

It is noteworthy that Spain has adopted in form of national standards, not only the draft EN cement specifications (ENV 197), but also a certification scheme largely identical to the proposed <<European Certification Scheme>>.

4. PRINCIPLES OF THE ISO 9000 QUALITY SYSTEM

The ISO standard series 9000 on quality systems has become effective in 1987. It describes models and elements for quality systems in different contractual situations and for the internal quality management.

ISO 9000:	Quality management and quality assurance standards—Guidelines for selection and use.
ISO 9001:	Quality systems—Model for quality assurance in design, development, production, installation and servicing
ISO 9002:	Quality systems—Model for quality assurance in production and installation
ISO 9003:	Quality systems—Model for quality assurance in final inspection and test.
ISO 9004	Quality management and quality system elements—guidelines

For the purposes of the cement industry, the ISO 9002 is usually applicable. The main points of emphasis of these standards can be summarized as follows:

- Company commitment to quality; quality responsibility on top management level.
- Clear definition of quality relevant procedures, including organisation and responsibilities, throughout the production process and the various departments involved
- Clear definitions of measures in case of non-conformity to quality targets
- Training of personnel
- Requirement for quality audits
- Implementation of a continuously updated quality documentation.

The ISO 9001-9003 Standards for quality systems explicitly describe models for quality assurance. Their application has to be made company (or industry) specific.

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5. ASPECTS OF QUALITY SYSTEM DOCUMENTATION ACCORDING TO ISO 9002

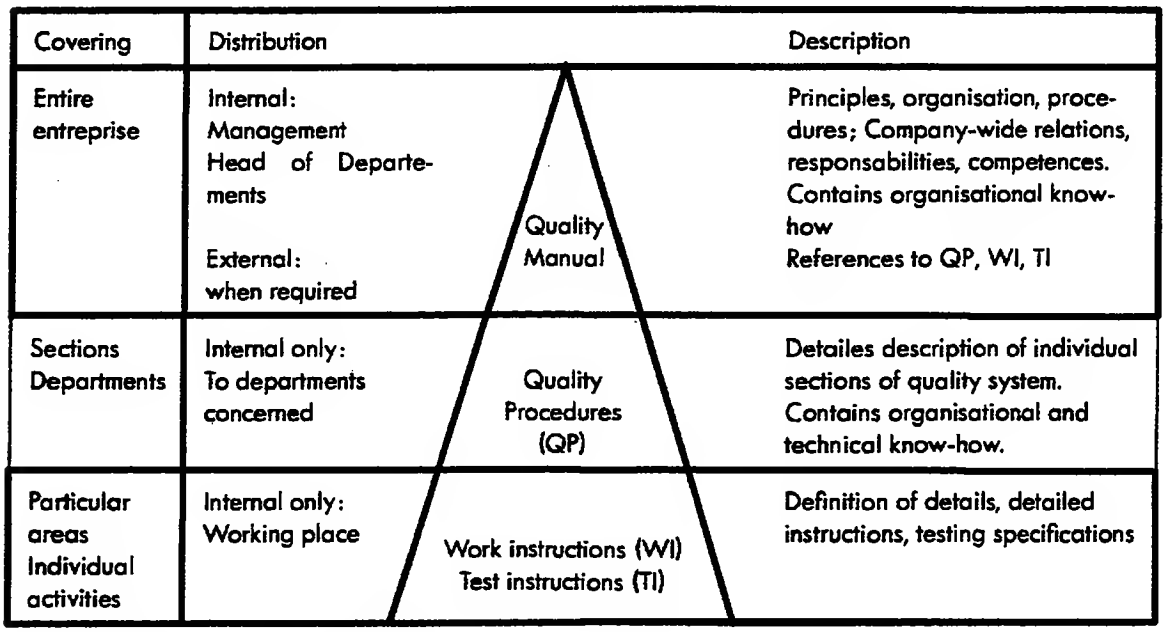
5.1 General

A quality system has to be documented in order to make it visible, applicable, and auditable. The ISO 9000 explicitly demand the establishment of a quality system documentation.

It is recommended to apply the <<pyramid structure>> for the organisation of the quality system documentation (Fig. 1). Usually, three document levels are distinguished:

- Quality Manual
- Quality Procedures (for Quality Guidelines)
- Work instruction and test descriptions

Fig 1 Structure of Quality System Documentation



(Source: SAQ, Leitfaden zur Normenreihe SN-ISO 9000 Bern 1989)

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The <<pyramid-structure>> offers the following advantages:

- The quality manual may be kept relatively short. It provides a transparent guideline to the concepts of the quality system, its organisation and procedures. It provides the connections to the other document levels by referencing.
- The quality manual largely covers the organisational aspects of the quality system and may thus be made accessible to externals (customers) without disclosing technical know-how.
- The quality manual is not subject to frequent modifications.
- On the workshop level, only parts of the quality procedures and works instructions are required, which can be selectively distributed.

When applying the ISO 9000 standards, the structure of the quality manual should, but need not, follow that of the standards, i.e. the list of contents should correspond to that of the corresponding standard. If a different structure is applied, which may be better adapted to the company's requirements, a correlation has to be made with the clauses of the standards.

For the administration of the quality manual and all other quality system documents, it is advisable to

- use loose sheets to facilitate exchange of pages or chapters
- indicate date and stage of revision on each page or chapter number the pages only within a chapter
- to have a control on the internal and external document distribution.

6. EXAMPLES OF ISO 9000 QUALITY SYSTEMS IN 'HOLDERBANK' GROUP COMPANIES

6.1 Queensland Cement Ltd. (QCL) Australia

(see article on page 38, this issue)

6.2 Cementos Boyaca (Columbia)

The implementation of a company-wide quality system at Cementos Boyacà started some four years ago, on an entirely voluntary basis. The reasons were, on one hand, a recognised lack of efficiency and of personnel motivation in most sections of the plant, and on the other hand, the objectives of the Company in the frame of PMI (Project Marketing Innovation) and TOC (Total Offer Concept), necessitating a market and quality orientation of the company. As a definite target and motivating factor to improve the company-wide quality assurance, the attainment of the national quality award <<Premio Nacional de Calidad>> was aimed at. This goal was reached in 1991, when the quality award was granted to Cementos Boyacà. (see 'Holderbank' NEWS 4/91, p.5).

6.3 HISALBA

As already mentioned in chapter 3.2, the Spanish national standard is very close to the proposed <<European Certification Scheme>>. The cement plants of the HISALBA group, therefore, have certified quality systems according to ISO 9002. A comprehensive information on his topic was recently published by HISALBA (Calidad y Automatization HISALBA).

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6.4 Groupe Origny (France)

In the context of the French scheme for approving cement trade marks (AFNOR), all plants of the Origny group have had to dispose of quality manuals (<<plans qualité>>) for many years to give evidence of existing quality systems. The requirements on the manuals have not been very high; the document could be kept very short and had admittedly little significance for the practical day-to-day quality assurance in the plant

Recently, the Groupe Origny took the decision to reelaborate these <<plans qualité>> in such a way that they would become practice relevant, binding working tools. Among others, the following objectives were put forward:

- Quality improvements of final and intermediate products should be achieved by increased efficiency, at minimum investment costs. Quality deficiencies should, without exceptions, be detected (and eliminated) internally and not by customers.
- Preparation for the anticipated higher requirements on quality systems and documentation within the future European Certification scheme (ISO 9002).
- Maintenance of company image and market position in view of similar efforts to implement documented quality systems being made by competitors

With these objectives in mind, Origny made the interesting approach to describe in detail all quality relevant procedures from quarrying to cement dispatch, by applying just two types of formalized sheets. These sheets contain instructions on routine procedures, frequencies of sampling and testing, targets, acceptance and alarm limits, instruction in case of exceeding the limits, etc. and are distributed to all concerned and continually updated in order to strictly conform with actual practice at all times. About 90 sheets were required to cover the entire process of the Origny plant.

Origny intends to incorporate the entity of these sheets in the future quality documentation as required by ISO 9002.

6.5 Conclusions

From the yet limited experience within the 'Holderbank. Group'—and the cement industry in general—it is an undisputed fact that the implementation and documentation of a quality system according to ISO 9002 is an immense task. It has also become evident that an implementation can only be successful (and durable) with the uncompromising support of the company management.

The opinions regarding the benefit of such a system are still divided. The main criticism expressed by the sceptics is that the written procedures are, in the best case, declared intentions, over-emphasising in their very nature the quality aspects of cement manufacturing. In practice, it is claimed, pragmatic considerations of other production or cost related aspects at a given situation and point in time, may result in decisions not congruent to the instructions contained in the quality documentation. Furthermore, it is pointed out that it is questionable whether the considerable effort will always be made to update the system and the documentation in the course of changes arising in the process, in the materials used, and in the organisation of the company.

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On the other hand, the astonishingly good acceptance in companies where such systems have been introduced, is quite remarkable. The benefits often put forward better organisation, clarified responsibilities, well defined decision criteria etc. resulting in better plant operation, improvements of intermediate and final product quality and hence in strengthened market position—indicate that significant weak-points in these respects exist in many companies. It thus appears that the detailed scrutinizing of all activities and procedures which is mandatory for establishing a documented quality system, is by itself already very valuable, as many deficiencies can already be detected and rectified at this stage. Attention is also drawn to todays more rapid fluctuations in personnel in the context of which well-documented procedures and instructions serve as most valuable training tools.

The experience is not yet sufficient to permit definite conclusions on the above argument The present trend, however, is in the direction of increased application of ISO 9002 standards as a consequence of market or standard requirements. It is therefore advisable that the companies closely follow the developments and start their preparation in time. In a situation where the implementation of a complete ISO 9002 quality systems is not yet very urgent, a stepwise elaboration starting on the work instruction level is a recommendable practical approach, as it was carried out by Origny. The ultimate application of a complete ISO 9002 quality system will, in many cases, not be a question of choice but of survival.

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Chapter 9

Quality Control: Concept, Control Plan, Procedures

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Quality Control: Concept, Control Plan, Procedures

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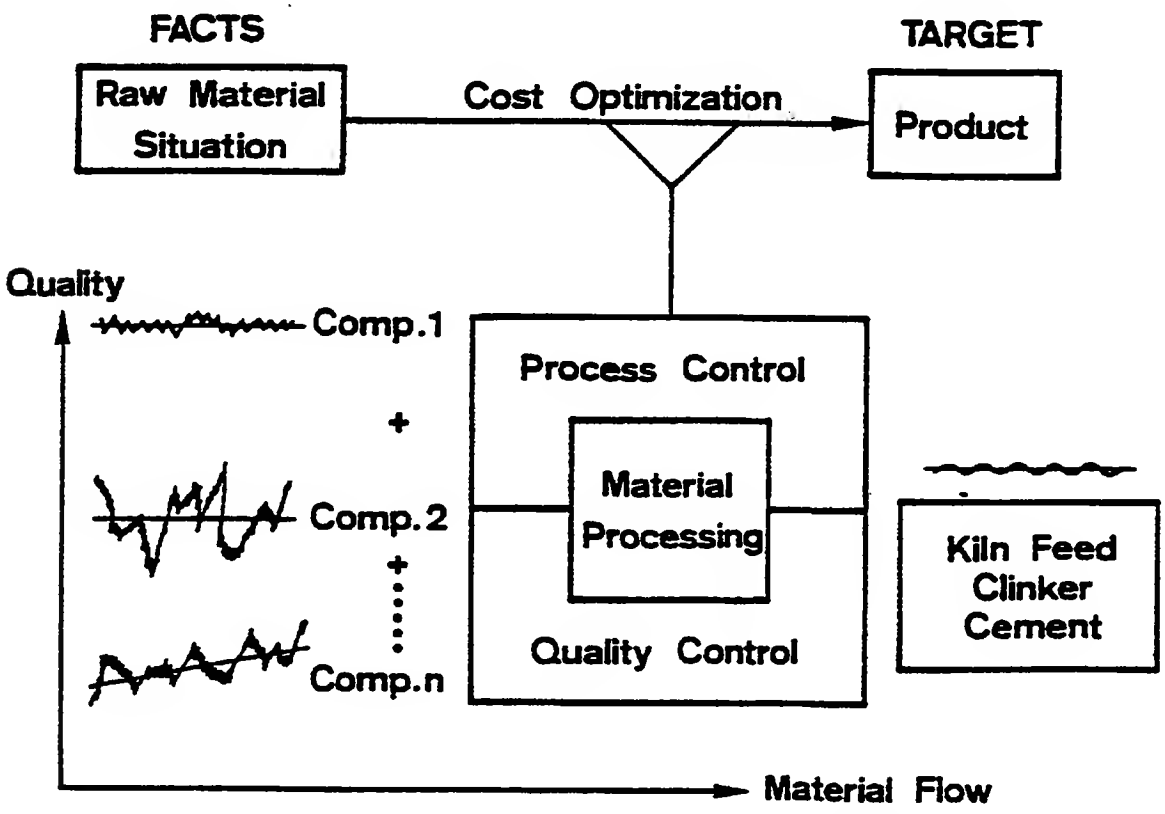
1. INTRODUCTION

The principle aim of this paper is to highlight aspects concerning quality concept, plans and procedures that meet with today's process and market requirements. It is certainly not possible to make a generally valid statement as to the most suitable way of carrying out quality control and to define the most adequate degree of sophistication and automation for every plant. This must be part of the quality planning in each individual company and plant.

2. QUALITY CONTROL CONCEPT

Quality control normally complements the process control, with the aim being to monitor the product quality at the different production stages, so that the required quality level is finally achieved.

Fig. 1 Quality Control.



Tested properties may include chemical composition, moisture, granulometry as well as other properties relevant to the specific material.

Special procedures are being applied for the delivery control of combustible or non combustible waste materials.

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3. CONTROL PLAN

The control plan shows how the general control concept for each raw material, intermediate and final products is to be realised.

A detailed control plan for quality control is set up by a thorough analysis of the following questionnaire:

- What should be examined?
- What information is required for the control?
- How often must the test be performed?
- How accurate must the result of the testing be?

In every particular plant, the concept of quality control, with its objectives and targets, its detailed programme of testing , its technical features of the control system and its decision criteria should be carefully formulated in written and graphical form.

Thereby, the quality control procedure is clear, and adaptation to changed situations, whenever necessary, is facilitated. The enclosed figures 2, 3 and 4 are examples of such documentation.

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Fig. 2 Inspection and Testing Plan.

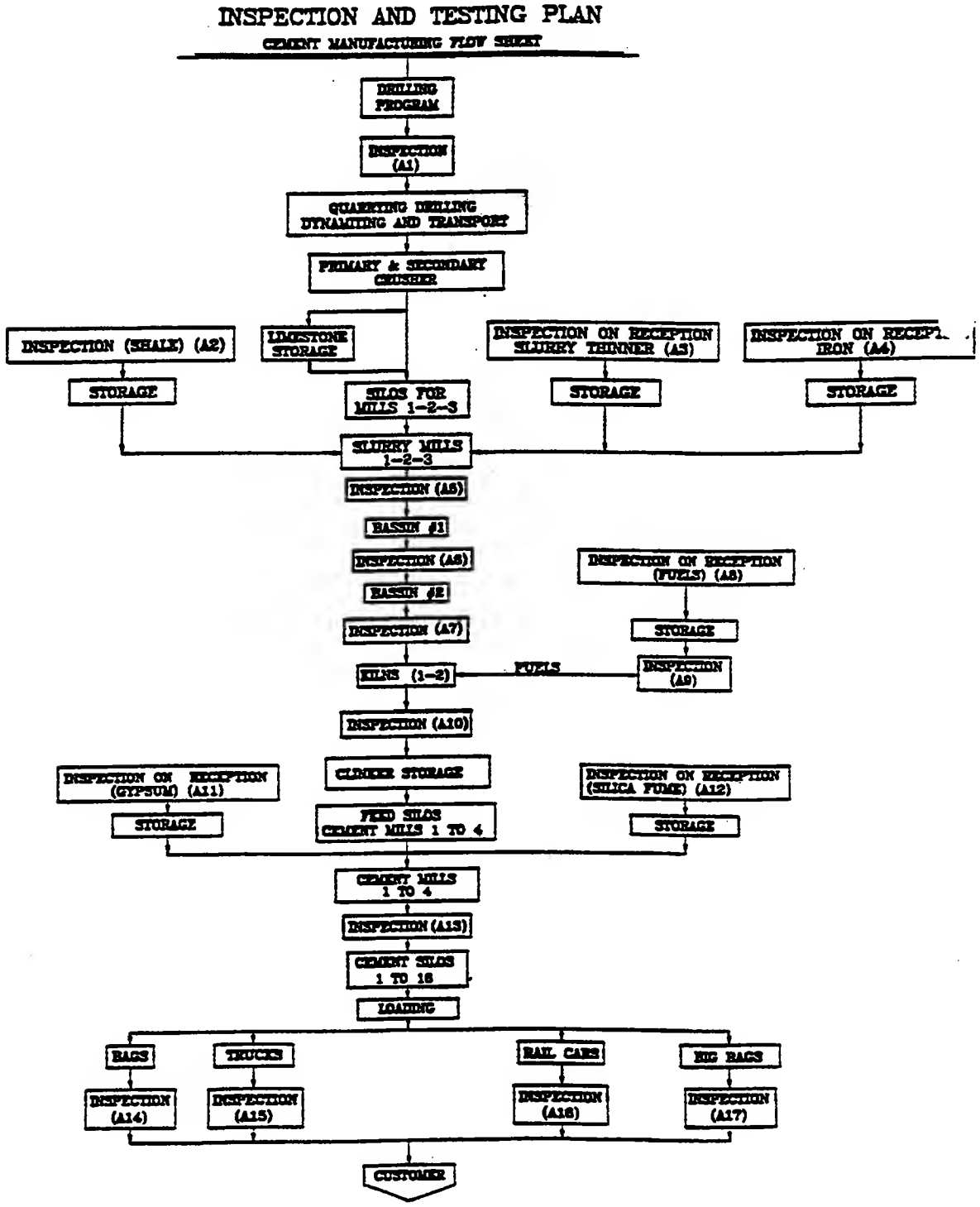

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Fig. 3 Quality Control Plan (extract).

No	Material	Sampling Point	Sampling	Sampling Frequency	Sample Transport	Test Frequency	Tester	Tests	Data Processing	Objective
8	Gypsum	Trucks (ship arriv.)	spot	1/y	AdD	1/y	AdD	SO_3, insoluble, humidity		Quality assurance
9		Cement mill	spot	1/15 d	MdT	1/15d $\bar{x}$/m	AdD AdD	Humidity SO_3	SR SR	Prod. control Quality assurance
10	Raw meals	before elevator to silos (M1, M2)	screw samp. with mixer	1/h	MdT	1/h 1/2 h 1/4 h 1/8 h $\bar{x}$/d	MdT AdT AdT AdT AdD	R150µm, R75µm XRF-tablett (LS, SR, AR) Titration Humidity XRF-tablett, humidity, LOI	SR SR SR SR DSR, MR, MSR	Mill control Blending contr. Prod. control Quality control
11	Kiln feeds	before elevator to kiln (H1, H2)	screw samp. with mixer	2/h	MdT	1/2 h 1/4 h $\bar{x}$/d	MdT AdT AdD	R150µm, R75µm, XRF-tablett (LS, SR, AR) XRF-tablett, LOI; kiln 2: XRF-perl	SR SR DSR, MR, MSR	Quality control Quality control Quality control
12	Hot meals	lowest cyclon (H1, H2)	spot	2/w	MdT	2/w	AdD	XRF-tablett (Cl, Alk,SO_3) LOI	SR	Process control
13	Clinkers	Kiln outlet (H_1,H_2)	spot	1/h	MdT	1/h $\bar{x}$/8h $\bar{x}$/d	MdT AdT AdD	Literweight Free lime XRF-perl, free lime	SR SR DSR, MR, MSR	Process control Process control Quality control

Fig. 4 Plan Qualite.

SOCIETE : / USINE :	PLAN QUALITE					Feuillet n° 4.7.2
ATELIER	NATURE PRODUIT	VENANT DE	ALLANT VERS	Date d'effet du feuillet : OCTOBRE 1989		
BROYAGE CIMENT	CIMENT	BROYEURS C1 OU C2	SILOS DE STOCKAGE CIMENTS FINIS	Annule et remplace le feuillet n° du :		

* PRELEVEMENT	RELEVE	VISITE	ENDROIT	FREQUENCE	QUANTITE	RESPONSABLE
MANUEL			AVANT PNEUMEX	TOUTES LES 2 HEURES SUR CHAQUE BROYEUR	1 LITRE	MEUNIER

CONTROLES	OPERATEUR	NATURE	METHODE	SEUILS LIMITE MINI	Seuils alerte MINI	Seuils alerte MAXI	SEUILS LIMITE MAXI	OBJET	SUPPORT
	MEUNIER	SURFACE SPECIFIQUE "BLAINE" (SSB REPRESENTE LA VALEUR DE CONSIGNE FIGURANT POUR CHAQUE QUALITE SUR LA FEUILLE "REGLAGES BROYAGE CIMENT")	20	SSB - 200	SSB - 100	SSB + 100	SSB + 200	REGLAGE DU BROYEUR POUR L'OBTENTION DE LA SURFACE SPECIFIQUE "BLAINE" DE CONSIGNE	FEUILLE DE MARCHE DU BROYEUR

ANOMALIES	DEVIATION - sortie des seuils d'alerte - PREVENIR	ANOMALIES	ACTIONS	DEFAUTS - sortie des limites - PREVENIR	DEFAUTS	ACTIONS
		SURFACE BLAINE	MODIFICATION REGLAGES SEPARATEUR ET/OU BROYEUR.		SURFACE BLAINE	1)MODIFIER REGLAGES SEPARATEUR ET/OU BROYEUR 2)PRELEVEMENT D'UN AUTRE ECHANTILLON APRES : UNE HEURE AU C1 UNE DEMI-HEURE AU C2 - SI CONFIRMATION SORTIE DES LIMITES: 3)PROCEDER A VERIFICATION COMPLETE DE L'INSTALLATION(SEPARATEUR,VENTILATION) 4)POURSUIVRE PRELEVEMENTS ET MESURES PERIODIQUES DE SURFACE "BLAINE". LIMITATION A 3 HEURES DU TEMPS DE MARCHE DE L'INSTALLATION DANS DES CONDITIONS ANORMALES

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4. GENERAL PROCEDURES AND CONTROL PARAMETERS

The following sections give a general survey of the most important control positions for raw materials, intermediate and final products in a cement plant (emphasis on dry process). Reference is being made to the schematic plant layout in figure 5.

Fig. 5 Cement Plant - Material Flow.

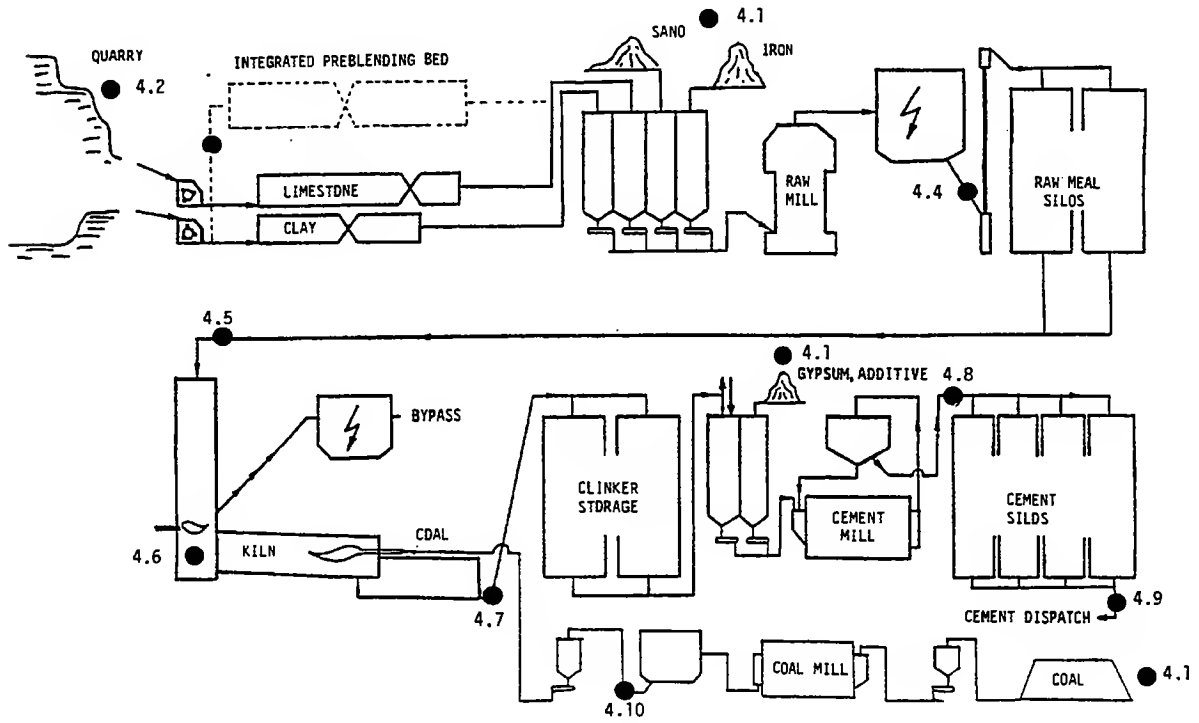

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Fig.6 Sampling Tower

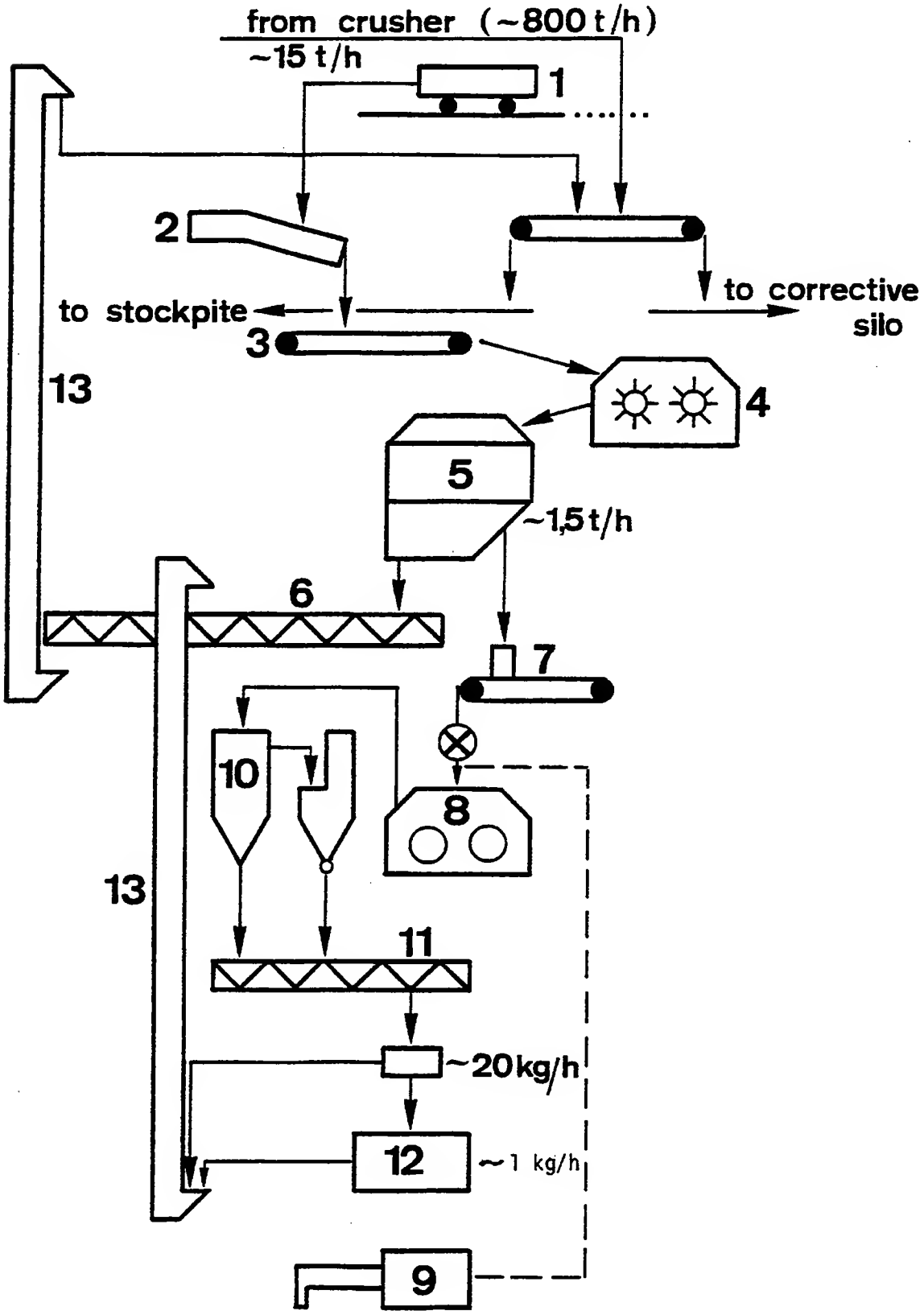

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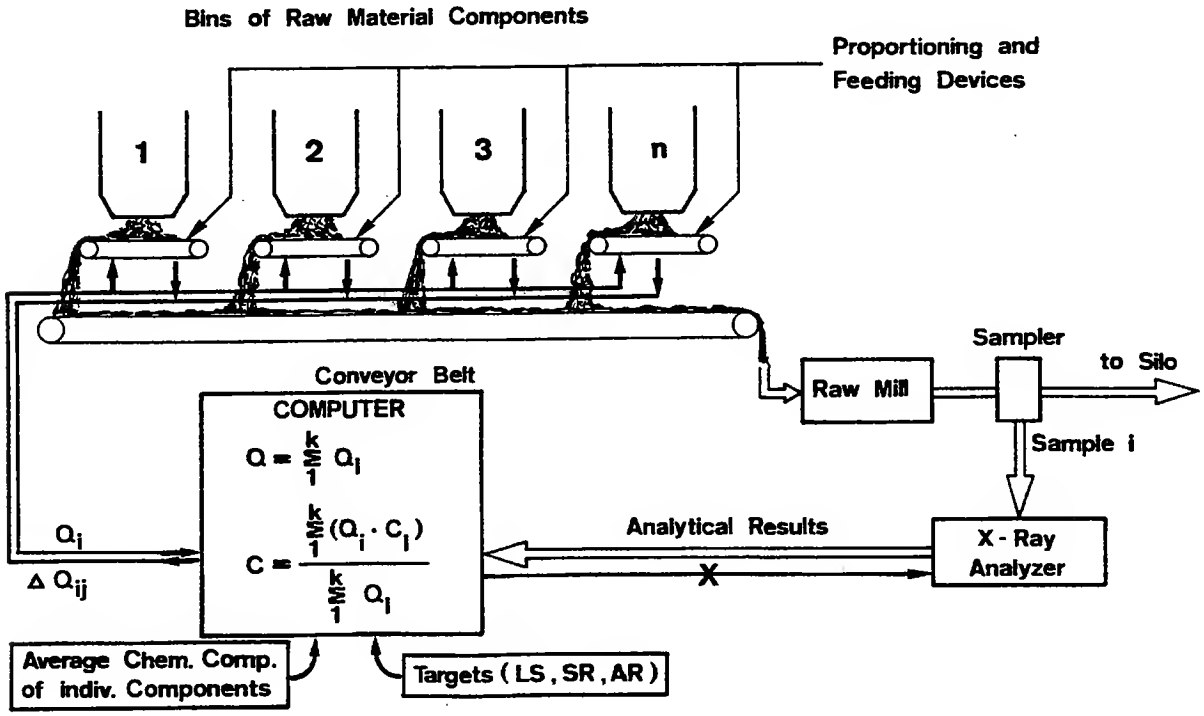
Fig.7 Automatic on-line Control of Raw Mix Composition



In a concise form, the purpose of control, typical control parameter and typical techniques for sampling and testing are given, as well as additional comments where appropriate. In view of the large variety of materials involved, of plant layouts, processes, sampling and testing methods, the survey does not apply to all cases and does not attempt to be complete.

4.1 Supplied Materials

Purpose

The testing of supplied materials such as raw mix correctives, gypsum, mineral additives (slag, fly ash, etc.), combustibles, serves to verify their compliance with delivery contracts and hence their suitability for the anticipated utilisation.

Procedure

The extent of sampling and testing varies strongly, depending on type of material, its source and delivery contracts. Whenever possible, sampling should be done from conveyed material rather than from trucks or stockpiles.

4.2 Quarry

Purpose

Control of the materials composition prior to excavation enables selective quarrying to achieve

- correct composition of (integrated) stockpiles
- medium to long-term uniformity of stockpile composition
- optimum utilisation of materials and equipment.

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Control parameter

Chemical composition, often with emphasis on specific elements such as SiO_2, MgO, SO_3, alkalis, C1.

Sampling

Usually blast drill hole dust.

Process interaction

Based on manual or computerised data evaluation:

- selection of area to be blasted
- selection of (blasted) material to be fed to the crusher.

Alternatives

Information on the chemical structure of the quarry can also be obtained from analysis of crushed stone by online bulk material analysis (neutron activation technology) or mechanical sampling followed by conventional analysis.

4.3 Pre-blending

Purpose

Blending of raw components on (integrated) pre-blending stock piles to a given target composition.

Control parameter

Chemical composition, moisture.

Process interaction

Selection and dosage of material fed to the crusher.

Procedure

Today, two different concepts are applied to monitor and integrate the composition of the material fed to a pre-blending bed.

- **Sampling tower**
  Mechanical sampling of 1-2% of the materials feed and subsequent steps of sample comminution, drying where required, and splitting in a sample tower (Fig. 6), to receive typically 1 kg of fine, sample per hour for analysis.
- **Bulk material analysis**
  Continuous chemical analysis of the entire or partial materials stream by means of a bulk material analyser applying the technology of prompt gamma neutron activation analysis PGNAA (see reports MA 89/3702/E and VA 92/5975).

4.4 Raw Meal

Purpose

The raw meal control serves the following objectives:

- Blending of components to obtain the target composition of raw meal and clinker.
- Achievement of target meal fineness to obtain an appropriate clinker burnability.
- Achievement of a sufficient uniformity which, together with the homogenisation in the subsequent silo, results in a high kiln feed uniformity.

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Control parameters

- Chemical analysis, and parameters derived thereof such as lime saturation, silica ratio, alumina ratio, potential clinker mineral phases C_3S, C3A, etc.
- Fineness, usually in terms of sieve residues (e.g. on 200 µm and 90 µm sieves)
- Residual moisture content.

Process interaction

Adjustment of feed rate to the mill

off-line: manual, based on mix calculation

on-line: automatic, based on blending programme (fig. 7).

To be observed: Dynamics of grinding process (retention time, internal circulation, dead time of analysis).

Sampling

Usually, automatic compositing sampling from a chute or an airslide is applied. Standard samplers, typically screw samplers or airslide samplers are used, with or without integrated mixing facilities.

Frequency

Typically every ½ to 2 hours.

Testing

Chemical composition:	X-ray fluorescence complexometric titration carbonate titration
Fineness:	Sieving, laser granulometer

Alternative techniques

Bulk material analysis (PGNA technology) can be applied to continuously monitor the composition of the mill feed or the raw meal.

Comments

The set points of the raw meal composition have to be such that the target composition of the clinker is being obtained. Allowance has therefore, to be made for:

- the coal ash incorporation in clinker
- the kiln dust absorption in the raw mill (kiln dust composition can significantly deviate from the kiln feed composition)
- the discarding of kiln dust
- systematic errors in sampling and analysis.

The primary target is the clinker composition. If any of the above factors change, the raw meal set-point has to be adjusted accordingly.

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A practical example for a case where such changes have evidently occurred after a long kiln stop, but without any reaction on the raw meal set point, is given below.

	Plant with oil firing Monthly average values.		
	Raw meal LSF	Clinker LSF	Clinker CaO_F
December	99.1	98.1	1.4
January	98.0	96.8	0.8
February	97.8	97.1	0.9
Kiln stop			
June	96.0	99.7	2.1
July	97.0	100.0	1.9
August	97.2	100.4	2.1

As an illustration for the compositional range of industrial clinkers, the lime saturation, silica ratio and alumina ratio of the clinkers produced in the Holderbank group are graphically displayed in figures 8 and 9

Fig. 8 Clinker Composition (Alumina).

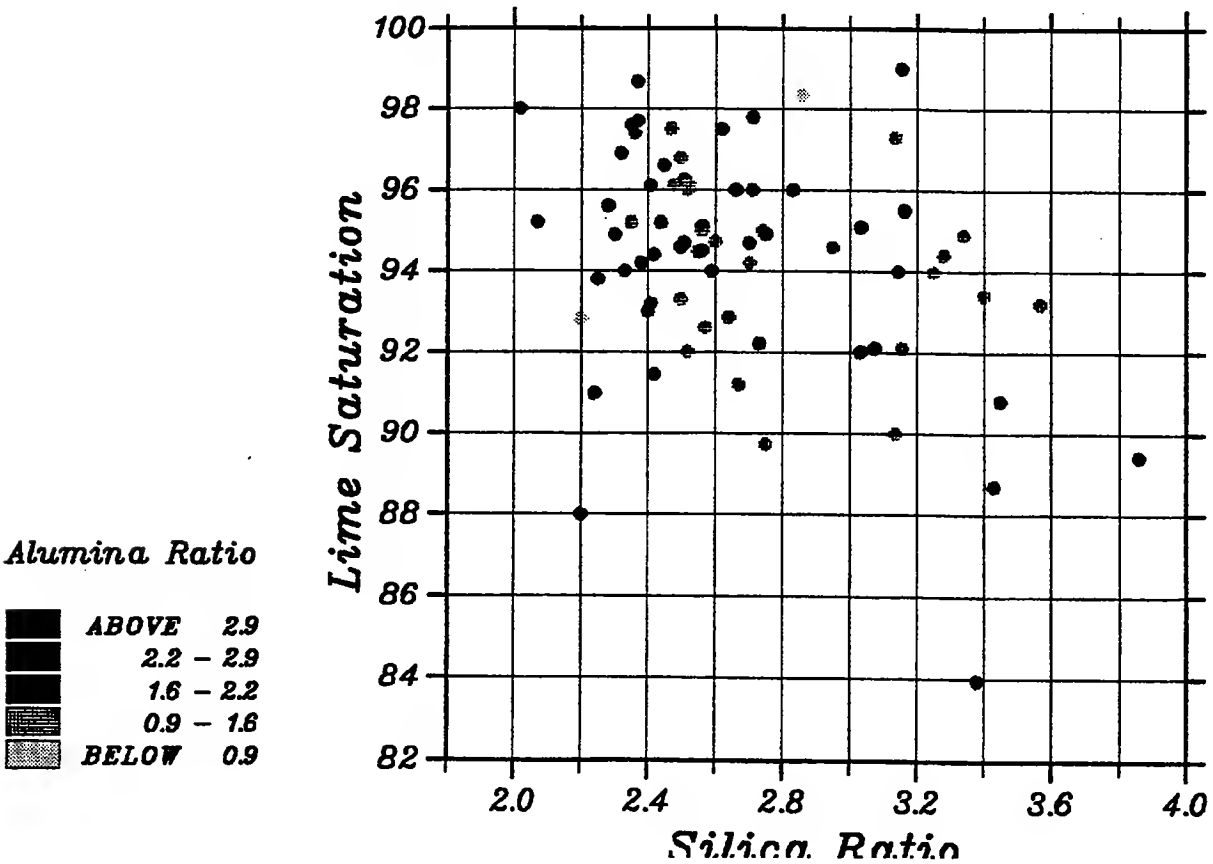
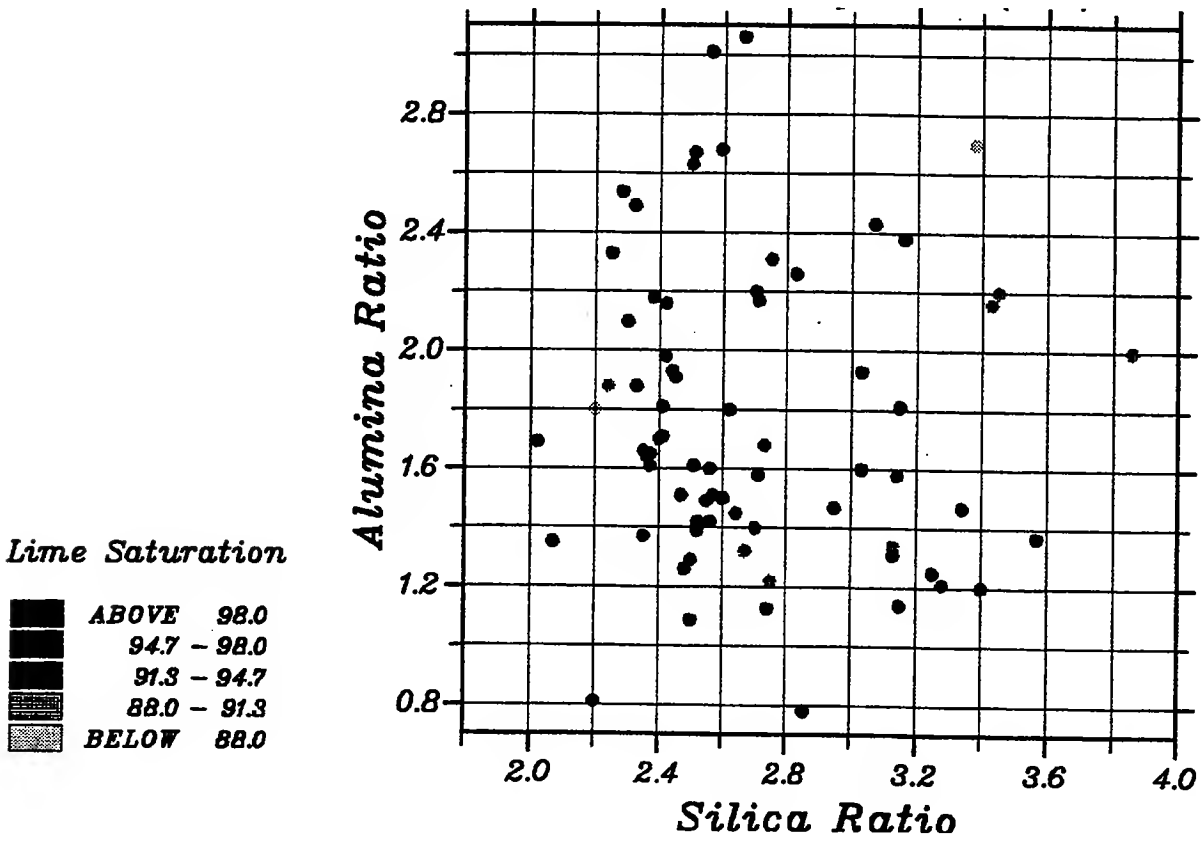


Fig. 9 Clinker Composition (Lime Saturation).



4.5 Kiln Feed

Purpose

Kiln feed control has only informative character. It provides guideline information (LS, SR, AR) for kiln operation on currently processed material, as well as indications on the raw material blending and homogenising efficiency. Figure 10a and 10b are an illustration of a poor kiln feed uniformity, caused by poor raw mill control and low homogenisation silo efficiency.

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Fig. 10a Kiln Feed.

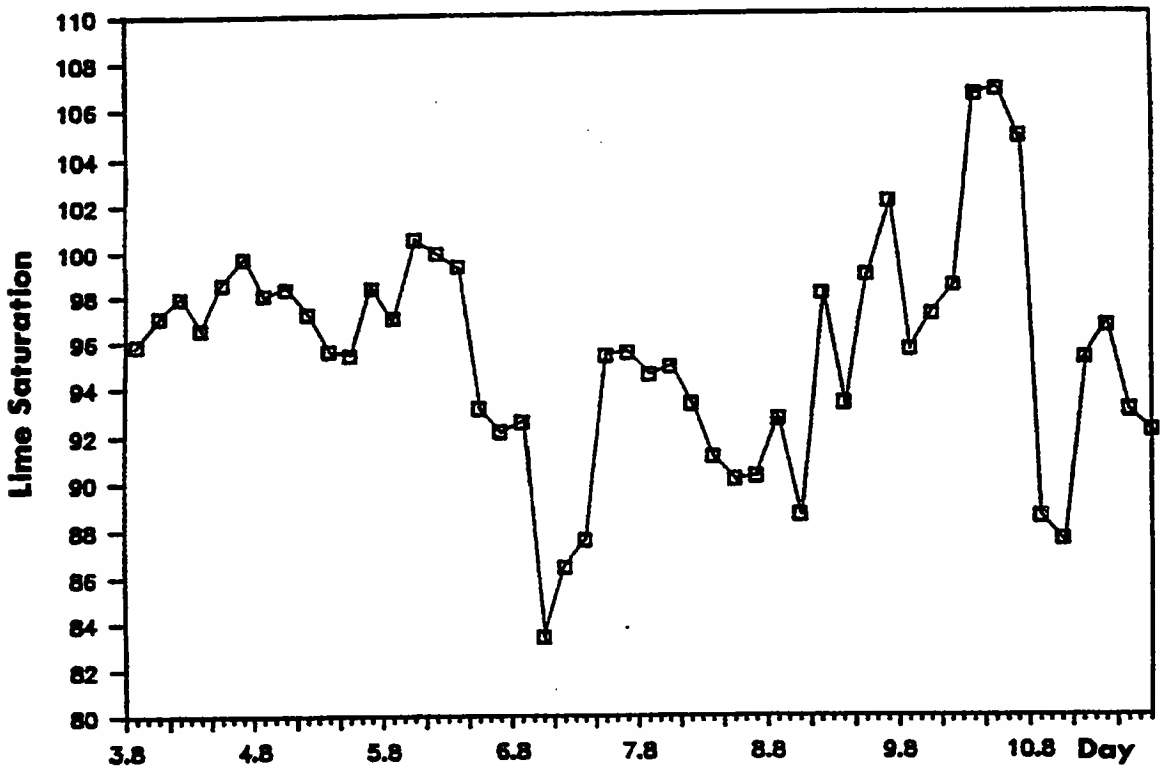
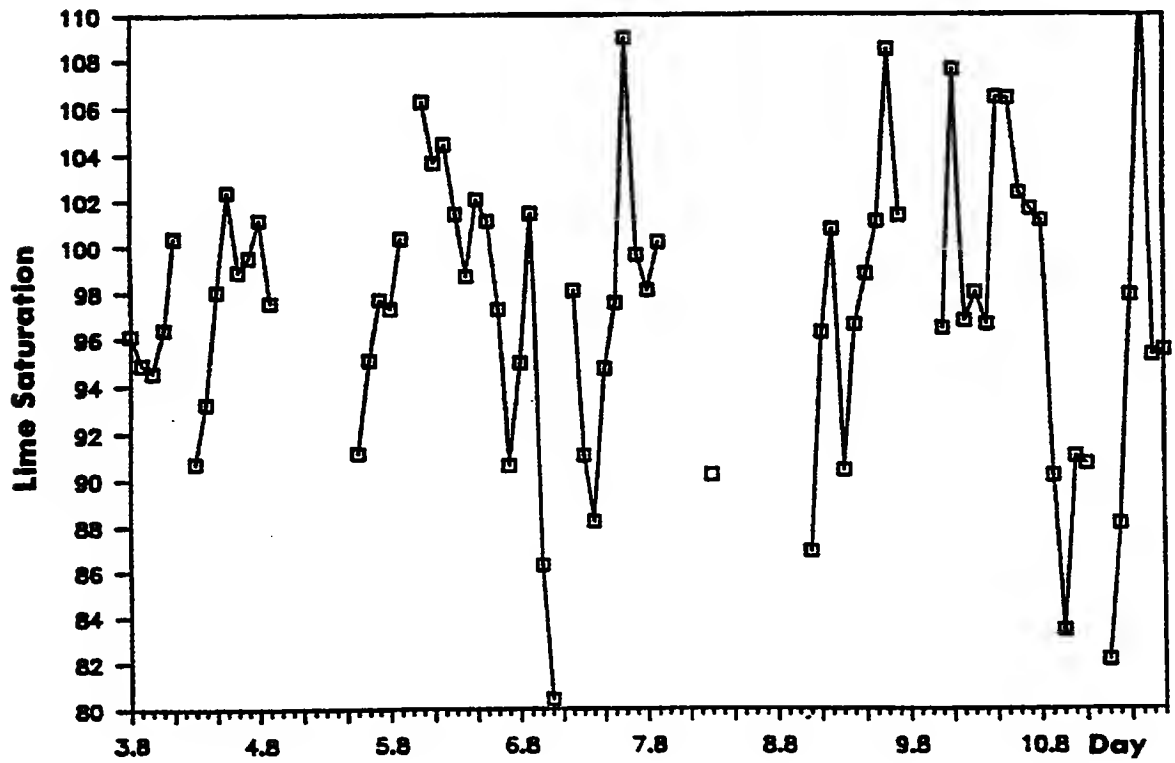


Fig. 10b Raw Meal.



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Procedure

Usual sampling positions are in front of a kiln feed airlift (e.g. with screw sampler) or from a belt weigh feeder (manually). Sampling frequency varies according to the plant specific requirements form e.g. 2 samples per shift to no control at all. Normally, the same chemical and fineness parameters are determined as for the raw meal.

4.6 Hot meal

Purpose

The control of the hot meal composition can have various objectives:

- Determination of concentration of volatile elements (SO_3, alkalis, Cl) in the case of kiln systems affected by build-up formation in the pre-heater/kiln inlet area. The level of volatile elements can provide indications or, in severe cases, can serve as control parameters for the operation of the kiln, with the objective to reduce the tendency for build-up formation. Moreover, they indicate changes in the absolute and relative input of circulation elements with raw meal and fuel, and corrective actions can - if possible - be initiated.
- Determination of non-burnt combustibles introduced with the kiln feed or the secondary firing. This can be of interest for assessing the combustion efficiency of pre-calciners, or in the context of problems associated with circulating elements (locally reducing burning conditions).
- Determination of the degree of pre-calcination in pre-calciner and pre-heater kilos.

The hot meal control can be carried out on a regular basis (e.g. 1 per shift), or in the context of specific investigations, depending on the prevailing plant situation.

Sampling

Hot meal is usually sampled manually from the meal duct downstream the lowest cyclone, with a cup welded to a steel rod. For determination of remaining carbon, quenching of the sample within the sampling cup is advisable. Fig. 11 shows a sketch of a sampler with a lid to facilitate quenching without water spilling into the sample.

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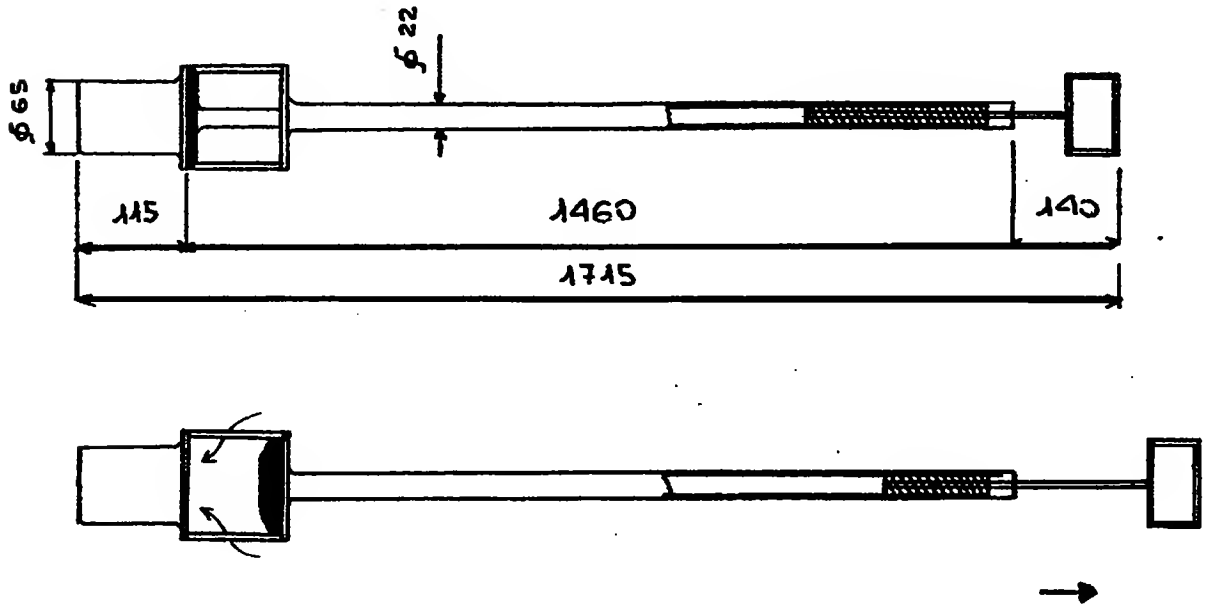
Fig. 11 Hot Meal Sampler.



Analysis

Accurate and sufficiently rapid analyses of the elements SO3, alkalis and C1 require analytical instruments which do not necessarily belong to the standard equipment of a plant laboratory. Suitable methods are:

SO_3	: LECO sulphur analyser
K2O, Na2O	: Flame photometry; atomic absorption spectroscopy
C1	: Potentiometric titration.

The determination of these elements in pressed tablets by Xray fluorescence spectroscopy a very fast method available in most plant laboratories is less accurate. Reasons for the inaccuracy are strong matrix effects and the lack of suitable calibration standards. Hot meals applied as calibration standards are chemically unstable (hydration, carbonisation) and hence not suitable. Better suited are artificial calibration standards prepared on the basis of raw meal, possibly also of clinker.

However, for practical purposes, the accuracy achievable with the XRFmethod can still be acceptable for a correct assessment of the prevailing situation regarding the levels of circulating elements.

4.7 Clinker

Purpose

The objectives of clinker control are:

- Assurance of an adequate clinker quality in view of the final product cement
- Support for an efficient kiln operation at low heat consumption.

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The primary criterion of the clinker quality is the free lime content; the main consequences of inadequate free lime as a result of too weak or too hard burning are:

in the case of too high free lime:

- Loss in strength potential
- Increase of cement expansion (Le Châtelier, autoclave), to the extent of nonconformity with specifications
- Disturbances in cement grinding

in case of too low free lime:

- Loss in cement reactivity
- Excessive heat consumption
- Poor grindability.

An adequate target free lime content has to be established individually for each production line (and clinker type). The following statistical figures for the average free lime content of the clinkers from the 'Holderbank' Group plants illustrate the range encountered in industrial clinkers (Fig. 12):

CaO_f:	x	0.96%	
	s	0.45%	
	min.	0.20%	
	max.	2.16%	(n = 71)

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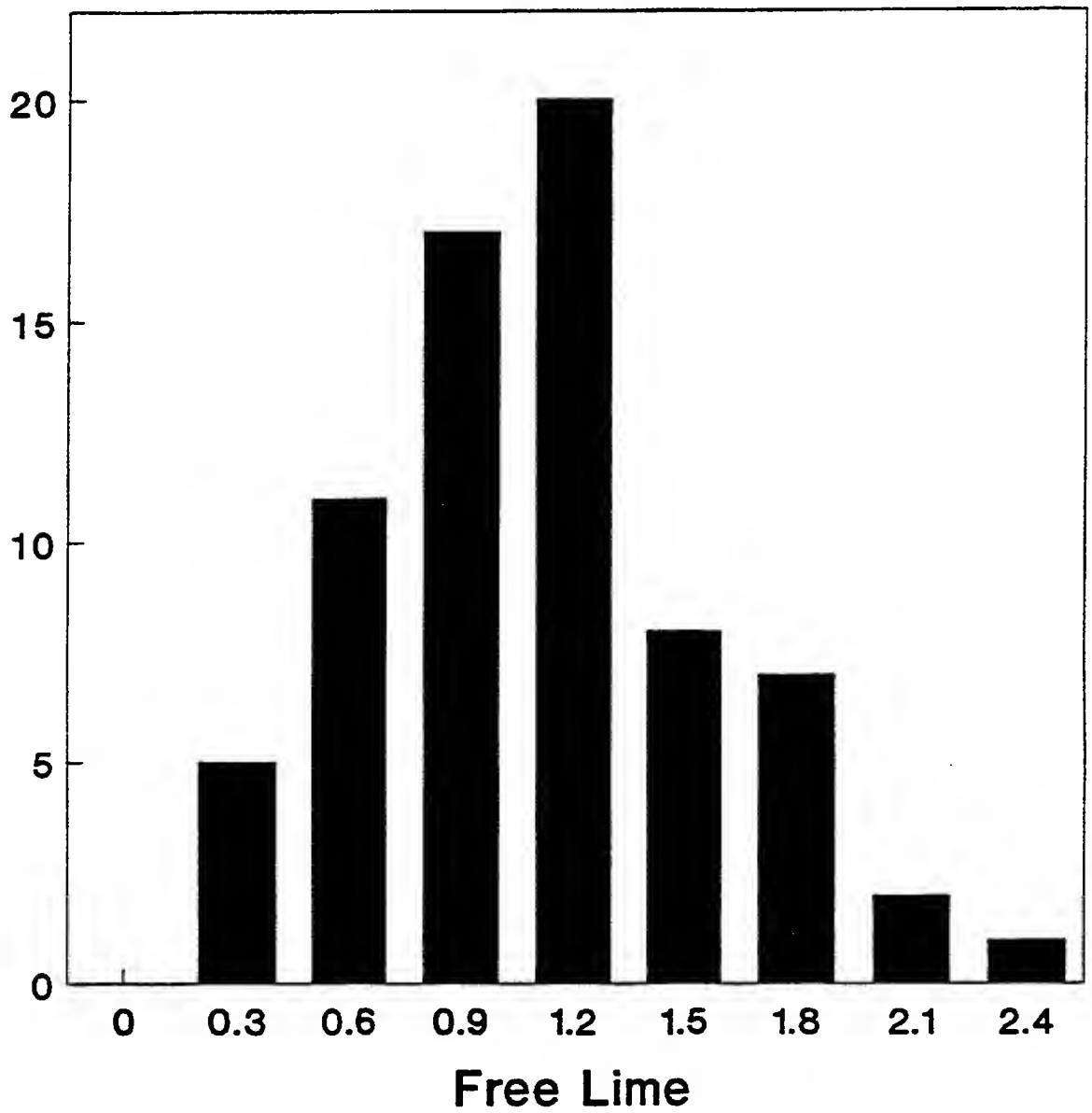
Fig. 12 Free Lime in Industrial Clinkers ('Holderbank' Group Plants 1988).



The target free lime content should be such that it permits a stable and economic kiln operation and that an appropriate cement quality can be achieved. Upper limits have to be defined, above which the cement quality will be seriously affected; clinker not meeting this internal specification should be stored separately and returned to the process in a controlled way.

The extent of clinker control too has to be established individually for each situation, depending on the uniformity of materials flow and general stability of kiln operation.

Frequent control (hourly or even more often), possibly with automated systems, can though need not necessarily in all cases lead to reduced fuel consumption and hence appreciable cost savings. The signals of the automatically determined control parameters (free lime, litre-weight) can be transmitted to and be utilised in kiln automation systems.

The free lime determination can be substituted or complemented for process control purposes by litre-weight or alkali content determination. In a number of cases good correlation's are also found with NO_X measurement.

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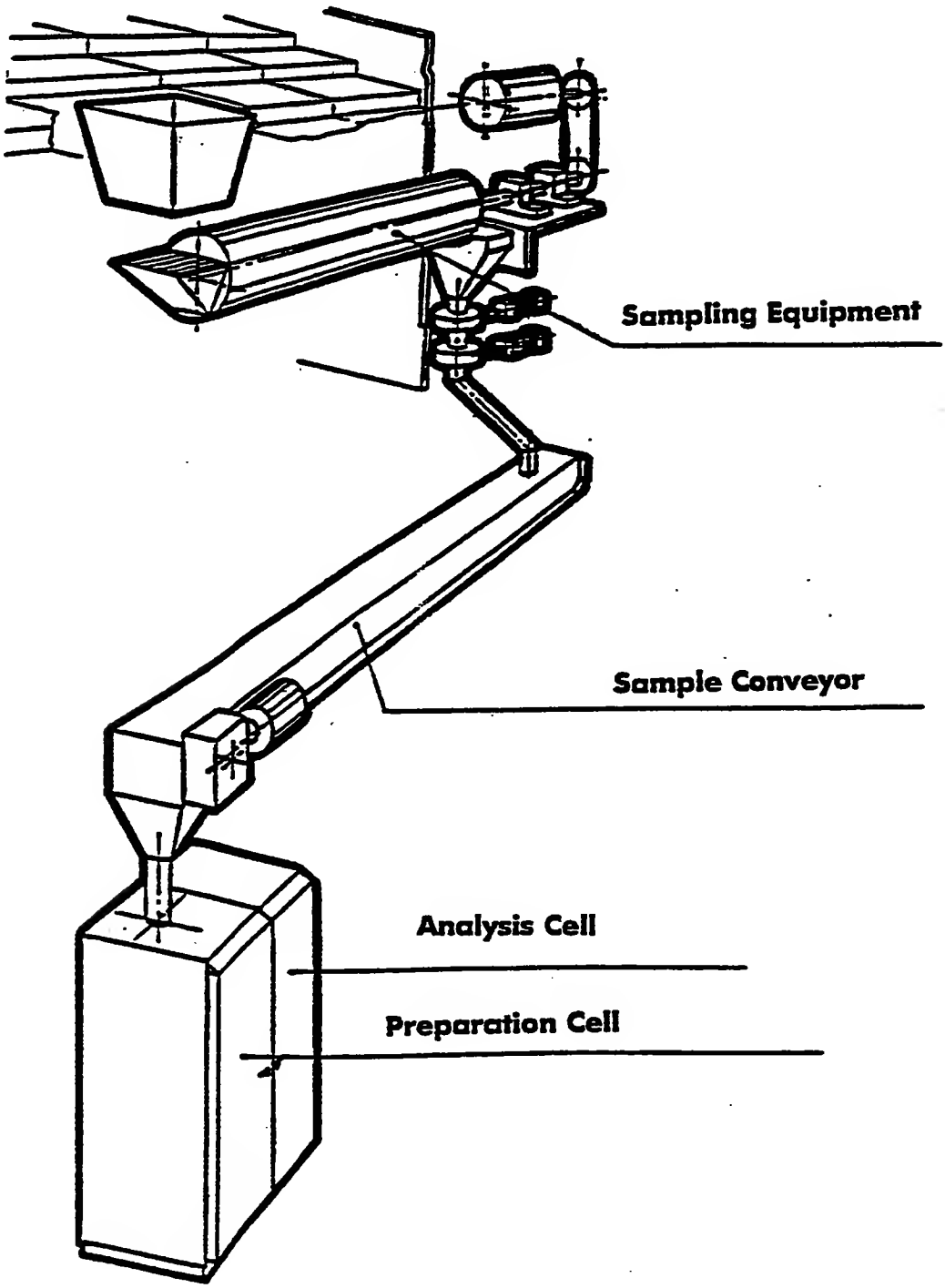
Clinker sampling

Clinker is still mostly sampled manually from a conveyor belt. Automated clinker sampling can either be done along the clinker transport or inside of grate coolers (Fig. 13).

Fig. 13 Clinker Sampling.



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Free lime determination

In addition to the established laboratory methods for manual free lime determination by titration (ethylene glycol method), there are different systems for automated free lime determination:

- **Conductometric free lime determination**

Automatic free lime analysers, particularly designed for installation outside the laboratory, e.g. close to the clinker cooler, are offered by two suppliers in France, namely ISYS and ITECA. The layout is shown in Fig. 13.

In a usual arrangement, clinker samples are taken underneath the cooler grate. Free lime is determined by extraction of the ground clinker with ethylene glycol and measurement of the resulting conductivity of the solution. A measuring cycle takes about 6 to 10 minutes. The results are transmitted to the central control room or laboratory.

Such systems are, with good experience, fairly widely applied in French and Belgium plants.

It should be noted that the conductivity measurement is influenced by soluble alkali sulphates. The calibration has to account for this, though a more or less uniform alkali sulphate content in clinker is a prerequisite for reliable measurements.

- **Free lime determination by X-ray diffraction**

This method is applied in a number of plant laboratories, usually in connection with the X-ray fluorescence spectroscopy and quite often integrated in automatic systems. The chemical composition and the free lime content can be determined on one and the same pressed tablet by XRF and XRD respectively. For the free lime determination, the peak intensity or integrated intensity of the CaO 200-reflection (or 220-reflection) is measured.

Calibration curves have to be established individually for each plant; due to the chemical instability of CaO_{free} containing calibration standards, secondary standards containing NiO are usually applied.

The accuracy of the method as compared to the wet chemical methods is not excellent but mostly sufficient for practical purposes (standard deviation of linear regression in the order of 0.1 0.15% CaO_f).

It has to be noted that only CaO_{free} is determined, in contrast to the wet chemical or conductometric method, where the sum of CaO_f and $Ca(OH)_2$ is measured.

Litre-weight determination

The determination of the clinker litre-weight on a sieved fraction typically 4 8 mm is a fairly fast method frequently used for clinker control on an hourly or two hourly basis. It is usually complemented by free lime determination on shift or daily composite samples.

The litre-weight determination can only be applied, if there is a sufficiently good correlation with the free lime content. From Fig. 14 and 15 it can be derived that the free lime content is generally a reliable indicator of the degree of burning, whereas the litre-weight is not in all cases.

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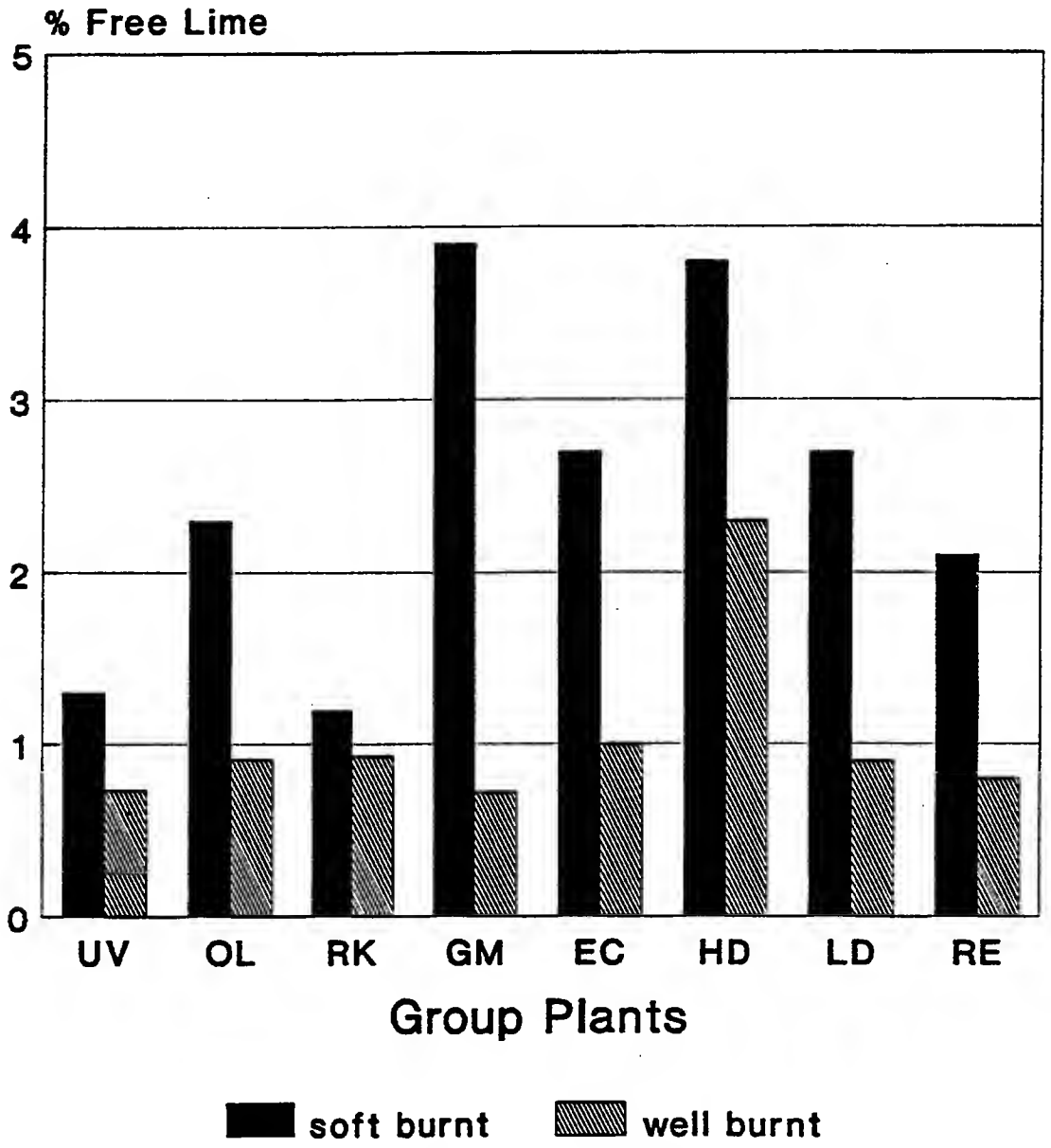
Fig. 14 Free Lime Content Soft and well burnt Clinkers.



CLINKER GRINDABILITY

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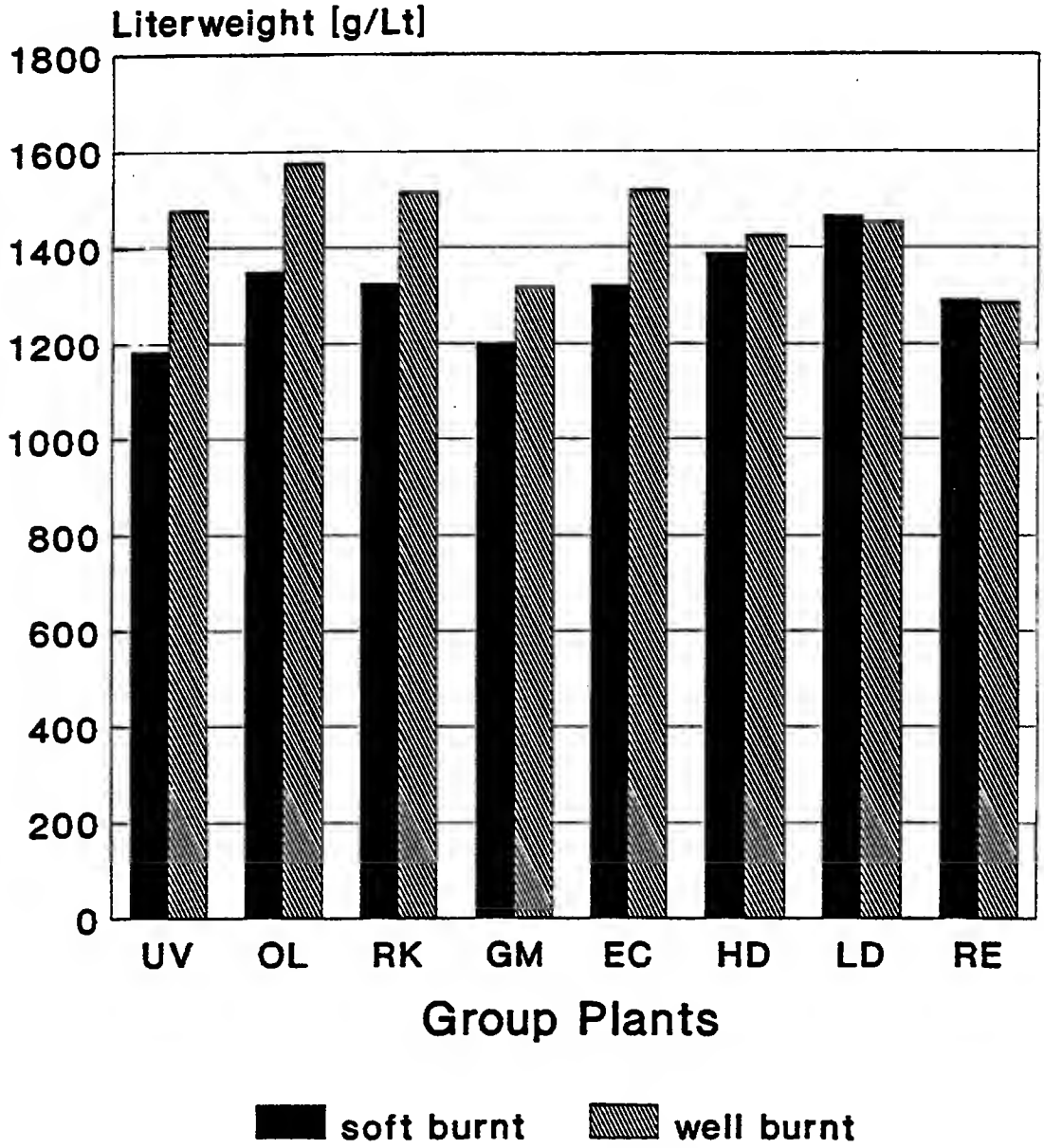
Fig. 15 Litre-Weight (Soft and well burnt Clinker).



CLINKER GRINDABILITY

In those cases where the litre-weight is a good indicator of the degree of burning, the correlation with free lime can vary in dependence with the clinker composition (Fig. 16). Furthermore, the linear increase of the litre-weight with harder burning can inverse at very hard burning (Fig. 17).

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Fig. 16 **Correlation between Free Lime content; Litre-weight and Silica Modulus.**

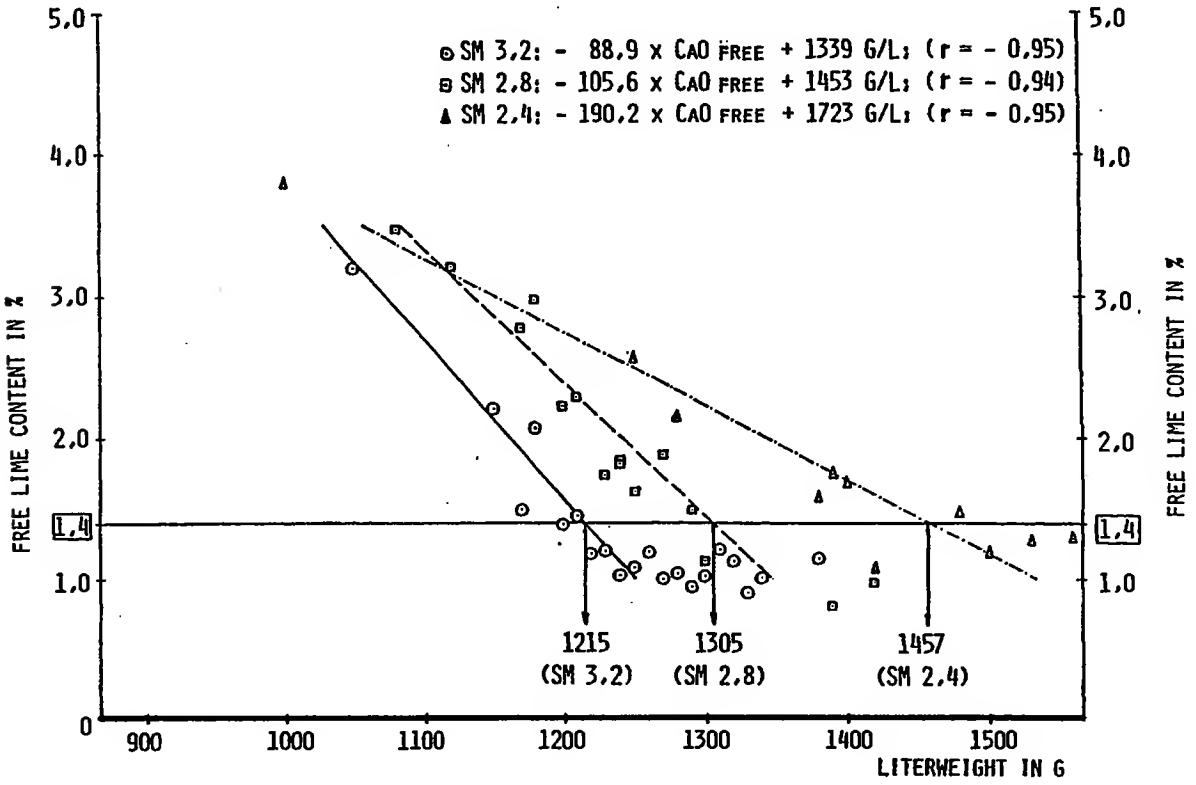

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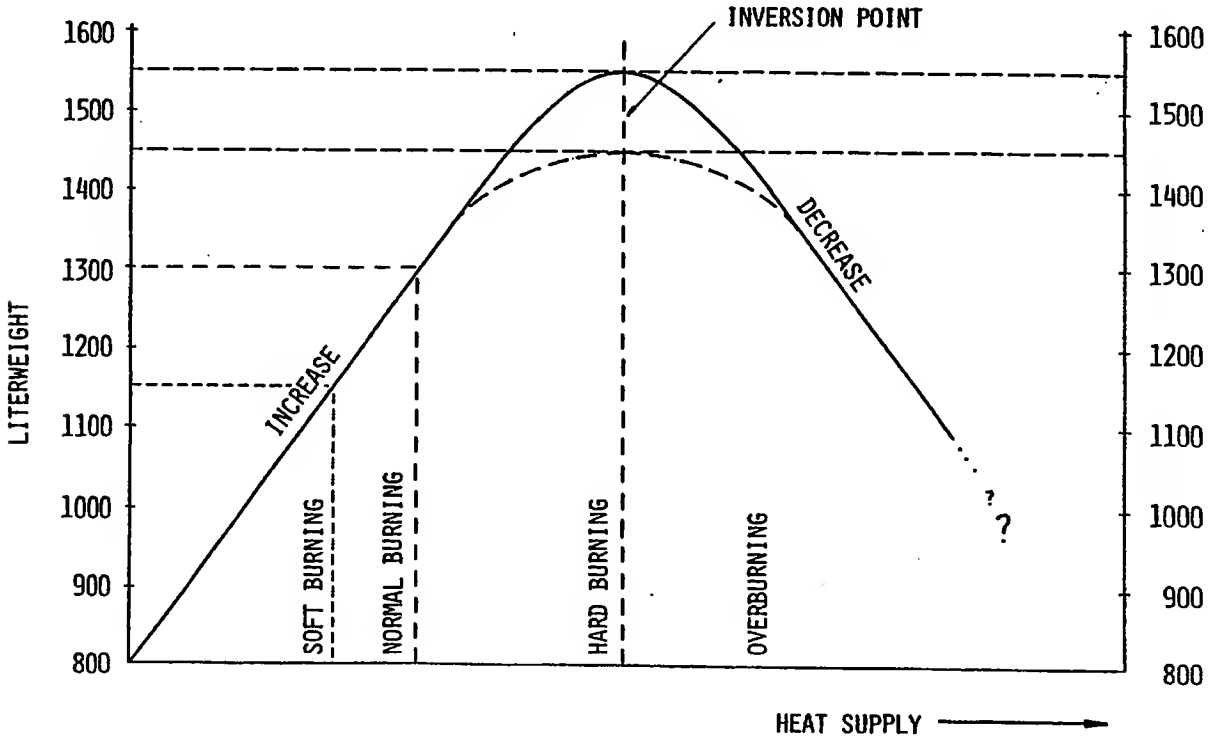
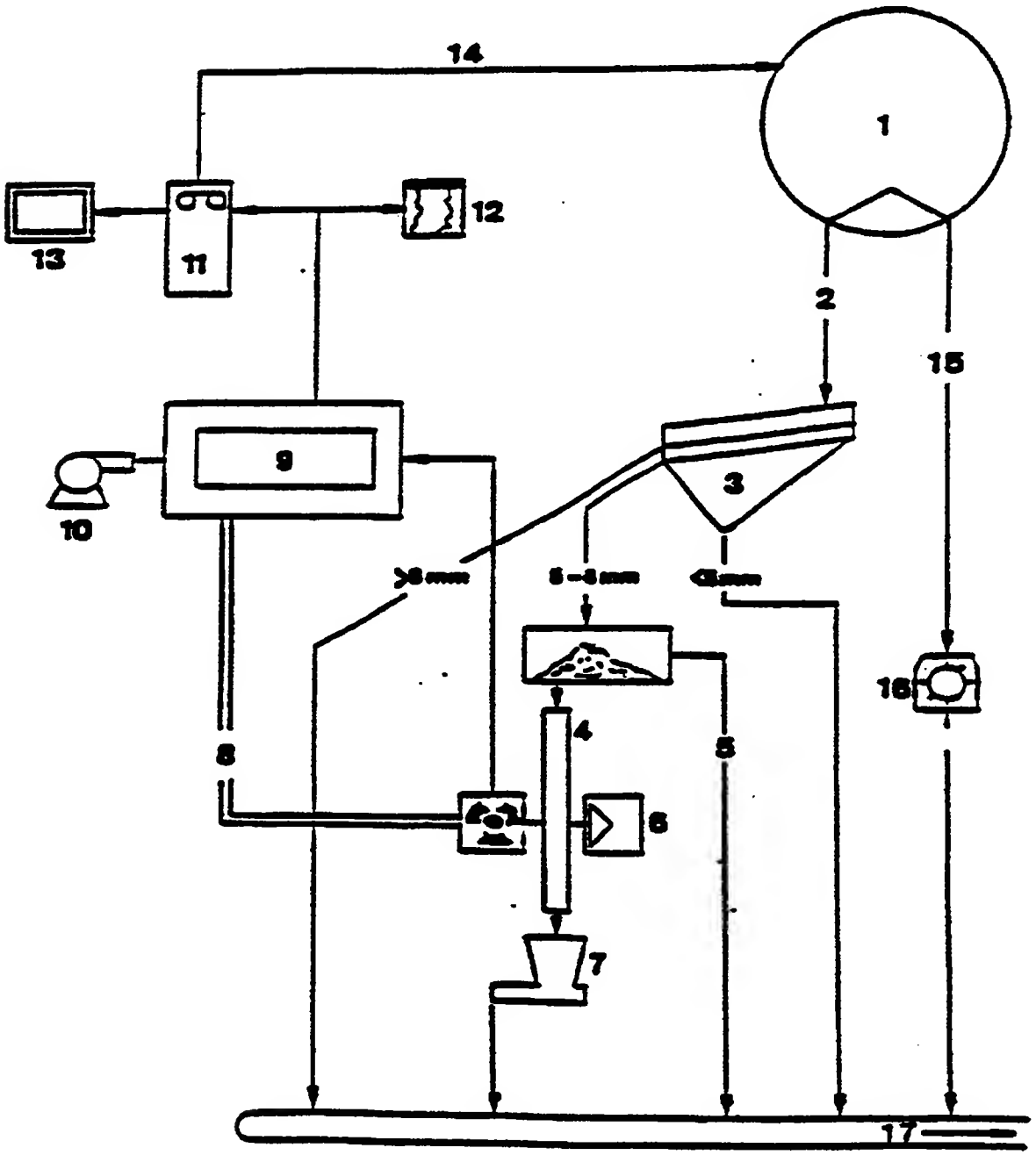
Fig. 17 **Correlation between Free Lime content and Litre-weight for over burnt Clinker**



At a 'Holderbank' Group plant in Switzerland, the manual litre-weight determination has successfully been replaced by an automatic online system, measuring the apparent clinker density with a nuclear densitometer. The system is schematically shown in Fig. 18a, along with the correlation of automatic and manual litre-weight measurement (Fig. 18b). It is worth noting that the density measurement is used as the primary set point parameter for the automated kiln control system.

Fig. 18a On-line Litre-Weight Determination at Eclepens



Circuit diagram for the continuous measurement of the apparent density of clinker.

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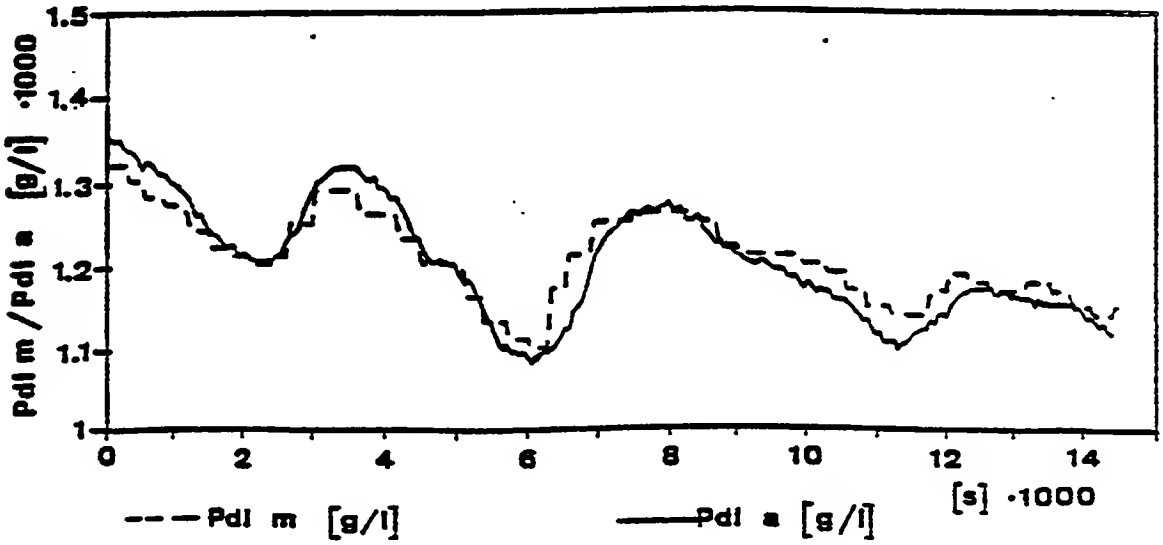
Fig 18b Manual and Automatic measurements.



Other control parameters

In a few plants, the levels of K_2O or SO_3 in clinker have been found to be the most sensitive clinker parameter with respect to the burning degree. The clinker is analysed in these cases on a hourly or two hourly basis by XRF spectroscopy.

Moreover, a correlation between the NOx level in the kiln gas after the pre-heated and the free lime content of the clinker was established in several investigations (Fig. 19). However, this correlation can be disturbed by various factors influencing the NOx measurement, and can, therefore, not make a clinker control altogether redundant.

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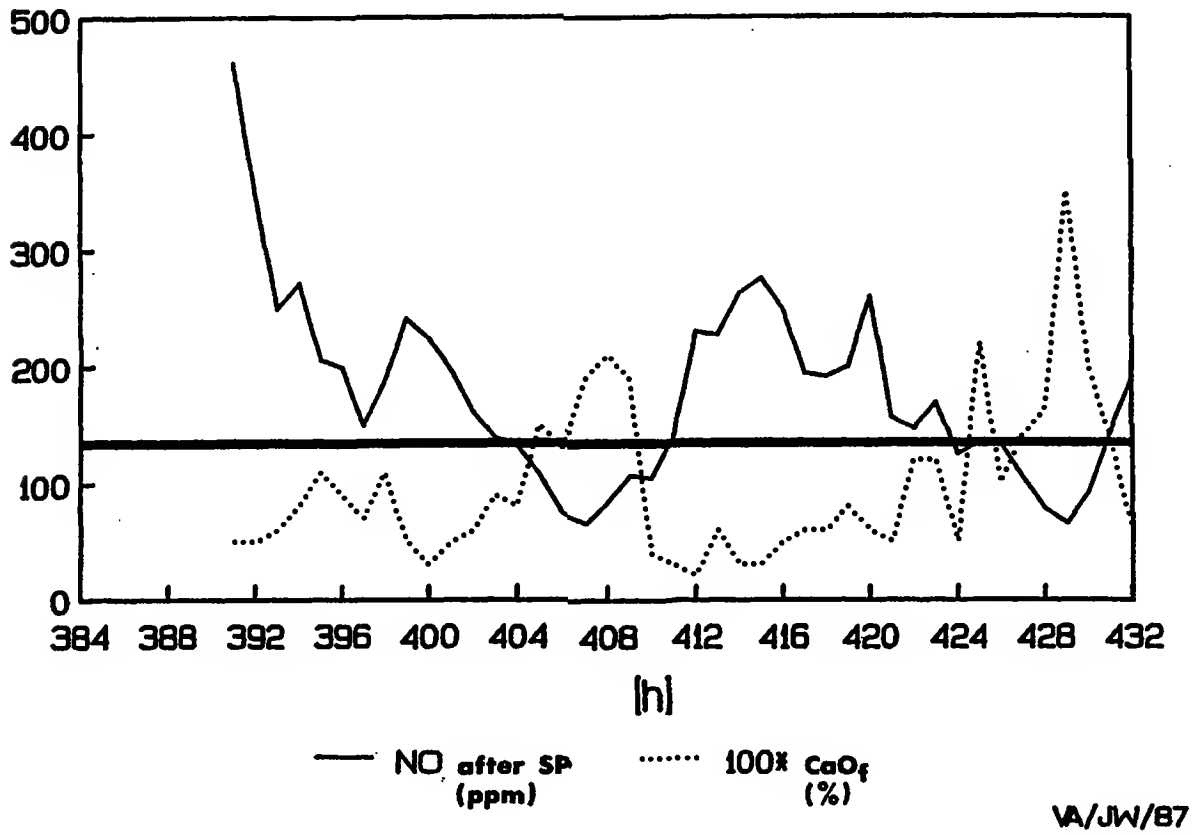
Fig. 19 NO and Free Lime.



4.8 Ground Cement

Purpose

The control of the ground cement serves to ensure the designed cement quality in terms of proportioning of constituents and achievement of the required mechanical and physical properties. Since the determination of the latter is time consuming, the process control has to be based on chemical, fineness and possibly temperature parameters, for which a correlation with the cement performance has been established.

Control parameters

- For process control:
  - Fineness: Blaine; Residues (e.g. R45 ~m); Parameter from particle size distribution (slope, size fraction)
  - SO_3: for gypsum dosage
  - Selected elements: for dosage of mineral additives
  - Temperature: to control gypsum dehydration
- For quality control
  - Standard chemical, mechanical and physical testing; usually on composite sample (e.g. daily).

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Process interaction

Adjustment of feed rate and of separator speed, usually in off-line, but increasingly also in online mode.

Sampling

Automatic or manual sampling from chute, airslide or conveyor belt. Various standard types of samplers are used, typically screw samplers or air-slide sampler.

Frequency

Typically one sample per 1 to 4 hours.

Testing methods

SO_3:	XRF, LECO
Chemical composition:	XRF
Blaine:	- manual or semiautomatic Blaine meter - on-line Blaine meter - on-line Gamma granulometer
Particle size distribution:	Laser granulometer.

4.9 Dispatched Cement

Purpose

Sampling and testing of the dispatched cement serves as a final verification of the cement quality. Testing of dispatched cement can be a requirement of product conformity certification schemes (so-called auto-control), or it can be requested in the context of delivery contracts to specific customers. Samples can be retained for a certain period without testing, as a reference in case of complaints.

Control parameter

Chemical composition, standard mechanical and physical properties.

Sampling

Manual or semiautomatic from silo outlet, loading device or bags. Sampling procedures described in pertaining cement standards.

Frequency

Daily or according to applicable sampling schemes. The European standard ENV 197 requests a minimum of 2 samples per week and type of cement for assessing standard conformity.

4.10 Ground Coal

Purpose

The control of fine coal is required to ensure adequate flame properties (fineness), to verify the calorific value, as well as to verify ash and sulphur content which influence clinker composition.

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Control parameters

- For process control:
  - Fineness: sieve residues R90 µm, R200 µm
  - Moisture
- for verification and trend analysis:
  - calorific value
  - proximate analysis (moisture, volatiles, fixed C, ash)
- S, Cl, if required H, N (ultimate analysis)
- ash composition.

Sampling

Manual slitsampler or automatic screw sampler.

Frequency

Every 1 to 4 hours for fineness. Other tests on composite sample (e.g. daily).

Testing

- Fineness : Sieves
- Calorific value : Calorimeter
- Proximate analysis : According to pertaining standards
  : Automatic proximate analyser
- Ultimate analysis : According to pertaining standards
- Sulphur : Gravimetric (fusion)
- Chloride : Titration.

5. UNIFORMITY

Significance

Uniformity is one of the most important prerequisites for smooth and efficient kiln operation.

Excessive fluctuations can easily upset the often less than robust balance of appropriate burning behaviour, thus giving rise to operational disturbances. Furthermore, since the clinker should always be adequately burnt, unnecessarily hard burning is practised for much of the production time.

Consequences are:

- high fuel consumption
- increased tendency to ring and build-up formation
- reduced brick lifetime
- fluctuations in product quality

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Sources of fluctuations

- Raw components: chemical and mineralogical composition
- Raw meal: chemical and mineralogical composition, fineness; disturbances in feed rate to the mill
- Combustible: ash content, sulphur content, calorific value, fineness
- Dust return: different modes of return during direct or indirect operation
- Feed rates: (Kiln feed, combustibles) equipment related fluctuations at constant settings
- Abrupt manual adjustment of kiln operation parameters
- Abrupt and excessive adjustment initiated by automatic control loops.

Acceptable variability of kiln feed composition

Since the reduction of material variability is associated with costs for homogenisation and control, an acceptable, optimised target has to be established for each specific situation. Raw meals or clinkers of an extreme composition (e.g. LS close to 100, high SR, low AR) or with unfavourable fineness characteristics require more stringent variation limits than easy to burn clinkers. What eventually can be achieved depends on the complexity of the raw material situation, the pre-blending and homogenising facilities as well as on the efficiency of the quality control system.

Some values which are considered to provide a good basis for kiln operation, even when using difficult materials, are given below.

Guide values for feed variations

- Accuracy of the feed system: 1 to 1.5%
- Regular calibration of the system mandatory

Guide values for good chemical uniformity of the kiln feed
(for random samples over 24 h; on raw meal rather than clinker basis)

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	Without analytical error	Including analytical error
Titration method	s = 0.2 % $CaCO_3$	s = 0.25 % $CaCO_3$
max.	s = 0.3 % $CaCO_3$	s = 0.35 % $CaCO_3$
XRF method:	s = 0.2 % CaO	s = 0.15 % CaO
max.	s = 0.17 % CaO	s = 0.20 % CaO
SiO_2	s = 0.07 % SiO_2	s = 0.2 % SiO_2
Al_2O_3	s = 0.07 % Al_2O_3	s = 0.2 % Al_2O_3
Fe_2O_3	s = 0.04 % Fe_2O_3	s = 0.2 % Fe_2O_3
LS	s = 0.7	s = 1.2
SR	s = 0.04	s = 0.06
AR	s = 0.06	s = 0.08

Means of improving uniformity

The kiln feed uniformity is a result of various factors along the preparation process, starting from the raw material deposits and going through several stages of homogenisation and blending.

Possibilities for improvements can be found in the following areas:

- Exploitation planning
  Medium to long-term exploitation planning, based on accurate raw materials inventory. Application of computer based methods.
- Quarry scheduling
  Short-term quarry scheduling, based on blast hole dust analysis. Application of computer based methods.
- Raw material and coal pre-blending
- Blending control
  Blending control at integrated pre-blending stockpile and raw mill. Appropriate selection of sampling regime and analytical methods. Various degrees of automation from manual to online, or even to 'real-time' with bulk material analyser.
- Raw meal homogenisation

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6. EXTENDED QUALITY CONTROL

In addition to the basic test methods for routine control, a large variety of sophisticated methods and procedures is applied for material technological investigations in connection with special manufacturing and application problems and in research and development. Characterisation of raw materials, clinker, additives, etc. is not only limited to chemical analysis, but also the mineralogical and textural features are important. Thermo-analytical (DTA, TGA), microscopic (light and electron microscope) and X-ray diffraction (XRD) methods are used as routine procedures in this context. Microprobe is a powerful tool to determine the elemental composition of individual clinker minerals. Burnability and volatility tests are used to assess the raw mix behaviour in the industrial kilos.

Although most of these methods are only applied in central or research laboratories, some of them are also used in a number of plant laboratories for routine quality control, such as X-ray diffraction to monitor the quartz and clay minerals of raw materials and to examine free lime in clinker; or differential thermal analysis to control the dehydration of gypsum in cement during the grinding process. However, if these specialised testing procedures are promoted too much, there is the danger that they might finally be included in the standard specifications. This has to be avoided because it would make the quality control procedures too sophisticated and expensive for the majority of plants.

In addition to the traditional tasks and methods of quality control, in many plants the scope of control is significantly extended, not only due to more stringent process conditions and to higher quality requirements and special requests by the customers. The following tasks require extended application of already established quality control methods or even introduction of new techniques:

- Refractories testing, to ensure the material quality of the installed kiln brick lining which requires apart from visual inspection extended chemical analysis, simple physical and thermal tests.
- Quality control of additives for production of blended cements; depending on the kind of additive, its source and supply conditions, more or less extensive chemical, mineralogical and performance tests have to be carried out; with the increasing use of industrial by-products, such as fly ashes, slags, etc., efforts are also undertaken to establish reliable and rational methods for their control and for their determination in blended cements.
- The Blaine value for control and characterisation of cement fineness with respect to grinding process and cement quality is only of limited significance. therefore, grain size analysis by laser diffraction or automatic sedigraphs is used increasingly for quality and process control; in the future, the laser diffraction technique might even be used in online installations.
- Use of waste materials as raw mix components and particularly as combustibles (e.g. tyres, colliery wastes, battery casings, chemicals, etc.) require adaptation of existing or new testing methods.
- Last but not least environmental concerns and the resulting pressure upon industries force plants to extend their quality control into the area of emission control (and, in some instances, even emission control); this implies development and refinement of techniques for gas analysis especially for SO2 , NO_X, but also for hydrocarbons as well as for dust determination and collection; analysis of traces in the ppm or even the ppb-range for toxic elements such as Ti, Cd, Pb, Hg, etc., nowadays is not an unusual task for the cement industry.

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Chapter 10

Basic Principles of Sampling and Testing

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Basic Principles of Sampling and Testing

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1. INTRODUCTION

When dealing with bulk materials, the properties of a certain quantity of material, such as the contents of a preblending bed, a shift's cement production or a coal delivery, cannot be determined on the entire quantity, but have to be estimated on the basis of measurements carried out on a limited number of individual samples. The reliability of such an estimate depends on the heterogeneity of the material and on the procedure of sampling, sample preparation and analysis. A high reliability - as may be required for production or market purposes - is usually associated with comprehensive sampling, with precise preparation procedures and analytical instruments, and may thus incur considerable costs. The objective of a well designed testing procedure is, therefore, to extract the necessary information on quality parameters of a material at the lowest possible costs.

This paper reviews various aspects relevant to the design of an appropriate sampling regime, and summarizes the main factors influencing the reliability of test results. Emphasis is given to practical aspects rather than to the theory and statistics behind them; for the latter, reference should be made to the paper "Statistics" contained in the seminar documentation.

2. SAMPLING REGIME

2.1 Basic considerations

In order to decide on an adequate sampling regime and corresponding equipment, some considerations are necessary concerning the objectives of sampling, the representativity of samples from bulk materials and the material characteristics to be determined.

2.1.1 Objectives

Bulk materials can be sampled either for a mere determination of the characteristics of a certain basic material quantity or for the purpose of process control.

a) Characterization of a given basic material quantity

Examples of such materials are:

- Area of quarry to be blasted
- Materials delivered to the plant (coal, slag, gypsum)
- Kiln feed
- Dispatched cement

The sampling regime has to be adapted to the given variability of the material and to the required precision of the measurement. The complexity of sampling can vary from occasional manual grab samples to sampling regimes prescribed in norms (bulk cement) or agreed upon between seller and purchaser.

b) Control of process

Examples:

- Preparation of preblending bed
- Raw and cement mill control
- Clinker control

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The sampling regime, in particular the sampling frequency, has to be adapted to the target variability of the output material, to the variability of the input materials and to the dynamics of the process, i.e. how fast and how precise corrections can be made and how fast they can be observed (dead time). This type of sampling can readily be automated, which permits a comfortable precision to be attained.

2.1.2 Representativity of samples

In order to obtain meaningful information from a sample on the characteristics of the basic material quantity, the sample must exhibit these characteristics in a representative way. Representative means that the value for a certain characteristic determined on the sample differs only randomly - and not systematically - from the corresponding mean value of the basic material quantity. For representative sampling, it is normally a prerequisite that all volume sectors of the material have an equal chance of being sampled, and that all particle sizes are correctly represented.

Representative sampling of bulk material is often rendered difficult, especially when stationary material is concerned; some of the reasons are:

- Large volumes can be inaccessible when sampling materials on stockpiles, in silos, on ships or on trucks
- Stationary materials can be stratified due to sequential deposition of different materials
- Materials with a broad particle size range tend to segregate, i.e. fines concentrate on the bottom of a material heap. If coarse and fine particles differ in characteristics of concern (e.g. composition), considerable inhomogeneities will occur, which can be an important source of sampling error. Excessive moisture behaves in a similar way.
- Separation effects also occur when materials with a broad particle size range are thrown off a quarry face or drop from one conveyor belt onto a second one running in perpendicular direction. In the latter case, strongly inhomogenous material streams result.
- A similar effect occurs when two material streams combine on a conveyor belt, e.g. returned filter dust added to the finish cement conveyor belt.

Sampling from stationary materials should be avoided whenever possible, and replaced by sampling during loading or discharge over conveyor systems. If this cannot be done, preference should be given to traverse sampling, i.e. a primary sample should embrace the whole cross section.

Examples for traverse sampling are:

- Blast hole sampling in the quarry
- Auger sampling of a railway car

Sampling from large stockpiles can be done according to the recommendations given in norms (e.g. ASTM D 75-82 for sampling aggregates) or according to special agreements.

For the reasons mentioned above, due attention must be paid to representative sampling of conveyed materials, independent whether manual or automatic equipment is used.

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2.1.3 Selection of characteristics to be measured

To keep costs to a minimum, it may be adequate to limit the number of characteristics to be determined. Preference should be given to characteristics easy to determine, which strongly correlate with other material characteristics or with process conditions.

Examples are:

Ash content of coal	correlating with	calorific value
CaO- and Fe_2O_3-content in raw-meal	correlating with	remaining elements
Literweight of clinker	correlating with	sinterzone conditions

The validity of correlation has to be periodically verified (e.g. on daily composite samples).

It is evident that the chosen characteristics must not be altered during sample preparation.

2.2 Guidelines for sample size and sampling frequency

The larger the number of samples taken and the larger the extracted quantity per primary sample, the more accurate the sampling of a basic material quantity.

Thus, the minimum number and quantity of samples, which are required for a specific precision of the mean value of a characteristic, have to be determined. They depend upon:

- the grain size
- the heterogeneity of the material

In a first approximation, the following applies:

The minimum size of a primary sample can be related to the maximum grain size of the material. The German DIN specification states that the minimum weight of the primary sample must be at least 10 times the weight of the largest grain. This third power relationship between sample mass and maximum grain size is usually applied in linearised graphical form, as depicted in Fig. 1.

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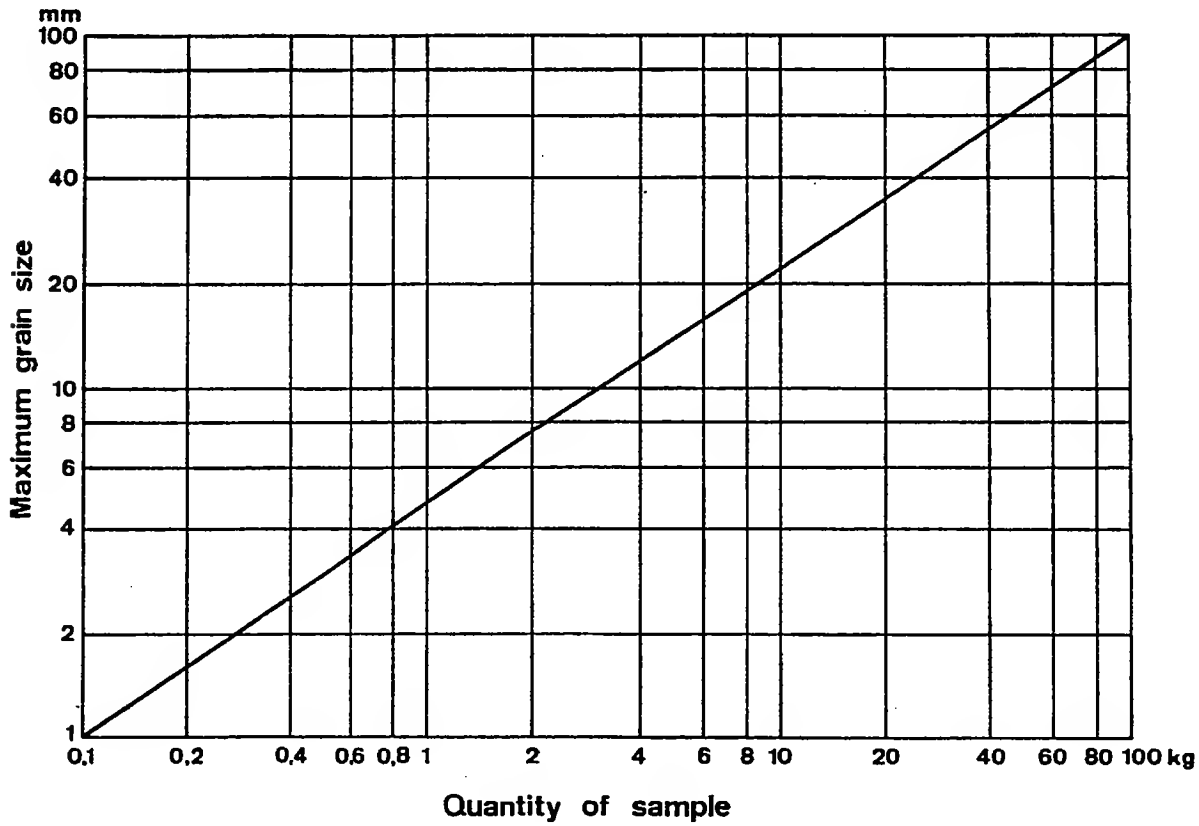
Fig. 1 Maximum grain size versus sample quantity



This graph mainly applies for coarse grained raw materials. For homogeneous fine ground materials such as raw meals or cements, the recommended sample quantity is a few hundred grams to 1 kg. For homogenous coarse grained materials, such as clinker, 1 to 3 kg are sufficient.

It may be noted that various national norm specifications relevant to the mining industry exist. The sample size / grain size relationship can differ considerably from that in Fig. 1, as illustrated by the example of manual sampling with norm scoops shown in Fig. 2.

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Fig. 2: Japanese Industrial Standard M 8100

Sampling scoops and their capacities in kg for sampling bulk solids

Top Size (mm)	Scoop designation	Width (mm)	Approx. (ml)	Weight (kg) of scooped sample at various bulk densities 0.75	1.5	2.0
				t/m^3	t/m^3	t/m^3
5	5	50	75	0.06	0.11	0.15
10	10	75	125	0.09	0.19	0.25
15	15	100	200	0.15	0.3	0.4
20	20	100	300	0.22	0.45	0.6
30	30	150	400	0.3	0.6	0.8
40	40	150	800	0.6	1.2	1.6
50	50	260	1700	1.3	2.5	3.4

The number of primary samples - or the frequency of sampling - directly bears on the sampling precision, since the sampling variance of the mean of n-samples is n-times smaller than that of a single sample.

Hence, if the variance is known for a given basic material quantity - e.g. determined by means of a sampling experiment - the number of samples can be calculated for any desired sampling precision. This rule, however, only applies if all samples come from the same population. If the mean of the observed characteristic changes with time (trend!), samples taken at different times can no longer be regarded as originating from the same population.

Fig. 4 illustrates a reasonable sampling frequency with respect to a given materials variability, as well as too frequent and insufficient sampling.

Automatic sampling normally ensures a more than sufficient sampling frequency.

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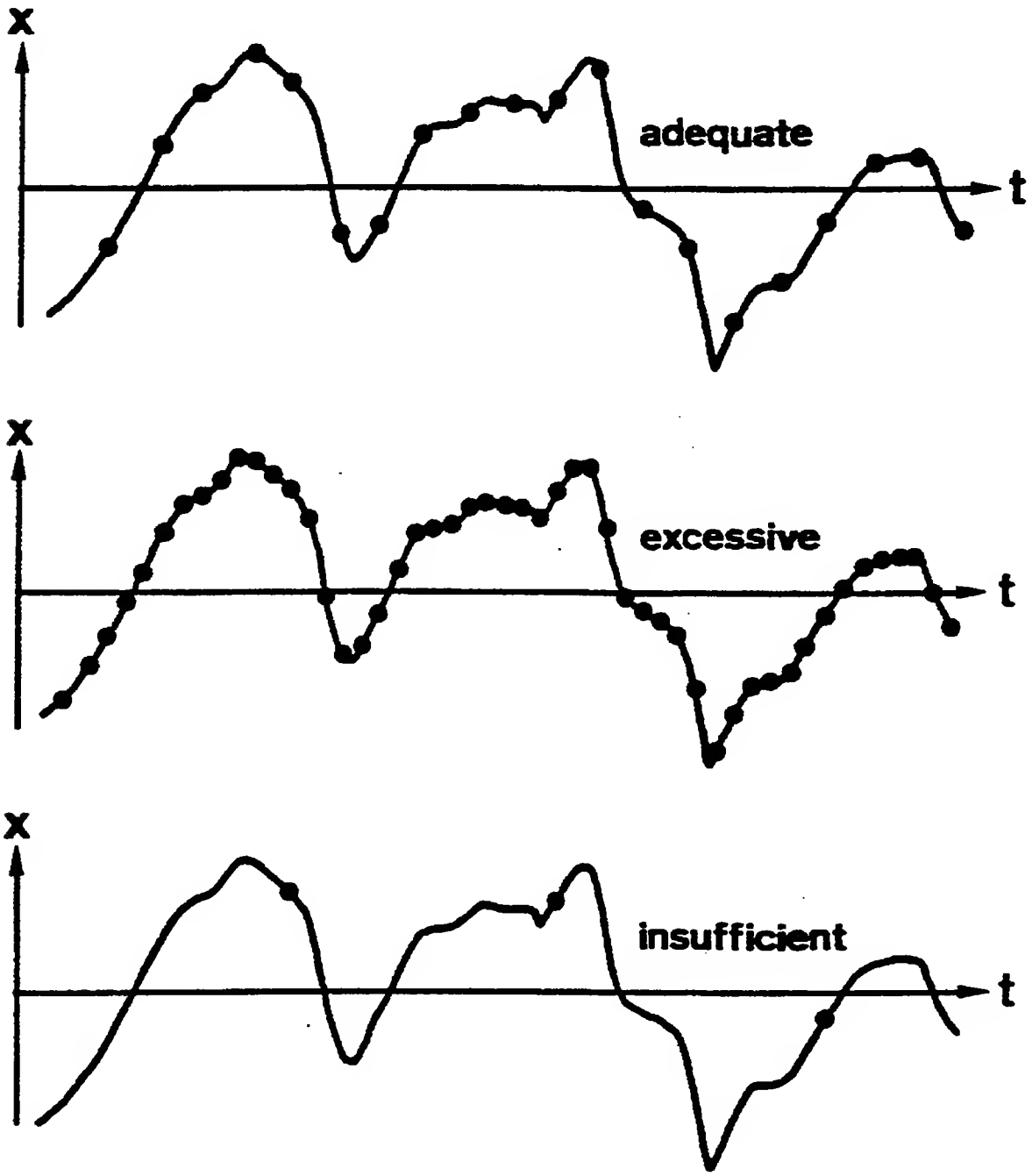
Fig. 4 Sampling Frequency



A different way to determine optimum size and number of primary samples is based on a sampling experiment in which a sufficient number of simultaneous pairs of samples of strongly different quantities are taken.

A rough evaluation of the test results - visualized in Fig. 5 - may already indicate whether much more precision is gained with a larger sample size or not. An exact evaluation can be made applying the sampling theory of Yisman [1]; an example is given in Annex 1.

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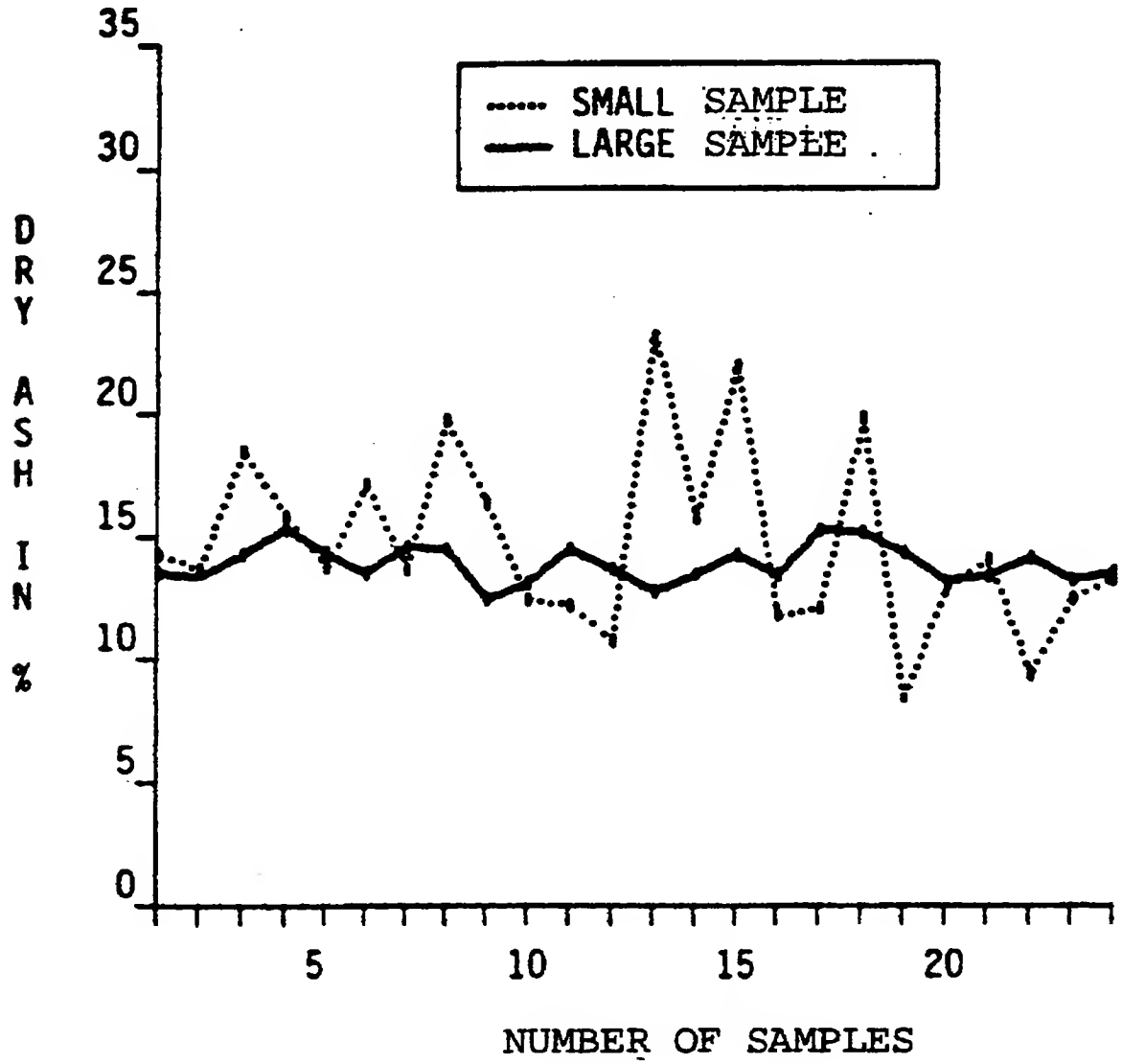
Fig. 5



3. RELIABILITY OF TEST RESULTS

As illustrated in Fig. 6, sampling and testing procedures consist of several independent steps. The reliability of the final test result depends on the reliability of each individual step. Three types of errors can occur at each stage, namely systematic errors, random errors and gross errors. They are characterised as follows:

Systematic errors:

Systematic errors cause a bias of the estimated value for a certain characteristic with respect to the (unknown) true value. They are caused by incorrect (non-representative) sampling and sample preparation, by the malfunctioning of instruments, by inadequate calibration, or by person specific peculiarities in performing the tests.

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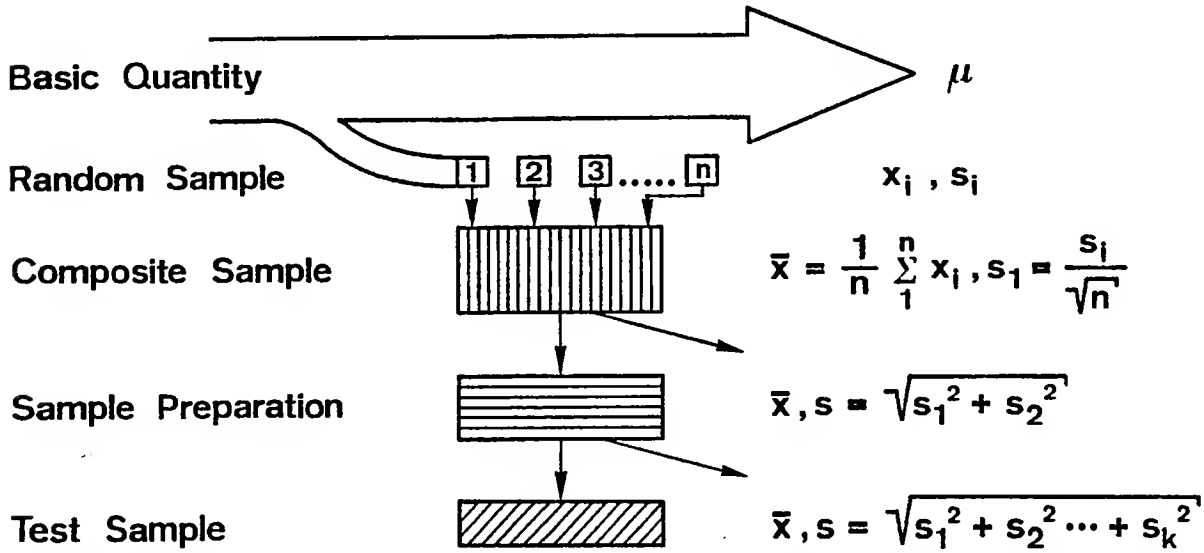
Fig. 6



Systematic errors can be detected by cross-checking with alternative, independent testing procedures.

Examples:

- Campaign of comprehensive manual sampling to test representativity of automatic sampler
- Norm-conform CaOf-determination to cross-check conductometric determination.

Systematic errors can either be of relevance - which requires their detection and elimination -, or they can be irrelevant, usually under the condition that the error is approximately constant.

Example:

- A systematic error in determining the C_3A-content may be irrelevant when producing cement of ASTM Type I, but rather problematic when producing Type II or Type V.

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Random errors:

Random errors arise due to the inherent heterogeneity of the sampled material and due to the imprecision of sampling, sample preparation and analytical techniques. The amount of random errors may be expressed by statistical measures of variability, such as variance or standard deviation (refer to paper "Statistics").

Random errors cannot be eliminated, but they can be reduced by :

Repetition, i.e. extraction of more primary samples, performance of measurements on several test samples, repeating measurements on individual test samples.

The variance (s^2) of the mean value of n measurements is n-times smaller than the variance of a single measurement

$$s^2(\bar{x}) = \frac{s^2(x)}{n}$$

Applying more precise procedures and instruments for sample preparation and analysis

Random errors of a sequence of independent testing activities add up to the total random error of the final result according to the law of error propagation (see Fig. 6).

$$s_{total} = \sqrt{s_1^2 + s_2^2 + ...s_n^2}$$

From this law of error propagation, the following important conclusions can be derived:

- an error which has been committed in the testing procedure cannot be compensated by increased precision in the subsequent testing steps
- the individual errors of the sequence should be as small as possible and of similar size: There is no point in increasing the precision of one step as long as others are decisive for the overall error. Therefore, it is necessary to determine the precision of the individual steps in each control.

Gross errors

Gross errors arise from the wrong performance of procedures, the wrong reading of scales, errors in copying data, miscalculations etc. Such errors can considerably influence statistical evaluations. They may show up as outliers in a series of data.

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Mémoires du Bureau de Recherches Géologiques et Minières No. 67,
Paris 1971

4. ANNEX 1

OPTIMUM SIZE AND NUMBER OF INCREMENTS ACCORDING TO THE THEORY OF VISMAN [1]

The total variance of a sampling and testing procedure is split up into the terms:

$$s^2(t) = \frac{s^2(c)}{am \cdot n} + \frac{s^2(d) +}{n} + s^2(p) + s^2(a)$$

$$= s2(s) + s^2(p) + s^2(a)$$

in which	$s^2(t)$	= total variance of mean value
	$s^2(s)$	= sampling variance
	$s^2(c)$	= composition variance
	$s^2(d)$	= distribution variance
	$s^2(p)$	= preparation variance
	$s^2(a)$	= analysis variance
	am	= size of primary sample (kg)
	n	= number of primary samples

The composition variance and the distribution variance can be determined by means of a sampling experiment, in which a number (~ 30) of pairs of primary samples differing strongly in size are taken and analysed.

$$s^2(c) = \frac{am(1) \cdot am(2) \cdot [s^2(1) - s^2(2)]}{am(2) - am(1)}$$

$$s^2(d) = s^2(2) - \frac{am(1) \cdot [s^2(1) - s^2(2)]}{am(2) - am(1)}$$

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in which	$s^2(c)$	= composition variance
	$s^2(d)$	= distribution variance
	$s^2(1)$	= variance for set of small samples
	$s^2(2)$	= variance for set of large samples
	am(1)	= average mass of small samples
	am(2)	= average mass of large samples

The optimum size of primary samples is then given by

$$m(opt)\frac{s^2(c)}{s^2(d)}$$

and the optimum number of primary samples to obtain a required total variance:

$$n=\frac{s^2(d)+s^2(c)/m(opt)}{s^2(t)-[s^2(p)+s^2(a)]}=\frac{s^2(d)+s^2(c)/m(opt)}{s^2(s)}$$

Example (compiled from [1]):

A coal delivery is sampled by means of 24 pairs primary samples of about 10Og and 50 kg size. The primary samples of small size are entirely dried and ground to analytical fineness. The primary samples of large size are dried, crushed and split and only a partial samples ground to analytical fineness. The ash content is determined in duplicate. The results are given in tables A1 and A2.

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Table A1: SAMPLING EXPERIMENT WITH SMALL SAMPLES

Sample Number	Mass in g	Ash on Dry Basis First	Second	Δ	Mean
1	89	14.20	14.23	0.03	14.22
2	126	13.27	13.66	0.06	13.69
3	152	18.44	18.42	0.02	18.43
4	109	15.87	15.75	0.12	15.81
5	149	13.71	13.81	0.10	13.76
6	87	17.10	17.13	0.03	17.12
7	110	13.64	13.70	0.06	13.67
8	142	19.71	19.64	0.07	19.68
9	111	16.34	16.31	0.03	16.32
10	140	12.49	12.38	0.11	12.44
11	121	12.19	12.21	0.02	12.20
12	150	10.73	10.75	0.02	10.74
13	160	23.24	23.20	0.04	23.22
14	96	15.71	15.80	0.09	15.76
15	105	21.83	21.89	0.06	21.86
16	132	11.78	11.73	0.05	11.76
17	141	12.14	12.05	0.09	12.10
18	123	19.84	19.67	0.17	19.76
19	135	8.34	8.37	0.03	8.36
20	150	12.80	12.85	0.05	12.82
21	108	14.02	13.91	0.11	13.96
22	99	9.35	9.31	0.04	9.33
23	102	12.46	12.38	0.08	12.42
24	129	13.15	13.18	0.03	13.16
Mean	123.9			0.063	14.69
Variance				0.0031	14.1448
Standard Deviation				0.0558	3.7610
Coefficient of Variation					25.6
Variance of the mean					0.5894
Standard deviation of the mean					0.7677
Coefficient of variation of the mean					5.2%

Sampling variance : total variance minus variance of analysis

$s^2(s) = 14.1448 - 0.0031 = 14.1417$

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Table A2: SAMPLING EXPERIMENT WITH LARGE SAMPLES

Sample Number	Mass in g	First	Ash on Dry Second	Basis Δ	Mean
1	53.0	13.61	13.34	0.27	13.48
2	53.2	13.15	13.46	0.31	13.30
3	55.9	14.34	14.20	0.14	14.27
4	41.1	15.33	15.26	0.07	15.30
5	46.1	14.17	14.51	0.34	14.34
6	40.6	13.60	13.41	0.19	13.50
7	48.8	14.76	14.38	0.38	14.57
8	50.2	14.25	14.72	0.47	14.48
9	55.7	12.54	12.29	0.25	12.40
10	48.1	13.02	13.21	0.19	13.12
11	43.4	14.33	14.74	0.41	14.54
12	48.4	13.70	13.63	0.07	13.66
13	45.3	12.80	12.68	0.12	12.74
14	40.1	13.26	13.64	0.38	13.45
15	41.8	14.09	14.41	0.32	14.25
16	48.0	13.52	13.28	0.24	13.40
17	56.2	15.44	15.11	0.33	15.28
18	50.3	15.07	15.28	0.21	15.18
19	48.0	14.40	14.26	0.14	14.33
20	44.3	13.06	13.22	0.16	13.14
21	48.5	13.22	13.47	0.25	13.34
22	41.6	14.26	13.90	0.36	14.08
23	39.9	13.18	13.12	0.06	13.15
24	52.3	13.44	13.56	0.12	13.50
Mean	47.53			0.241	13.87
Variance				0.0455	0.7903
Standard Deviation				0.2134	0.8890
Coefficient of Variation					6.6
Variance of the mean					0.0329
Standard deviation of the mean					0.1815
Coefficient of variation of the mean					1.3%

$s^2(s) = 0.7903 - 0.0455 = 0.7448$

calculation of composition and distribution variance:

$$s^2(c) = \frac{123.9 \cdot 47530 \cdot (14.1417 - 0.7448)}{47530 - 124} = 1664.2$$

$$s2(d) = 0.7448 - \frac{123\text{-}9 \cdot (14.1417 - 0.7448)}{47530 - 124} = 0.71$$

Hence:

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Optimal mass m(opt) = $\frac{1664.2}{0.71}$ = 2343g 2.3 kg

(entire primary sample can be prepared to analytical fineness, thus only analytical variance to be considered)

Optimal number of primary sample:

$$n = \frac{0.71 + 1664.2/2343}{s^2(t) - s^2(a)} = \frac{0.71 + 0.71}{s^2(s)} = \frac{1.42}{s^2(s)}$$

Error of mean (95% conf intev.)	Coeff. of Variation	s	s2(t)	s2(s) = s2(t - 0.0031	n
± 10%	5%	0.7	0.49	0.4869	3
± 5%	2.5%	0.35	0.12	0.1194	12
± 2%	1%	0.14	0.02	0.0165	86
± 1%	0.5%	0.07	0.0049	0.0018	(800)*

* Analytical variation outweighs sampling variation, i.e. error of mean is determined by analytical precision and not by sampling regime.

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Sampling Methods and Equipment

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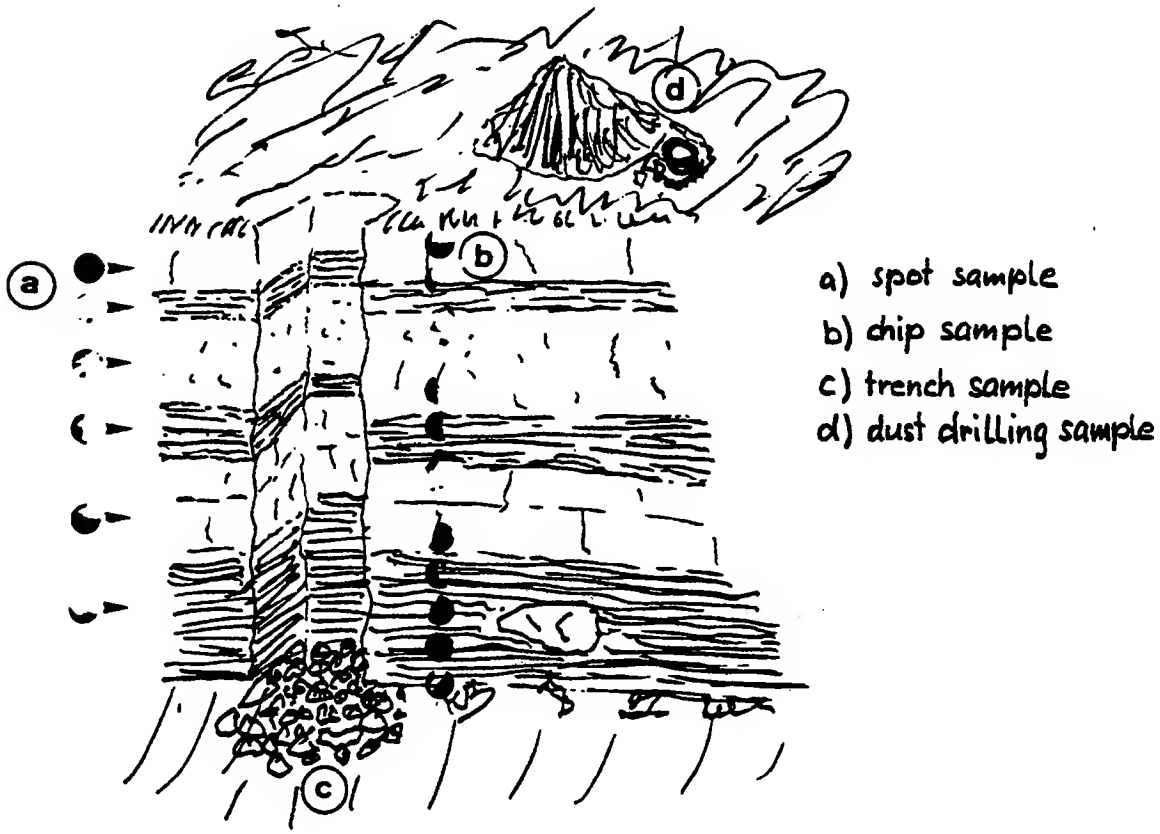
1. SAMPLING OF STATIONARY MATERIALS

Sampling of stationary material should, whenever possible, be replaced by sampling on conveyed material. In a number of cases, however, sampling of stationary material cannot be avoided. Such cases are reviewed in the following:

1.1 Field sampling

The mineralogical and chemical composition of sedimentary rocks generally vary far more rapidly in the vertical direction (stratigraphic sequence) than they do in the horizontal direction of the layer; therefore a vertically oriented sampling is required.



1.1.1 Surface sampling

In the sequence of increasing reliability, the following methods of surface sampling on an exposed face can be applied:

a) unweighted spot sample
for each rock layer of a rock complex only one sample is taken at a definite spot.

Information: only a rough judgement of the investigated material possible.

aa) weighted spot samples
more reliable information is achieved as the spot samples of each layer are weighted up according to their stratigraphic thickness.

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b) chip sample
small material samples are broken out of the wall face at regularly spaced intervals.

Information: reliable information on the chemical and mineralogical composition.

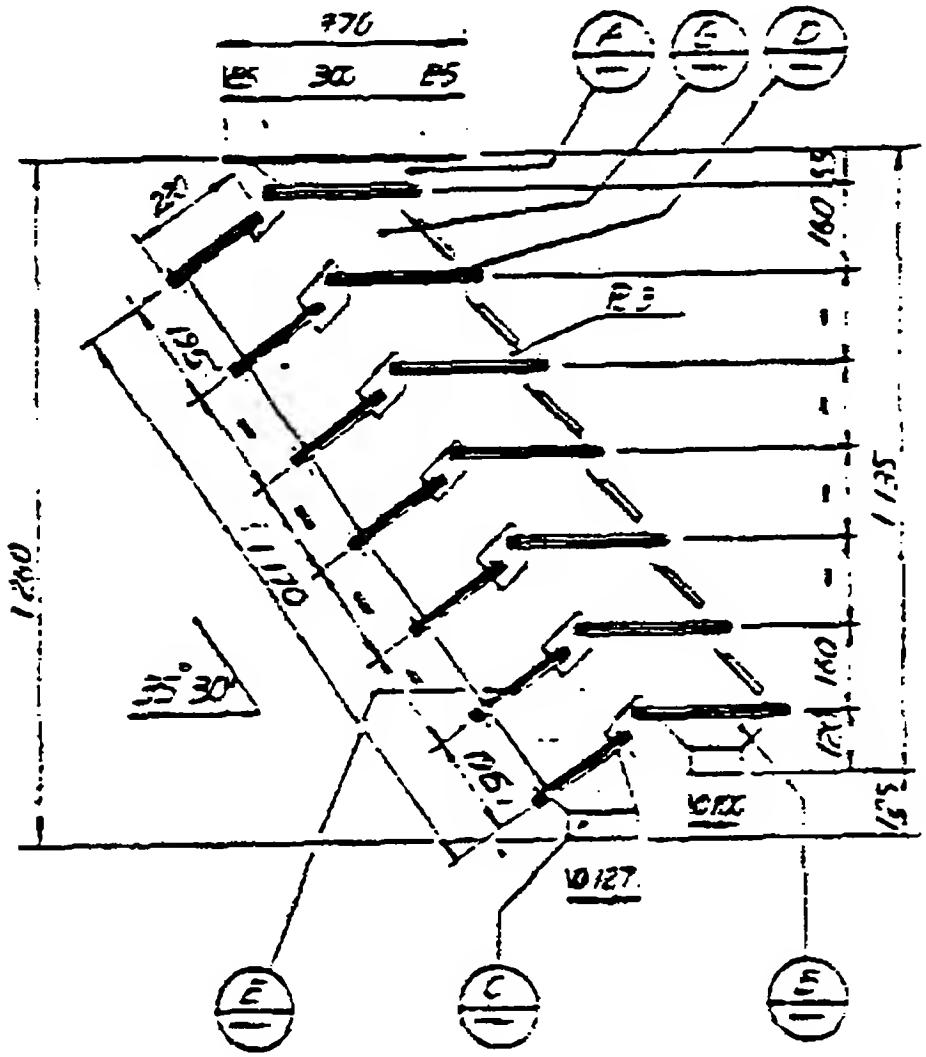
c) trench sample
the sample is obtained by cutting a trench of uniform width and depth across the face of the exposed rock and collecting the excavated material.

Information: reliable information on chemical and mineralogical composition as well as on moisture content.

1.1.2 Drill hole sampling

- Dust drilling sample. Mainly hard, compact dry material is recovered from hammer, tri-cone or rotary drilling.

Proper sampling requires a dust collector attached to the drilling equipment. The often encountered procedure of dust collection around the borehole is often a source of systematic errors due to the loss of the fine particles.



Core drilling sample

the core sample is obtained by rotary drilling with core barrel tipped with a steel or diamond bit. This method can also be applied below the water table. A representative sample for chemical-mineralogical investigations can only be guaranteed on a core recovery of over 90%. The cores can also be used to determine the technological properties of the rock.

1.2 Sampling of bulk cement

In addition to sampling at the material surface, it is also required to collect material from the interior using a tube sampler. For cement sampling tube samplers are often specified in the standard specifications e.g. ASTM C183.

2. SAMPLING OF CONVEYED MATERIALS

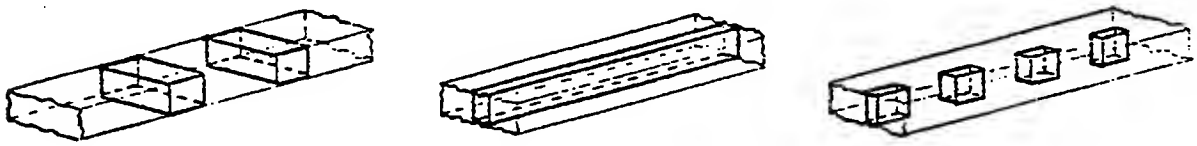
This term applies to all types of material which are transported continuously; for instance gravel on a belt conveyor, meal in an airslide or slurry in a slurry pipe.

There are three principal ways in which a material can be taken from a material flow:

1) Transverse sample consists of the whole of the material flow during short, predetermined time interval.
2) Longitudinal sample consists of a part of the material flow which is continuously deflected from the main stream.
3) Quasi-point sample is created by the branching off of a part of the material flow during a certain period of time.

A visual impression of these three categories can be obtained by imagining the samples taken from a stationary belt conveyor.

or drawn 3 - dimensionally:



2.1 Evaluation of the 3 categories

2.1.1 Transverse samples

This sampling method is, in principle, correct as the average value of the individual collected samples is almost the true average value of the "lot". The term "lot" should be understood as that quantity to which the measurements refer, as for example the heap of a mixing bed, a shipload, a rail wagon, a silo etc.

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2.1.2 Longitudinal samples

If all particle sizes were represented in their correct relationship in that part of the stream sampled, this type of sampling would yield a true picture of the "lot". In reality, however, the force of gravity causes a segregation of the particle sizes to a lesser or greater extent, and therefore excludes the fulfillment of the conditions mentioned above.

2.1.3 Quasi-point samples

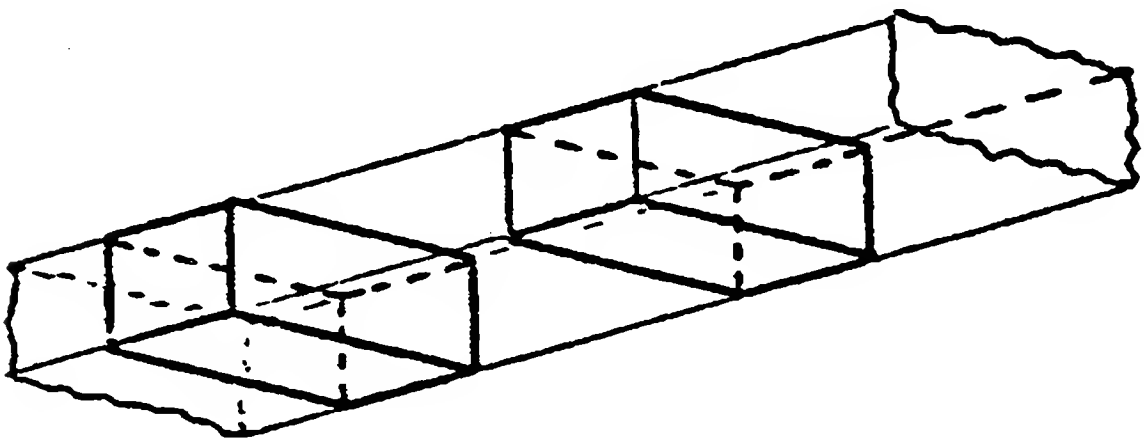
This type of sample is only a combination of the preceding two types. It incorporates no new advantages, but embodies the disadvantages of both the other categories.

The transversal sample therefore appears to be the only one promising success. It is the only one that can fulfill the condition of having an equal chance of sampling all the particle size fractions.

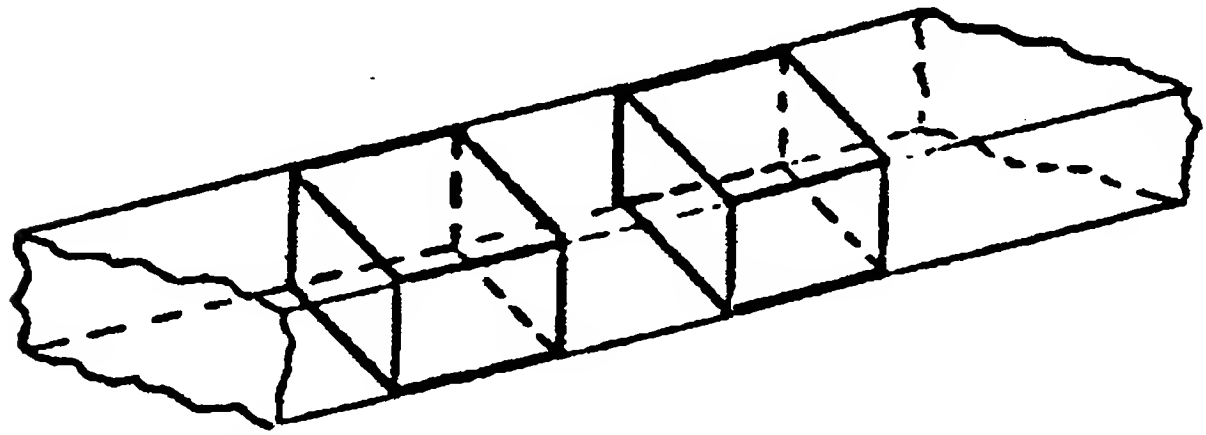
2.2 Various possibilities of collecting a transverse sample

The various possibilities are illustrated below in the form of a three-dimensional illustration of the material flow.

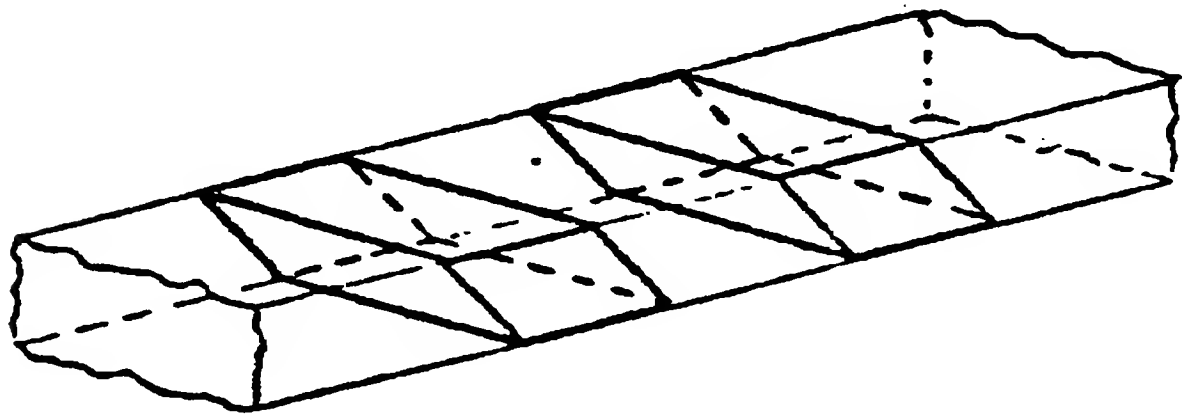
A) Rectangular cut



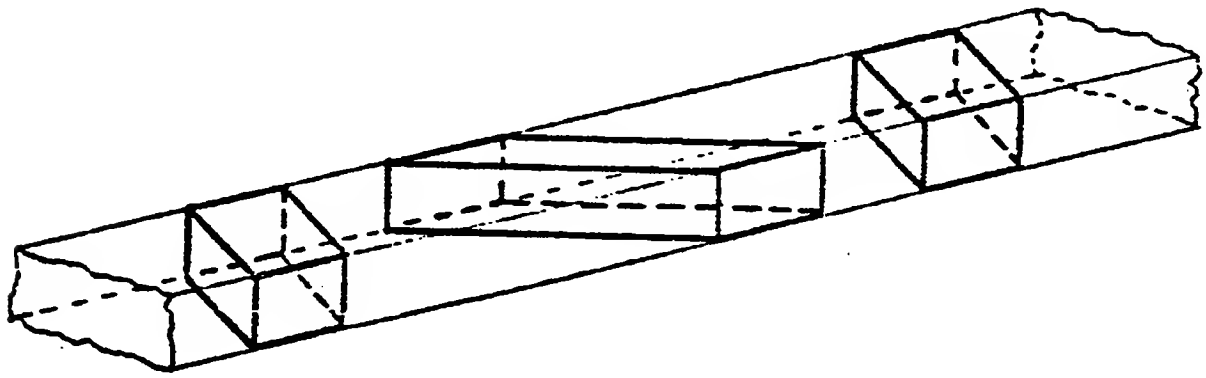
B) Horizontal inclined cut



C) Vertical inclined cut



D) Alternating horizontal inclined cut



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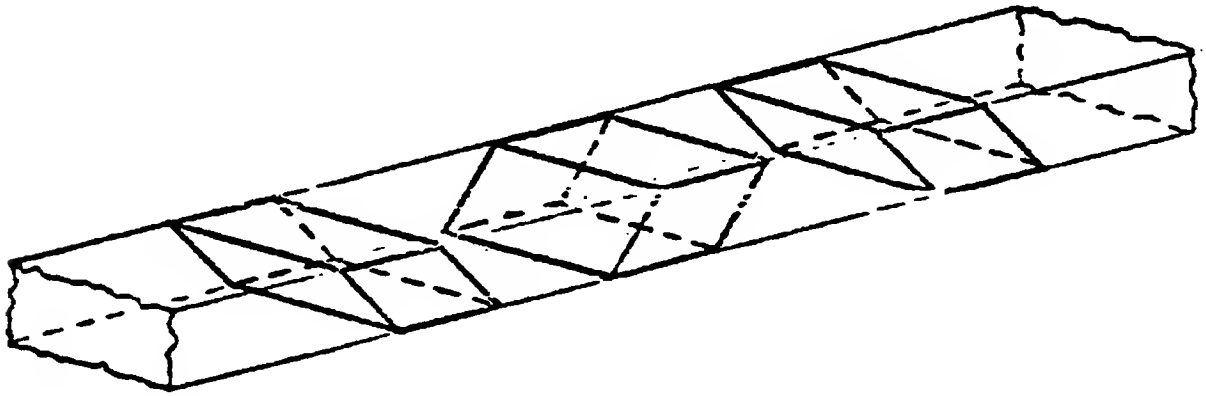
E) Alternating vertical inclined cut



From a theoretical viewpoint, these five cuts are of comparable value. The differences lie in the ease of their practical application.

Type A): Can only be realised on a stationary conveyor belt or by means of a travelling sampler.

Type B): Can be realised by means of a rotating sampler. The sampling aperture always moves in the same direction through the material.

Type C): A bucket wheel sampler belongs to this type.

Type D): This is the most commonly found type of equipment. The reciprocating slit vessel and the reciprocating chute types are typical examples.

Type E): This can only be realised by a radially mounted slit vessel; this type of equipment is generally too large.

2.3 Deviations from the ideal case

1) The particles do not drop one behind the other, but in a more or less compact stream and can therefore influence each other.
2) The particles make contact, not with a stationary separating wall, but with a moving one and can therefore be deflected at the moment of impact.
3) The opening of the sampler is not of an infinite size. Processes on the one separating wall can therefore be influenced by those on the opposite one.
4) The separating walls are not infinitely thin and can be bent in the course of time.
5) The particles themselves can break on impact. This would be of importance only when a screen analysis is to be prepared from the sample.

The design of samplers must be aimed at overcoming all these difficulties. What form these solutions take and what important rules must be respected, is described in the next chapter.

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2.4 Rules for the sampler

(1) Slit width and speed are inter-dependent.

Width of slit $B \geq 3\ d + 10$ mm
d = diameter of the largest particle.

This ensures that even for fine meal, the width of the slit does not fall under 10 mm.

Speed $w \geq 300$ mm/s
In special cases this can be increased to 400 mm/s, provided the slit width is increased to 4 d + 10 mm.

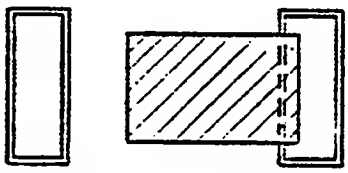
The conditions for the slit width must be respected for all slit dimensions.

(2) The edges of the sample apertures should be fitted with interchangeable sacrificial elements made of a wear resistant steel.

(3) The slit aperture must transverse the entire cross sectional area of the material flow. The return and stopping positions must be placed outside of this area.

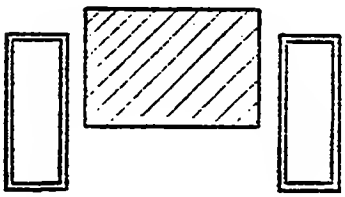
right

horizontal section

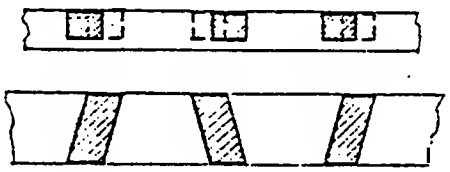


wrong return position

horizontal section



material flow is not entirely cut



vertical and horizontal section

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(4) The speed of the sampler during its journey through the material must be constant, during any single traverse as well as from one traverse to another.

- Quick starting electric motor with excess capacity. This drive is more suitable than a pneumatic or hydraulic cylinder, as the latters speed cannot be easily regulated.
- An acceleration zone must be provided outside the material stream.

Fig. Motor starts too slowly

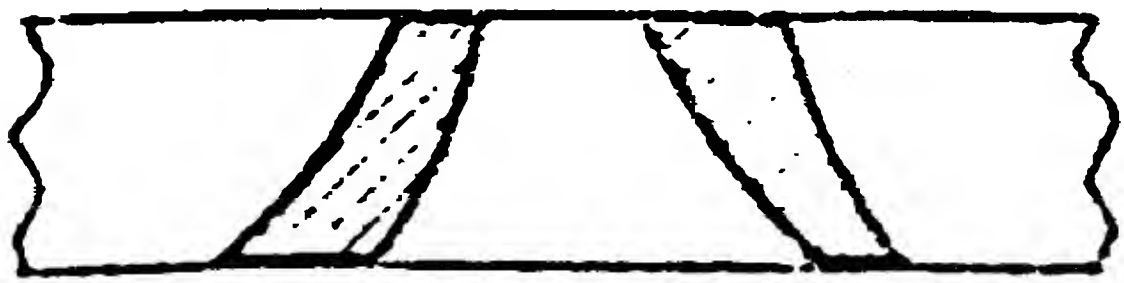
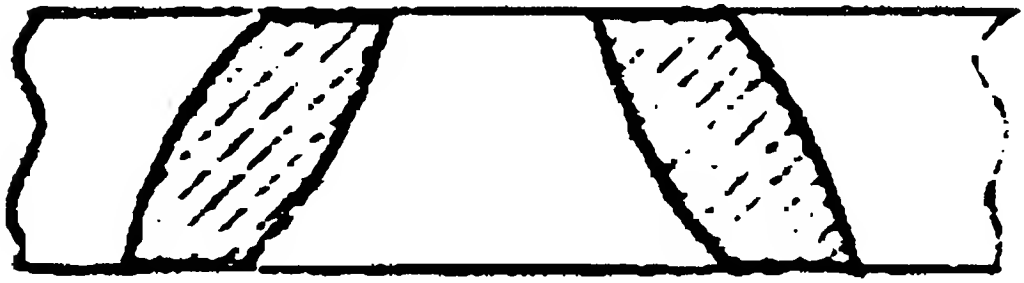


Fig. Motor too week



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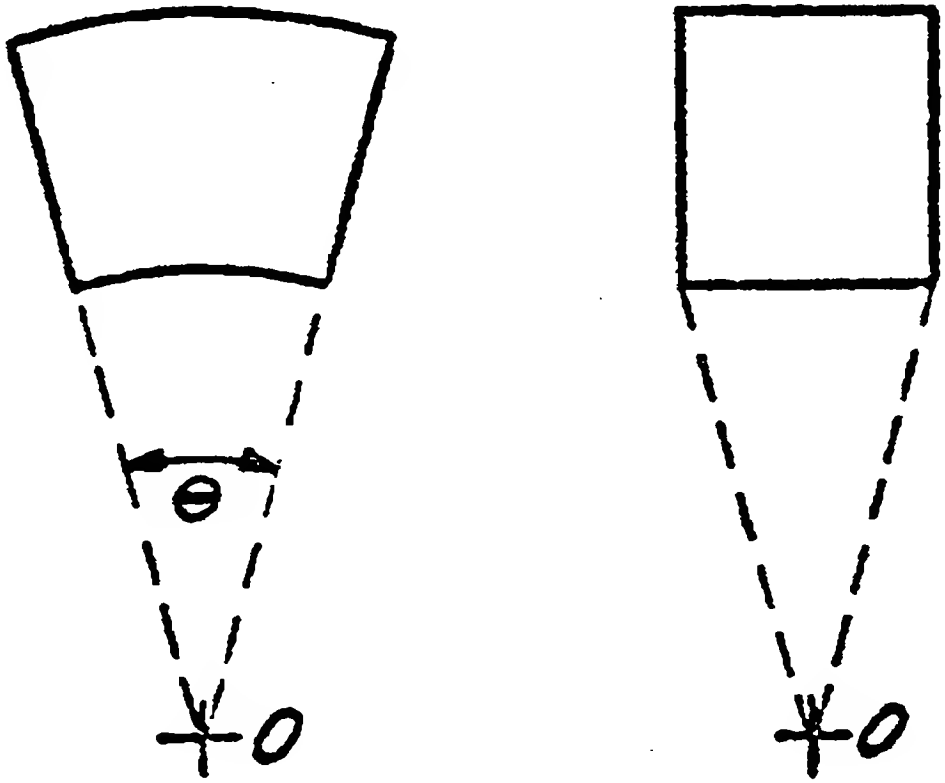
(5) The shape of the sampler aperture must be suited to the movement.

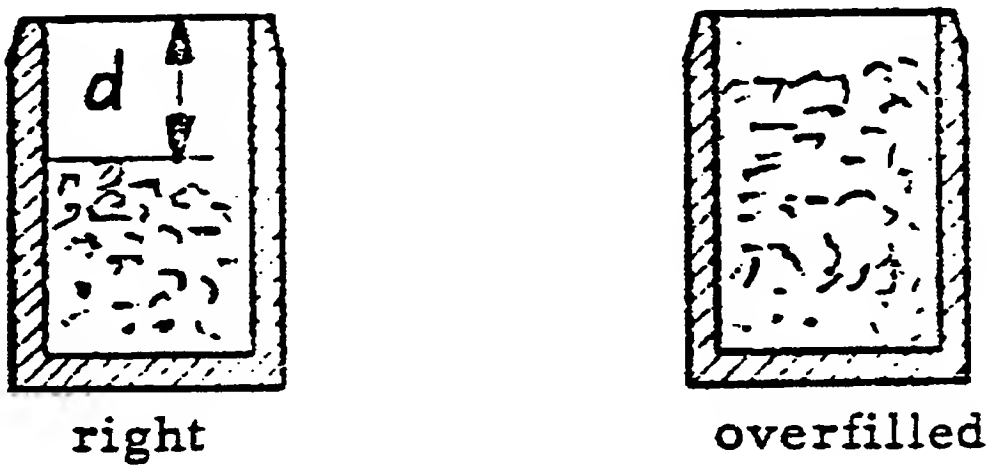
linear movement ⟶ the side walls must be parallel

rotating movement ⟶ the side walls must be positioned radially.



(6) Measures must be taken to avoid blocking and overflowing of the sampler.

Slit vessel cutter:
The capacity of the vessel should be twice as large as the nominal sample volume. In the filled vessel there should be a remaining free height equivalent to d or at least 100 mm.



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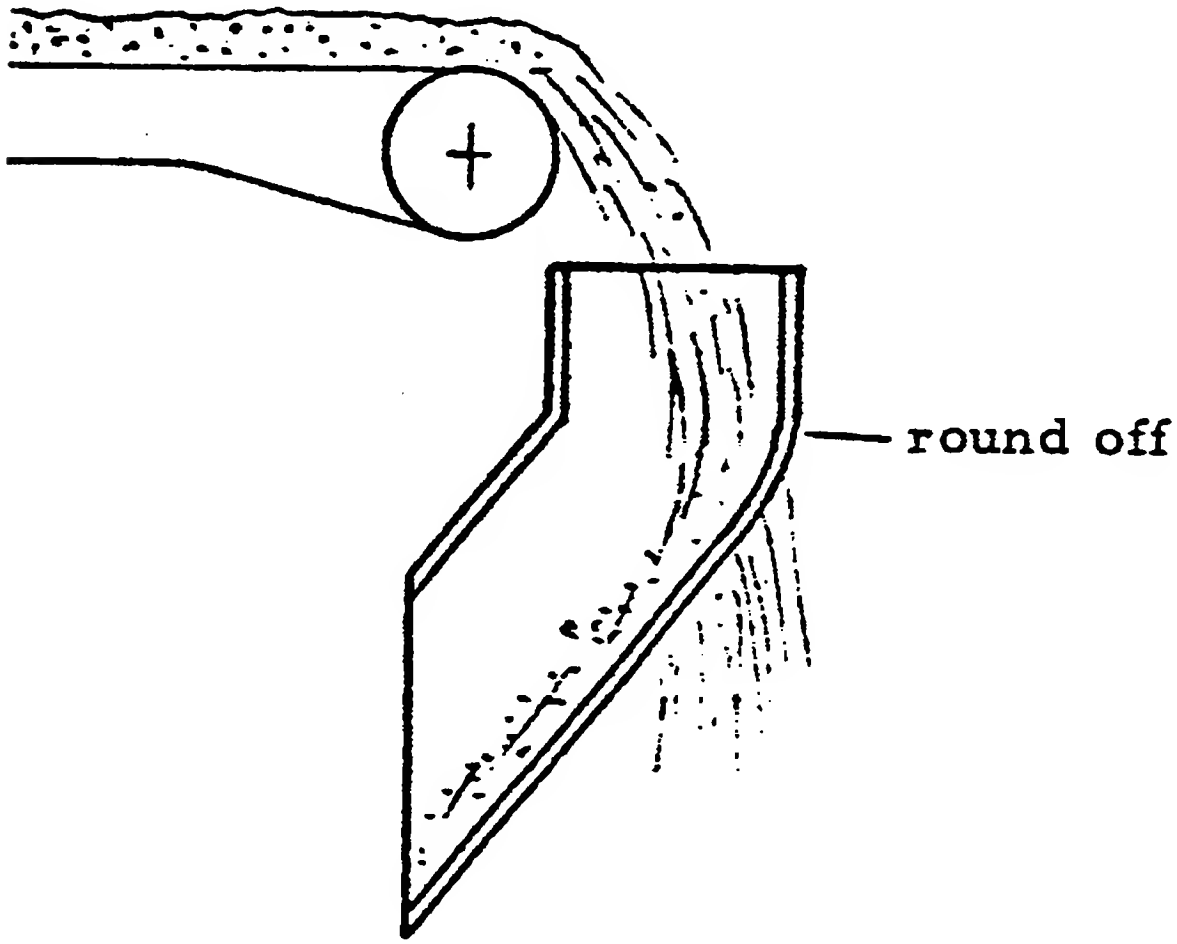
Chute type cutter:
The vertical walls must be of sufficient height to prevent the "splashing" of particles. To ensure a flawless material extraction, the inclination of the walls must be chosen to suit the properties of the wettest materials encountered. Care should be taken that there are no obstructions. Corners, joints and protruding screws are often the cause of build-ups. Stainless steel should be used for sticky materials and if necessary provision for heating should be made.



(7) The integrity of a sample must assured. Once a sample has been taken, it must be protected against all alterations, especially from the loss or addition of particles.

Loss of dust:
To reduce the drop height, the sampler should be mounted as close to the discharge point as possible. To avoid the formation of dust, the material can be slightly moistened before it reaches the sampler.

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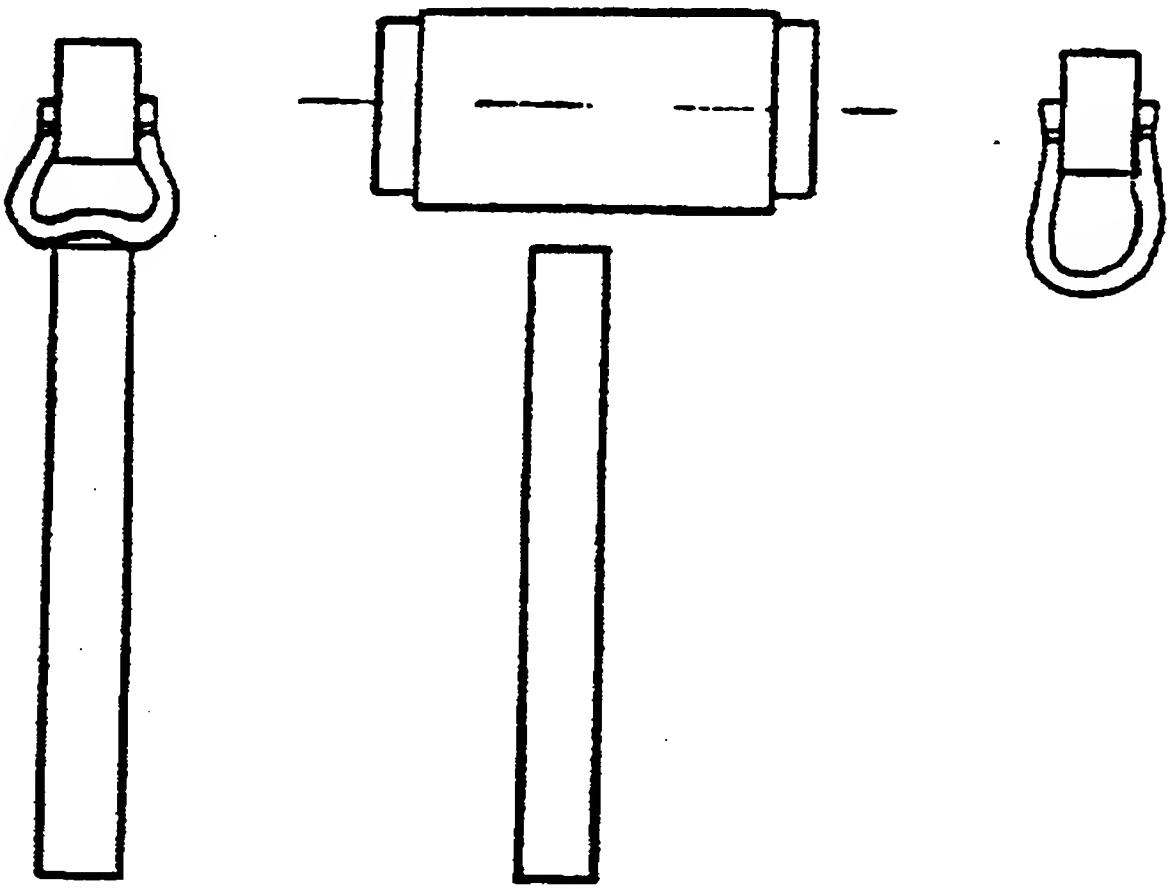
Addition of dust:
The whole sampling apparatus should be protected against dust. The aperture in the sampler can be closed by a lid in its static position.



(8) Calculation of the single sample volume m.

straight movement:

$$m = M\frac{B}{w}$$

M = material flow [kg/s]
B = slit aperture [m]
w = speed [m/s]

rotary movement :

$$m = M\frac{\theta}{\varphi}$$

θ = aperture angle [red]
φ = angular speed [rad/s]

Note

A sampler designed in accordance with the previous rules has a lower rated capacity. It would be wrong to try to decrease the sample quantity by an increase in speed or y a reduction of the slit width.

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2.5 Recapitulation of sampling rules

Fundamental principles which must be adhered to.	Principle No. 1 All sectors of the material stream must be sampled for an uniform time.	Principle No. 2 All particle sizes must have the same chance of being sampled.	Principle No. 3 There must be no alteration to the sample once it has been taken.
Measures to maintain the adherence to the fundamental principles.	1) Correct shape of the sampler aperture. 2) The whole of the material flow must be covered. Stopping points must lie outside the material flow. 3) The speed must stay constant during the traverse and from traverse to traverse.	1) Minimum slit size 3d + 10mm 2) Speed generally less than 300 mm/s 3) Sampling to take place at a location where the fine particles are not thrown up as dust.	1) Prevention of blockages and overfilling. 2) No material loss. 3) No material additions. 4) Protection against any alteration of the composition.

3. CRITICAL ANALYSIS OF THE BEST KNOWN SAMPLERS

3.1 Apparatus for transverse sampling with linear movement

3.1.1 Sampling from a stopped conveyor belt

The sample is separated from the remaining material by a frame. This system is the closest approach to the theoretical model of a right-angle cut. As the material is stationary, L only needs to conform to the condition of L = 1.5 d + 100 mm. The minimum sample therefore is smaller.

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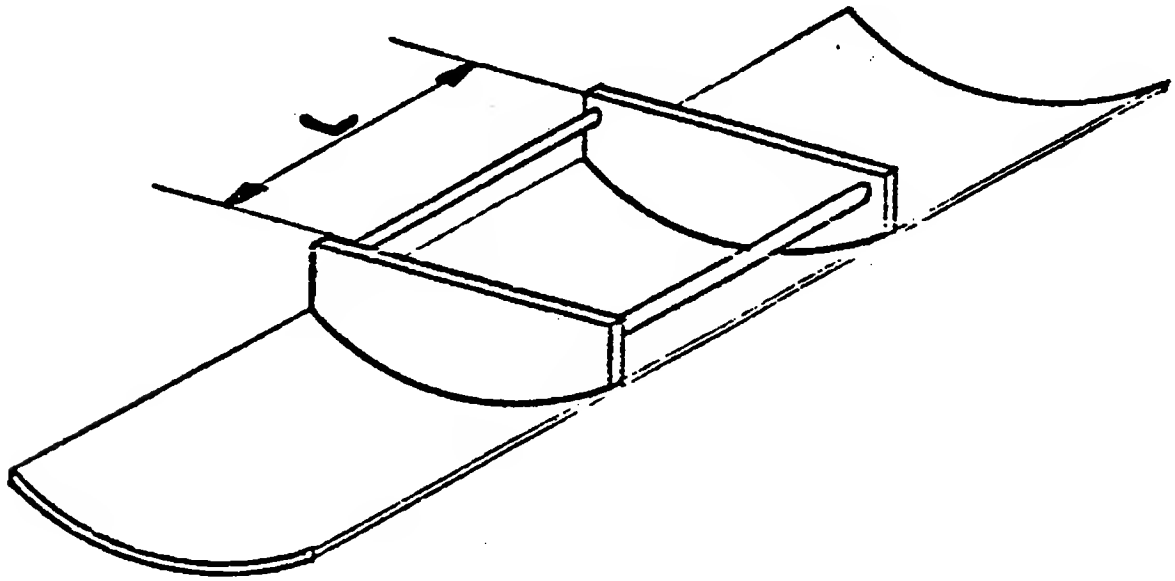


$$m1 = \frac{M}{v}(1.5d + 100mm)$$

m1 = minimum sample from the stationary belt

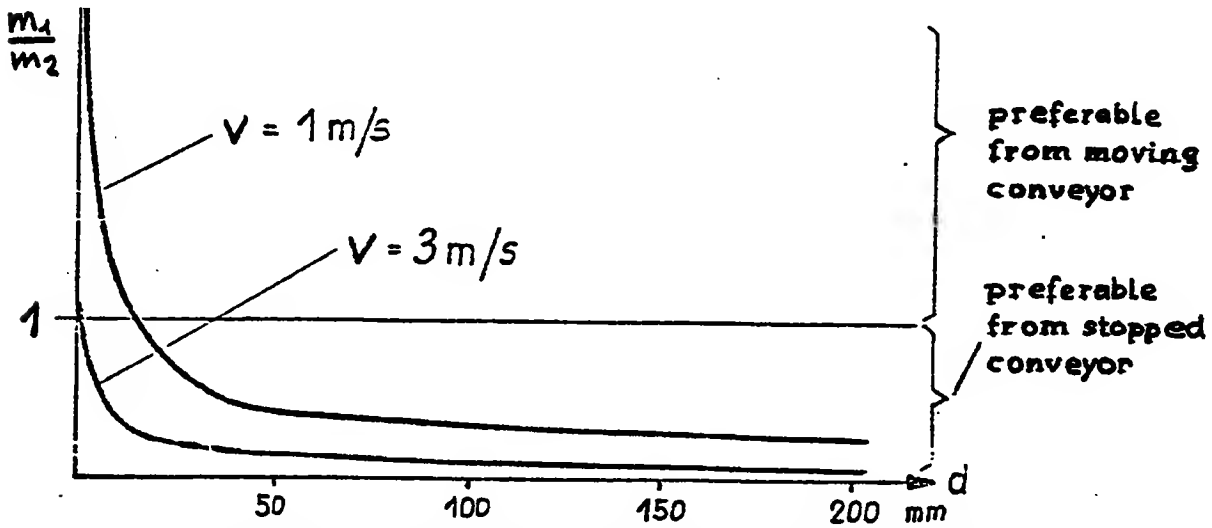
$$m2 = \frac{M}{w}(3d + 10mm)$$

m2 = smallest sample at material discharge point

$$\frac{m1}{m2} = \frac{w\,1.5d + 0.1}{v\,3d + 0.01}$$

v = belt speed

This function for w = 0.4 m/s appears as follows:



It can be observed that with larger particle sizes and at higher belt speeds the minimum sample quantity tends towards the sample from the stationary belt.

In practice it is, however, impossible to stop large conveyor belts for the purpose of sampling.

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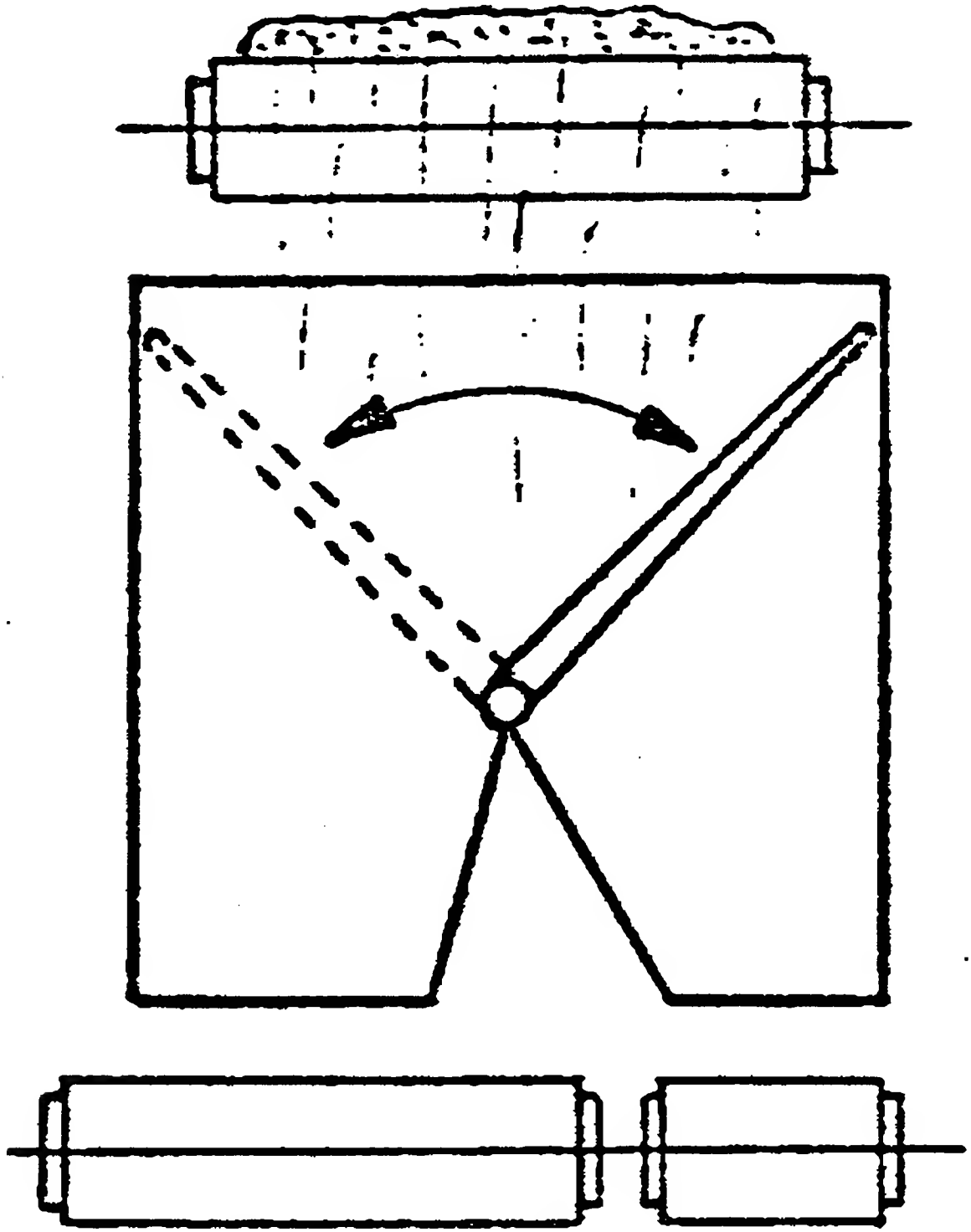
The sketch below shows a set-up which makes it possible to extract samples from a conveyor, fully automatically, without having to stop the main conveyor.

Fig. Sampling belt



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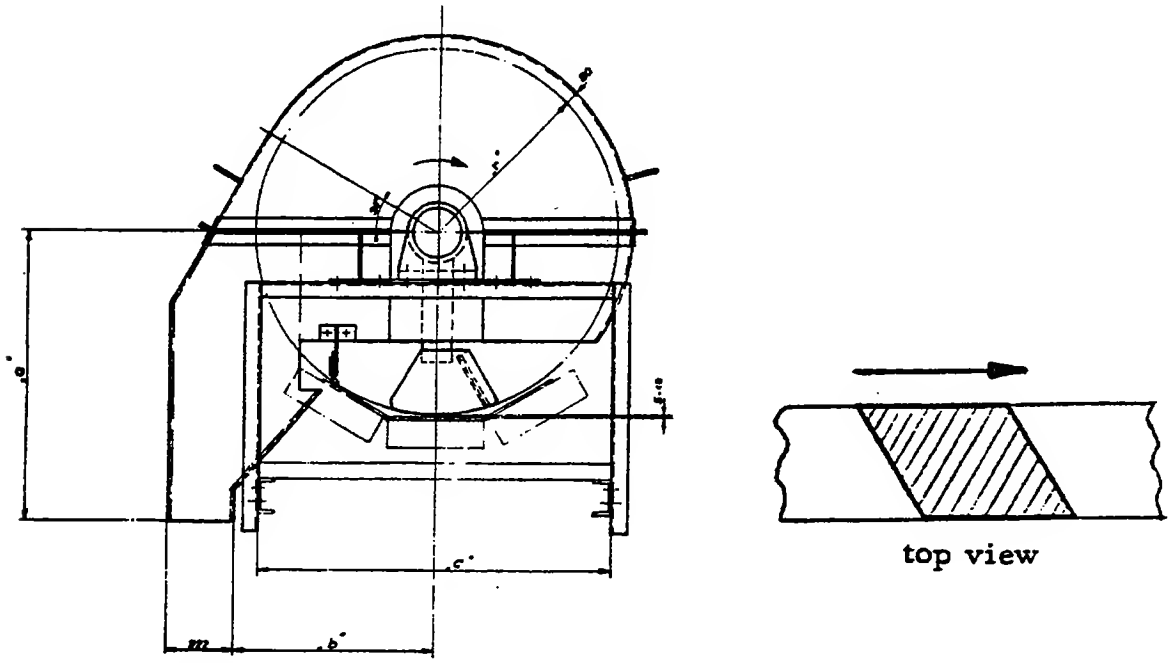
For a certain pre-set time, the material is deflected by the flap onto the sample belt. The sample belt is stopped after the return of the flap. The sample can now be taken off the sample conveyor and the process can be easily automated. The speed of the sample conveyor can be freely selected and kept high. This makes it possible to reduce a sample down to 1/lOth of the size of those from a sampler at the discharge end.

3.1.2 Hammer samplers



This equipment takes samples from a running conveyor. The hammer is designed as a scoop to ensure that the material is not Pushed aside. Brushes fixed to the end of the hammer clean the fine material from the belt. This type of sample is well-suited for non-sticky material.

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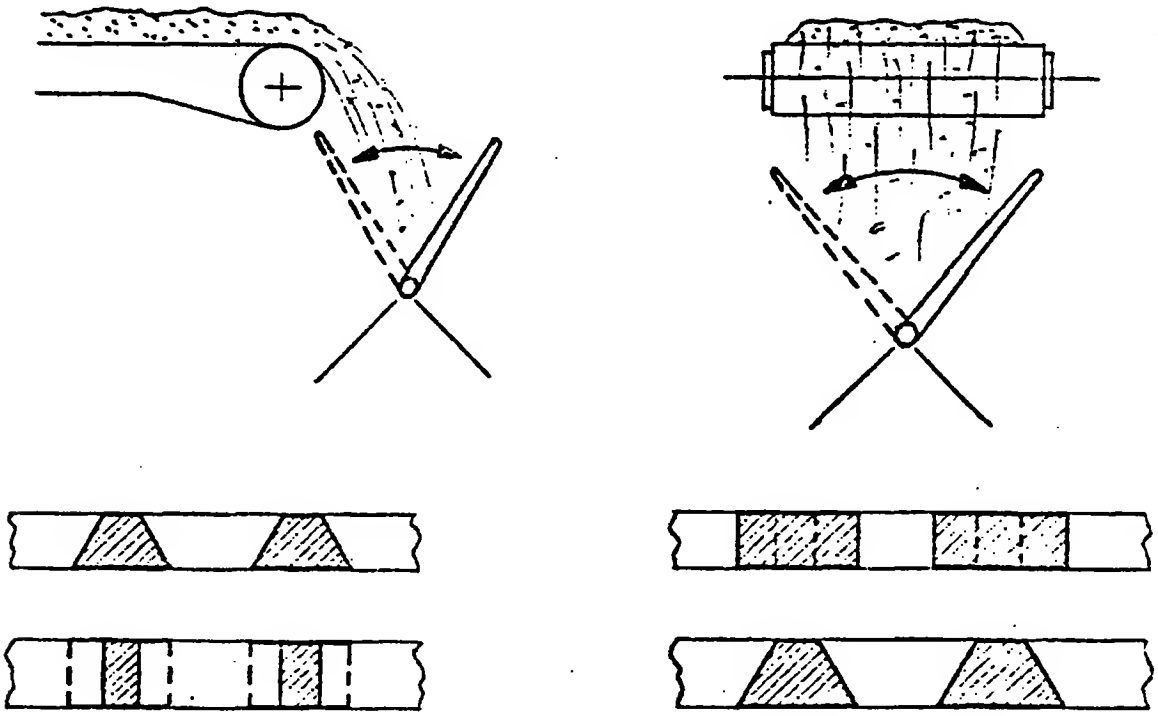
3.1.3 Sampler with flaps



The sectional illustrations show that the principle of uniform sampling of all material layers is not maintained. One might think that this systematic error could be avoided by a faster operation of the flap and by a longer residence time in the sampling position. In the first case, an unwelcome violent movement of the material would result whilst in the second case, the sample quantity will be very large or, with longer intervals, the integration error becomes unacceptable.

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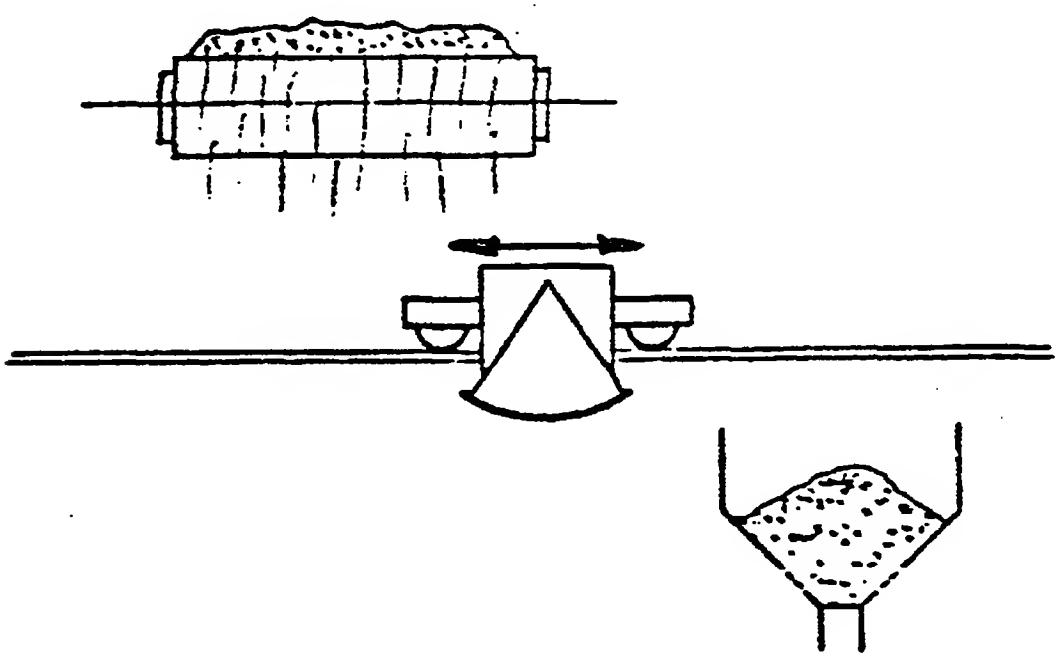
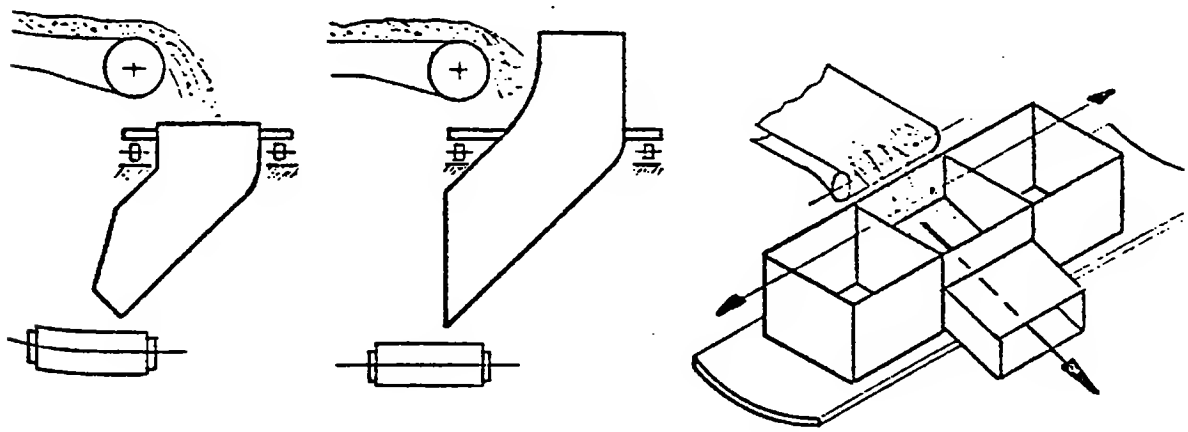
3.1.4 Slit vessel - sampler

Properly designed equipment of this type can fulfill all conditions.



3.1.5 Chute sampler

The chute sampler, like the slit vessel sampler fulfills all require meets. It is suitable for the sampling of slurry as well as of dry material. The chute is of a simpler design than the slit vessel, but requires more headroom.



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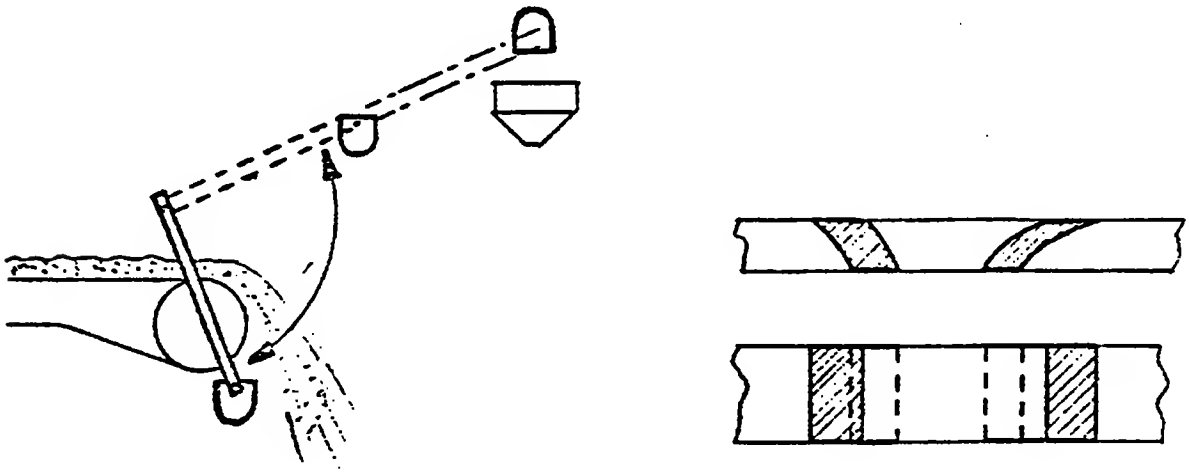
3.2 Equipment for Transverse Sampling with a Rotary Motion

3.2.1 Slit vessel on a radial swing



This type, when suitably adjusted for the materials properties, is precise and produces no systematic errors. The equipment is of excellent design by virtue of the small drop height of the material and allows the possibility to remove the sample immediately.

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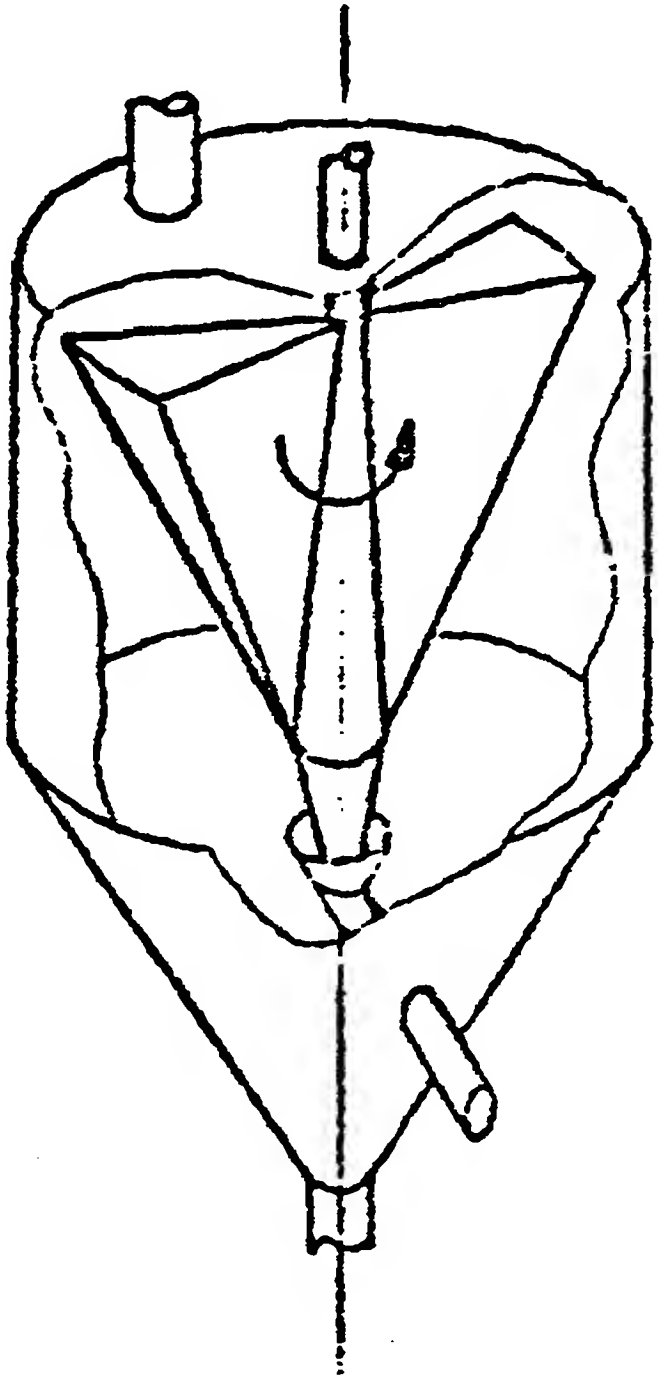
3.2.2 Vézin - Sampler



The design concept of this equipment is correct. It is unsuitable for installation at a discharge point, but rather for small material flows of relatively fine grained dry solids or for slurries. One advantage is the good separation of the sample from the remaining material.

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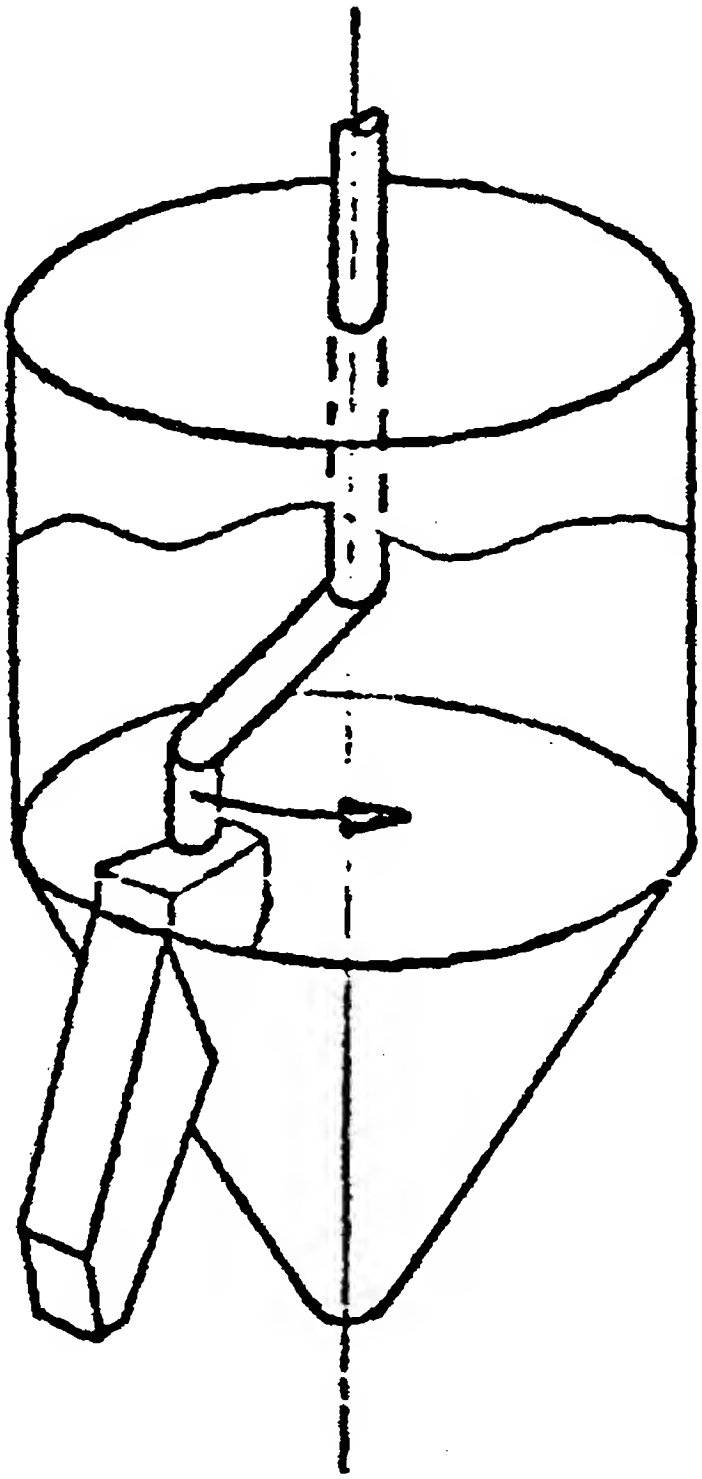
3.2.3 Sampler with rotating distributor



This type has been developed from the Yezin sampler by means of exchanging the rotating and stationary parts, and is therefore correct in its concept and principle. Due to the relatively high position of the distributor tube, the material reaches the sampler aperture at a high velocity. This is unfortunate when taking the sample and creates some dust. Furthermore, a demixing process does take place in the drop tube due to the considerable centrifugal forces present.

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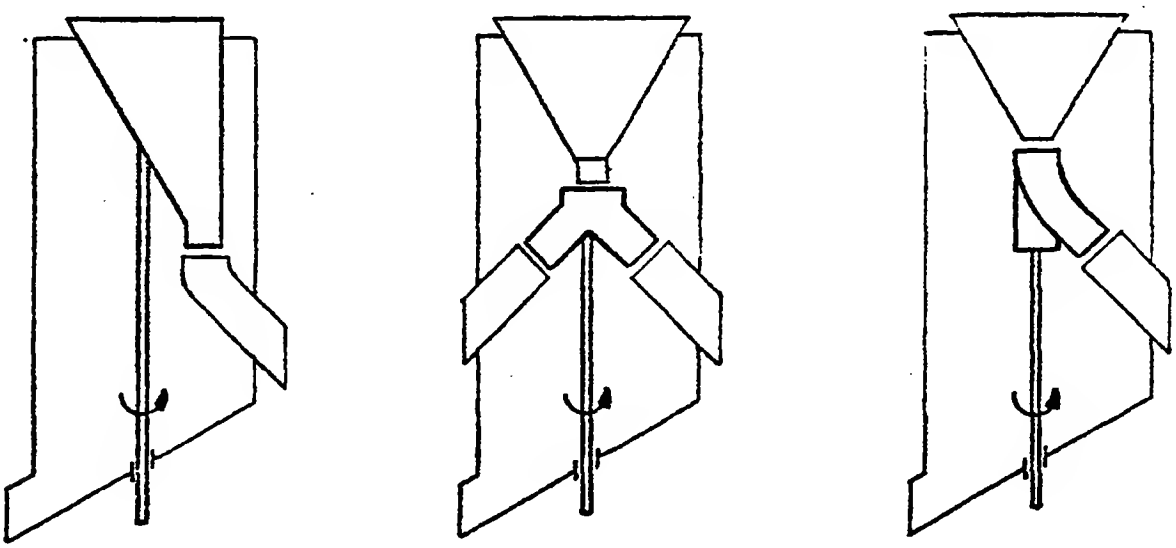
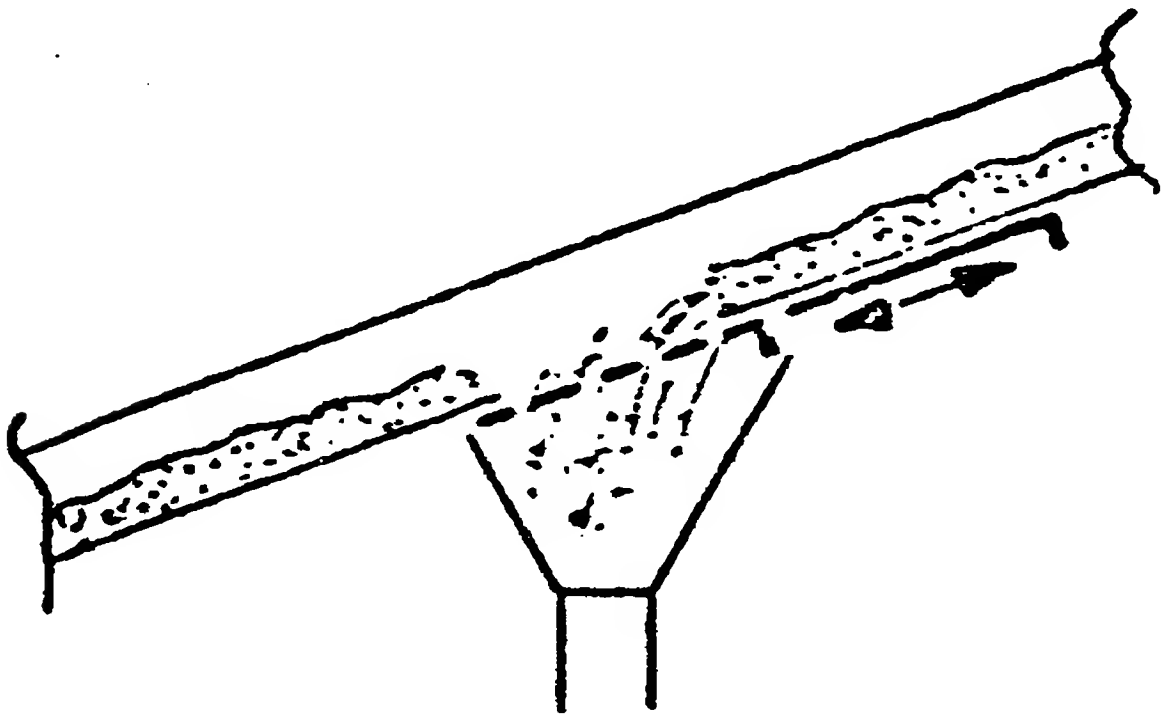
Further variations of this type are shown in the sketches below.



All these types suffer from the disadvantage that the whole mass of the undivided sample has to be accelerated.

3.2.4 Slides and flaps in ducts



In the case of the transport of bulk materials having a low bed depth, this sampling equipment may be sufficient.

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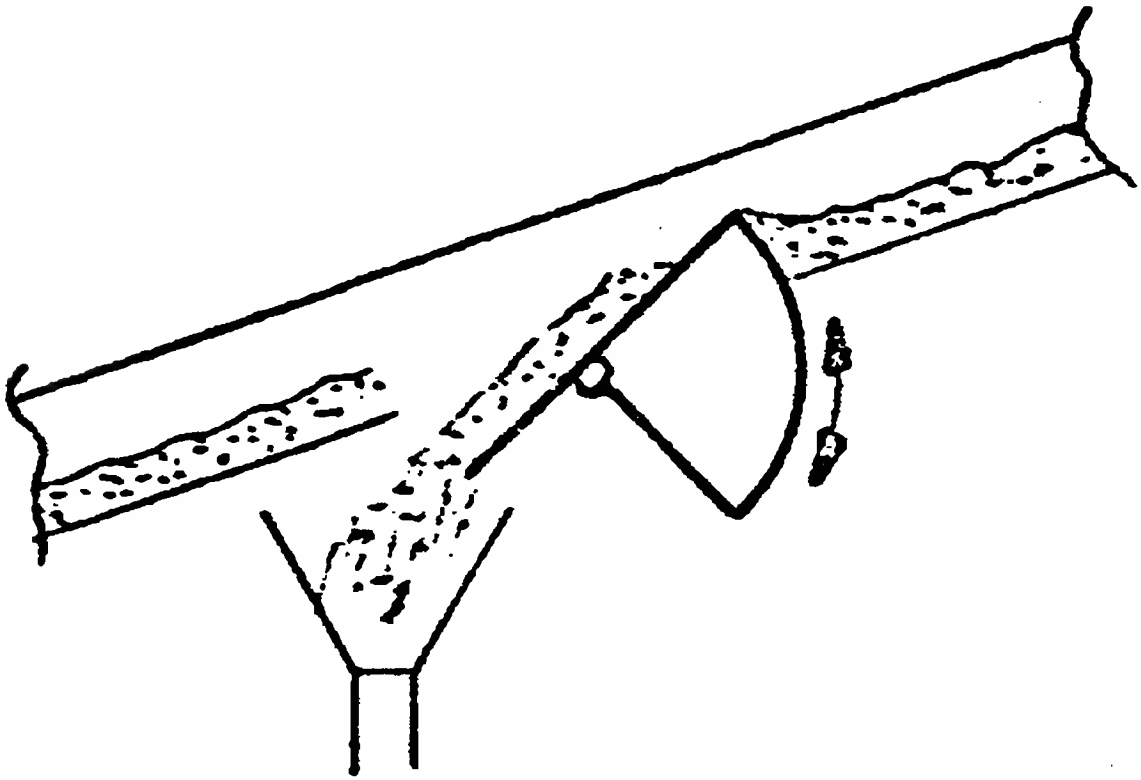

The material lying on the flap is extracted whilst the oncoming material is stopped during the sampling. Provided the material does not stick, and provided that the dimensions have been well-chosen, this sampler is just right and delivers reproducible samples.

3.3 Equipment for Longitudinal Samples

3.3.1 Multi-stage sample divider

This type of equipment has been extensively used in the past and is still in frequent use to reduce the quantity of samples. It consists of several vertically orientated samplers. That alone has two disadvantages:

1) The equipment requires considerable height for its installation and
2) the dust which is created by the partial free fall of the material is ducted upwards due to the chimney effect of the closed vertical casing, so that the sample on the bottom is lacking in the finest particle sizes.

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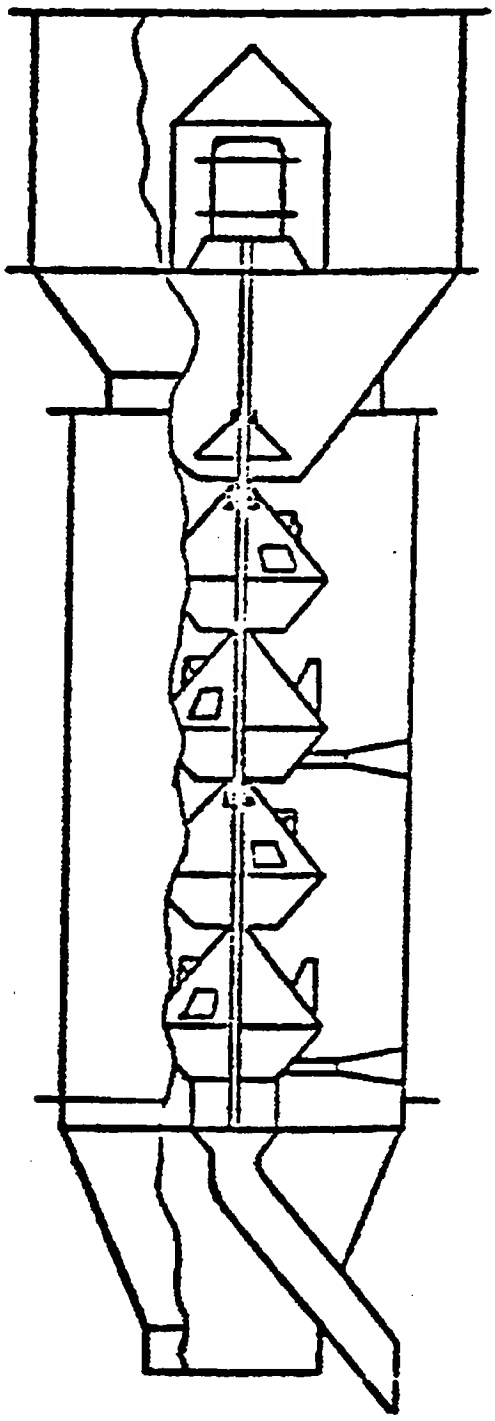
A known representative of this type is the "Koulakoff-Divider". It consists of a series of rotating and stationary double cones.



The ratio of partition can be widely varied through an increase or decrease of the number of steps. It can also obtain very large values. By means of precise tests, Pierre Gy was able to establish a systematic error of up to 20% using this equipment, caused by selective sampling of the various particle sizes.

Because of this considerable systematic error, the Koulakoff divider cannot be recommended for use in sample preparation.

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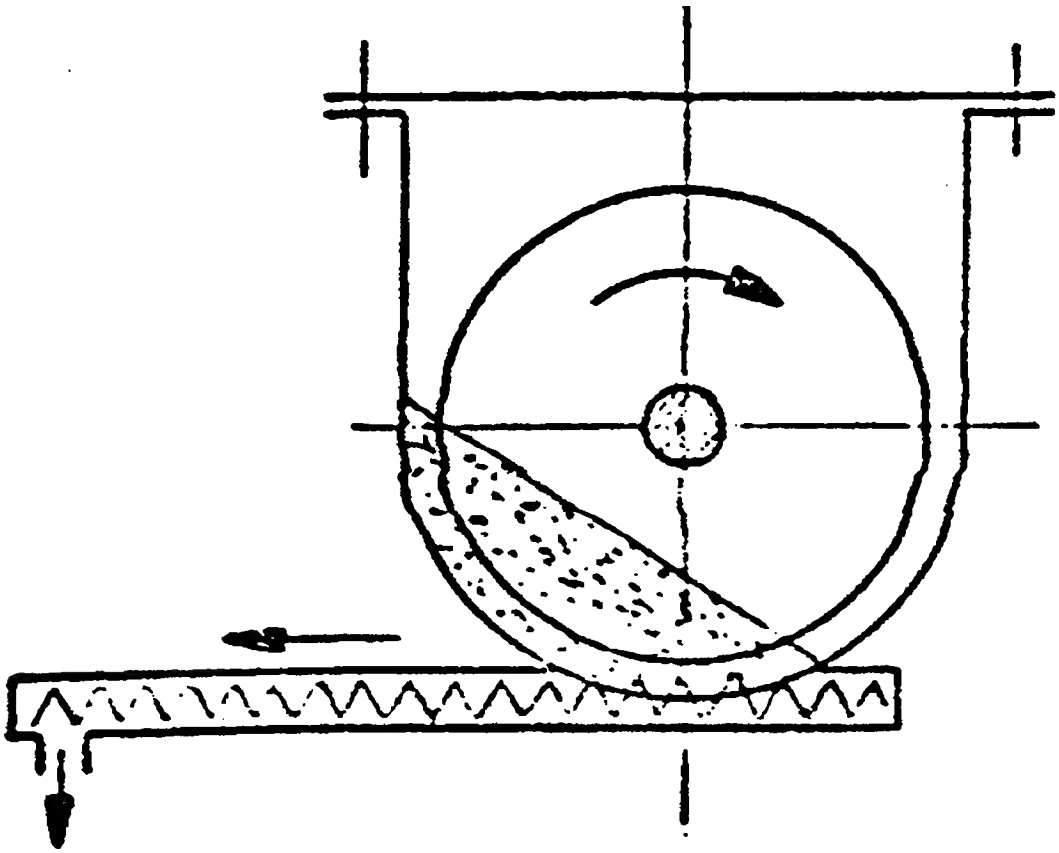
3.3.2 Sampling from screw conveyor



This type of sampling screw continuously takes part of the whole of the flow, therefore providing a longitudinal sample. As the material layer in a screw conveyor is heterogeneously distributed, this sampler will create a systematic error and therefore can not be recommended.

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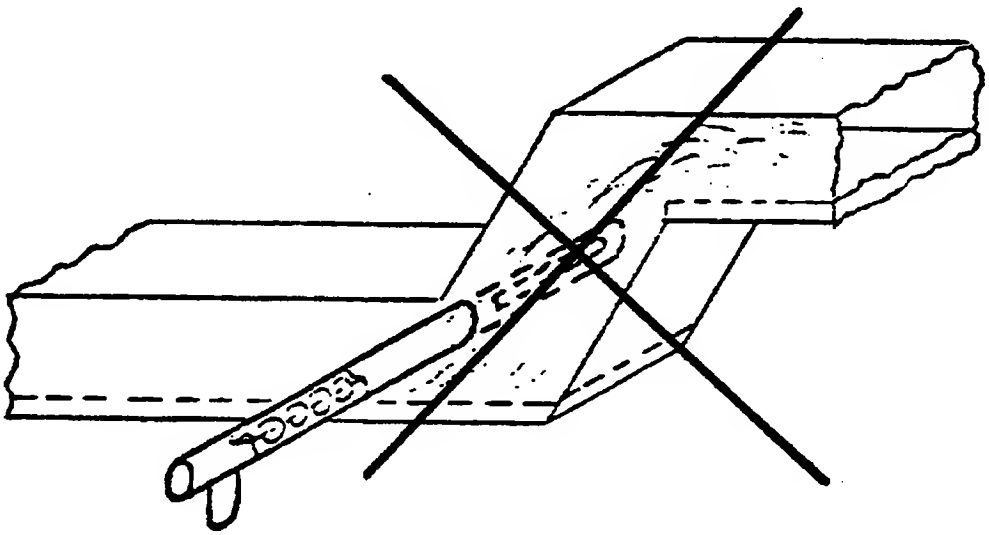
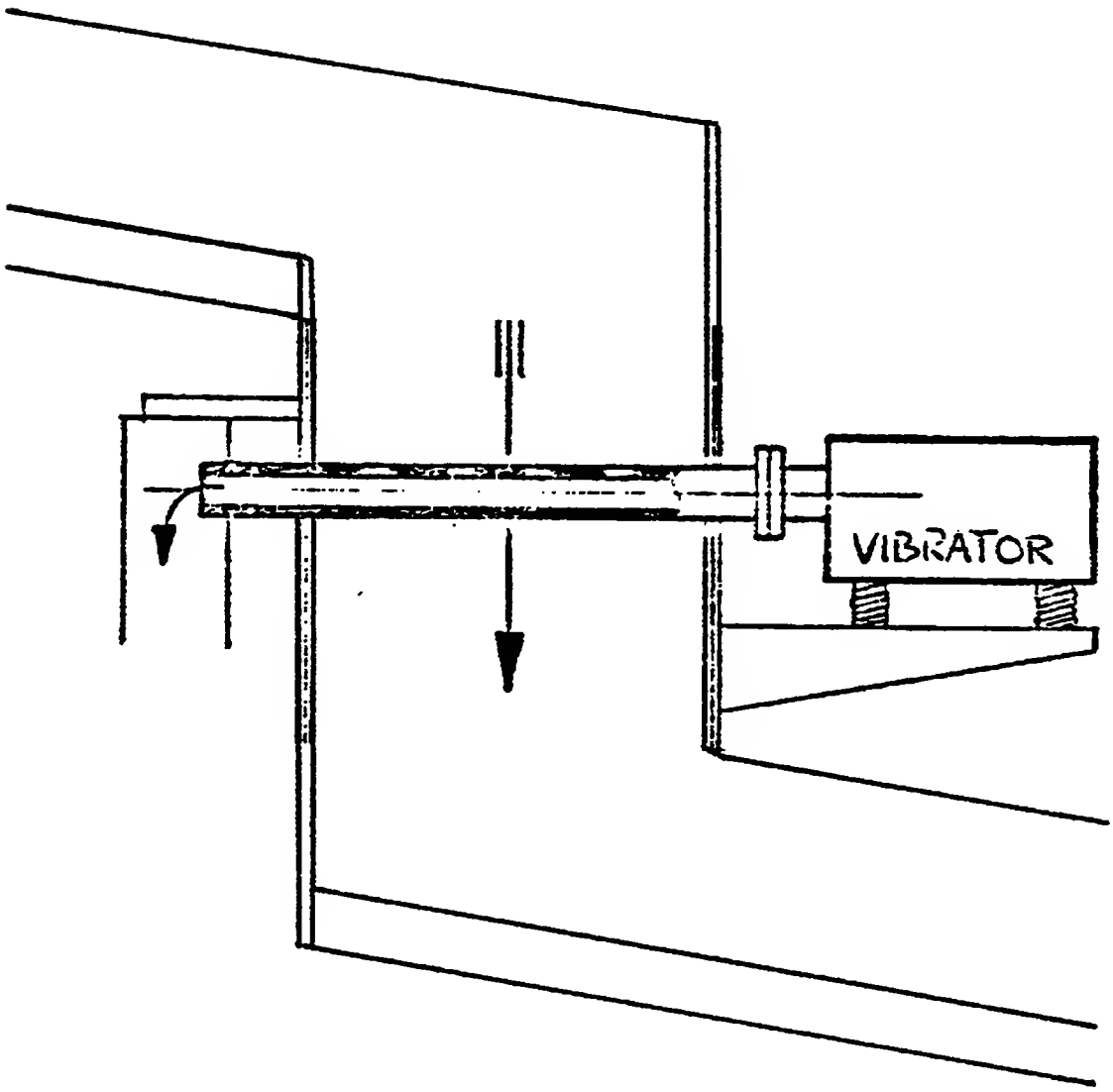
3.3.3 Sampling from a meal transfer chute





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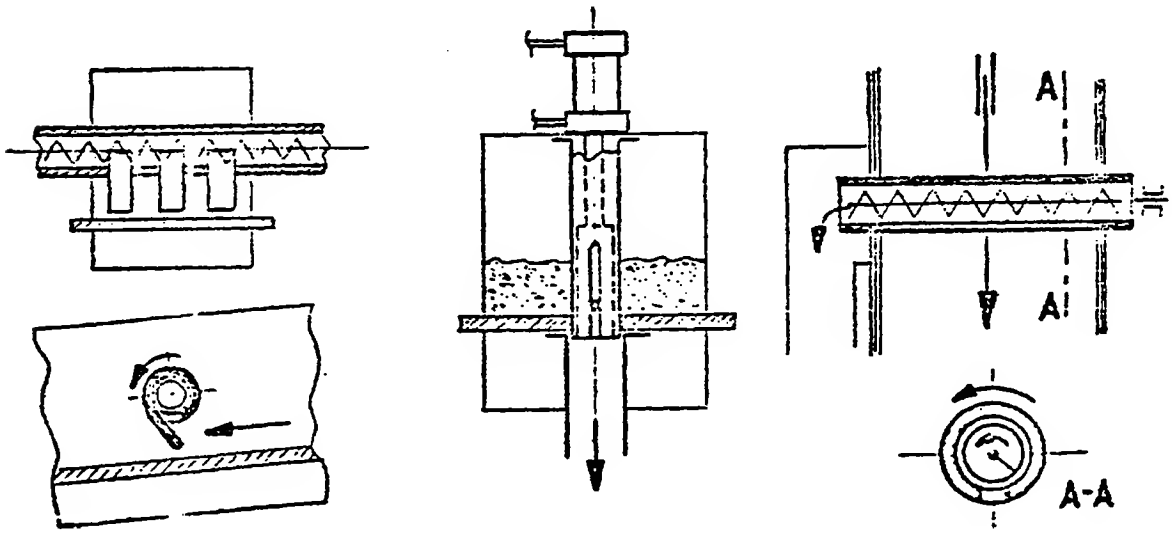
3.4 Equipment for Quasi-Point Samples

The same comments which were made for the category of section 3.3, are also valid for this class. The samples are only representative if the material is homogeneous which is mostly not the case shortly after a transfer station or a change in direction. Whether equipment of this type should be declined as a matter of principle depends mainly on the use to which it is put. In the sampling of fine meal and small grained bulk material, in which the units to be measured are not a function of the particle size, the resulting systematic errors can be ignored. In each case, however, it would be more elegant to install a sampler more suited to the material conditions. There now follows a few examples of meal samplers, which obtain quasipoint samples.

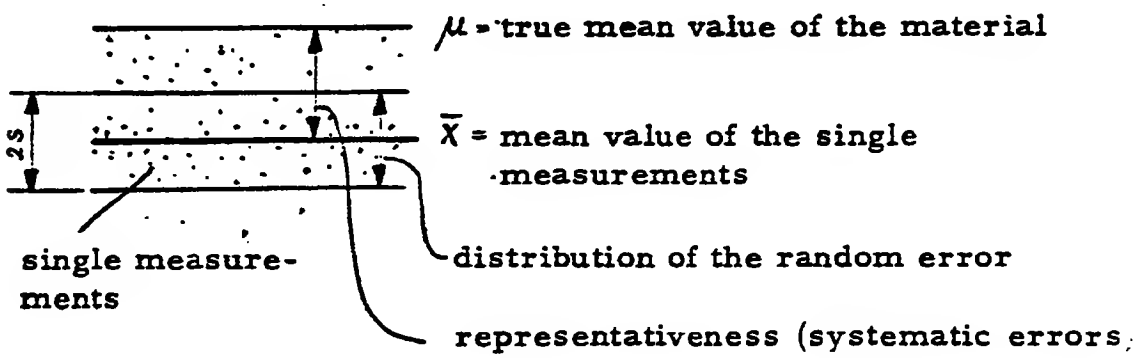


3.5 Experimental Assessment of Samplers

This chapter does not go into the process of defining the precision of the results for samplers or sampler systems, but limits itself to referring to the already mentioned book by Pierre Gy. We only wish to touch briefly on two elements of precision namely representativeness and distribution of accidental errors.

The term representativeness has already been mentioned and signifies the difference of the true mean value μ of the material and the mean value $\square$ of the single measurements. A representative sample is only obtained if the total sample error has a random variation with the mean value of 0 or in other words, if no systematic error appears. The distribution of the random error describes the reliability of the sampling. It is independent of the representativeness and can, for an example, be very small, even if $\square$ and μ are not the same. The illustration below attempts to explain these two expressions.

It is necessary to keep both terms strictly separated, and the appropriate methods must be applied to determine them. It must be further stated that here an experimental assessment of a sampling plant will cost a great deal of work and care, when done correctly.



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4. HOW TO IMPLEMENT AN ADEQUATE SAMPLING SYSTEM

For the design or selection of an appropriate material sampler which satisfies the plant-specific requirements, the following aspects must be considered:

- flow sheet, layout and detailed drawings of the area in question
  - definition of the proposed sampling point and its position within the circuit
  - space available
  - definition of potential criteria
- process-oriented criteria with regard to the potential sampling point
  - flow rate of material (t/h)
  - velocity of material (m/s)
  - pressure (in case of a closed system)
  - rate of sampling (kg/h)
  - frequency of sampling
- physical description of material samples
  - type of material
  - screen analysis
  - density
  - moisture or water content
  - stickiness, dustiness, viscosity (slurry)
  - etc.

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1. SAMPLE PREPARATION

1.1 METHOD

Much work has to be done before a sample of any material is ready for analysis. The usual sequence is as follows:

- weighing and registration of the received sample
- drying in a laboratory oven
- splitting of the sample to obtain smaller, representative samples
- coarse crushing in a laboratory jaw crusher
- re-splitting into smaller portions
- fine grinding with a swing grinding mill
- sieving with different sets of sieves (60 um for analytical purposes)
- fine re-grinding
- homogenizing of the final samples with a Turbula-mixer
- filling into bottles and labeling the samples

1.2 APPLICATION:

All types of material from raw material to cement and concrete

1.3 TIME OF PREPARATION

2- 3 samples per hour can be prepared to 60 un fineness depending on hardness and size of the original sample.

1.4 MAINTENANCE AND PERSONNEL

Installation and service of mills, crushers and mixers can be done by the in-house workshop and electricians. The work has to be done by trained personnel .

1.5 SUPPLIERS:

A. Fritsch & Co., Retsch Laborgeräte, Maschinen fabrik Herzog, Bachofen AG Maschinenfabrik for Turbula-mixer

1.6 COST OF EQUIPMENT

Approx. SFr. 140'000 to 150'000 for complete preparation laboratory

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2. GRAVIMETRY

2.1 METHOD

Gravimetric analysis or quantitative analysis by weight is the process of isolating and weighing an element or a definite compound of the element in as pure a form as possible. The element or compound is separated from a weighed portion of the substance being examined. A large proportion of the determination in gravimetric analysis is concerned with the transformation of the element or ion to be determined into a pure stable com pound which can be readily converted into a form suitable for weighing. The weight of the element or ion may be readily calculated from a knowledge of the formula of the compound and the atomic weights of the constituent elements.

The separation of the element or of the compound containing it may be effected in a number of ways, such as: precipitation methods, volatilisation or evolution methods, electro-analytical methods and miscellaneous physical methods.

One of the most important methods in gravimetry is precipitation, i.e. the constituent being determined is precipitated from solution in a form which is almost insoluble in water so that no appreciable loss occurs when the precipitate is separated by filtration and weighed.

2.2 APPLICATION

Determination of all elements in all kinds of material. The alkalis sodium and potassium are very difficult to determine by gravimetry.

2.3 TIME OF ANALYSIS

Very time consuming! At least five hours for the determination of the four main elements - or 4 samples per day

2.4 MAINTENANCE AND PERSONNEL

Minimum of maintenance required for balances and furnaces. Routine work should be executed by skilled and qualified personnel.

2.5 EQUIPMENT:

No special instruments and apparatus are needed apart from the normal equipment of an analytical laboratory such as glassware, furnaces, balances, etc.

3. TITRATION OF CARBONATES

3.1 METHOD

A sample of a raw material (e.g. limestone) or raw meal is dissolved in a known excess of hydro chloric acid:

$CaCO_3 + 2HCl \rightarrow CaI_2 + H_2O + CO_2$

The excess of acid is then determined by a so called back titration with sodium hydroxide solution

$HCl + NaOH \rightarrow NaCl + H_2O$

The end point of the back titration can be found visually by means of an indicator (dye stuff) which changes its colour or by measuring an electrical parameter, i.e. the potential difference between two electrodes dipping into the solution. The result is calculated with a programmable pocket or desk calculator.

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3.2 APPLICATION

All types of raw material and raw meal; including all carbonates which are expressed as $CaCO_3$ by calculation.

3.3 TIME OF ANALYSIS

10 minutes per sample

3.4 MAINTENANCE AND PERSONNEL

Instruments have to be installed and introduced to the operator by the supplier. Routine work can be performed by trained personnel.

3.5 SUPPLIERS

e.g. Metrohm, Mettler, Radiometer, etc.

3.6 COST OF EQUIPMENT

approx. SFr. 22'000 for semiautomatic titration apparatus.

4. COMPLEXOMETRIC TITRATION

4.1 METHOD

The complexometric determination of many elements is based on a titration with a solution of EDTA (Ethylene Diamine Tetra Acetic Acid) using many special indicators. The end point of the titra tion is found either visually or with the aid of a spectrophotometer which is more sensitive to colour changes of many of the indicators. The results are calculated from the consumption of the EDTA solution.

4.2 APPLICATION

Ca, Fe, Al, and Mg in cement, clinker, raw meal and raw material such as limestone, clay and sand. Titanium and manganese may be determined also by complexometric titration whereas silica has to be analysed by gravimetry or colorimetry.

4.3 TIME OF ANALYSIS

3 - 4 hours per sample including silica, or 6 - 7 samples per day

4.4 MAINTENANCE AND PERSONNEL

Installation and implementation by the supplier or a chemist. Routine work requires laboratory assistant.

4.5 SUPPLIERS

of spectrophotometer and semiautomatic titrators potentiographs: Metrohm and Mettler

4.6 COST OF EQUIPMENT

ca: SFr. 20'000.

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5. SPECIFIC SURFACE ACCORDING TO BLAINE

5.1 METHOD

The Blaine air permeability apparatus is used to draw a definite quantity of air through a prepared bed of cement of definite porosity. Number and size of the pores in the prepared bed (cement layer of 2.7 9 and 1.5 cm thickness) is a function of the size of the particles and determines the rate of flow through the bed. The cement is filled into a cylindrical cell and pressed by a plunger to form the bed. The cell is attached to the U-tube of a manometer, one arm of which is now slowly evacuated until the liquid reaches the top mark on the glass tube. After opening a valve, the liquid flows back again and passing two other marks, a definite volume of air is pressed through the bed of cement.

The time which is required for the meniscus of the liquid to pass from one mark to the other is measured. It is necessary to use a complicated formula for the calculation of the specific surface, there fore it is advantageous to split this formula into three factors, one of them depending only on the density of the cement, the second one depending on the temperature and the third one being the square root of the stopped time. The first two factors may be taken from a diagram which has to be established once for every Blaine apparatus.

5.2 APPLICATION

Cement only! Although the method has been used for the determination of the fineness of various other materials, it should be clear that in general relative rather than absolute values are obtained.

5.3 TIME OF ANALYSIS

Double determination for one material takes approx. 20 minutes if the density of the cement is known. Otherwise, additional 45 min. are needed for the determination of the density.

5.4 MAINTENANCE AND PERSONNEL

Trained personnel.

5.5 SUPPLIERS

e.g. Schiltknecht, Zürich; Tonindustrie for semi automatic instrument.

5.6 COST OF EQUIPMENT

SFr. 800 to 6'000 for semiautomatic instrument

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6. DETERMINATION OF SULPHUR AND CARBON DIOXIDE

6.1 METHOD

The solid sample is mixed with metal powder (copper, iron or tungsten) as an accelerator and burnt in a high frequency or resistance induction furnace at cat 1500°C in a stream of pure oxygen. The gases produced in the chemical reaction during combustion are sulphur dioxide, carbon dioxide and water steam. With different analysers it is possible to determine either SO_2 or CO_2 or even both compounds. The simplest and cheapest method uses a titration with potassium iodate executed in an automatic burette which is connected with the induction furnace. Another possibility is the use of an infrared analyser instead of a burette. The IR analyser determines the sulphur dioxide by its absorption of infrared light and displays the result as percent SO_3. - Carbon dioxide can be determined with another model which gives the result as total carbon. Instruments for the determination of both SO_3 and carbon by infrared light are also on the market.

The Leco Corporation has another type of instrument for carbon analysis which does not apply infrared absorption but a heat conductivity cell.

6.2 APPLICATION

Total sulphur and carbon in all types of solid material, with special instruments or by means of accessories and chemical reagents also in fuels such as coal and oil.

6.3 TIME OF ANALYSIS

6 - 12 minutes per sample with sulphur titrator depending on material. 3 - 4 minutes per sample with infrared or heat conductivity analyser

6.4 MAINTENANCE AND PERSONNEL

Instruments have to be installed by supplier or specialist. Routine work can be done by trained personnel. Service by the supplier is very important for infrared and heat conductivity analysers.

6.5 SUPPLIERS

LECO Corporation (USA) for: Titrator DB 64 and IR 232 for SO_3, IR 212 and WR 12 for carbon, CS 244 for sulphur and carbon

Leybold-Heraeus (Germany): Series 302 for sulphur and carbon

6.6 COST OF EQUIPMENT

Titrator ca. SFr. 26'000
Infrared analyser ca. SFr. 50'000 to 70'000
Infrared analyser for C + S ca. SFr. 90'000

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6.7 ATOMIC ABSORPTION SPECTROMETRY

6.8 METHOD

The sample has to be dissolved in acid, generally hydrochloric acid. This solution is then sprayed into a very hot flame (ca. 2000.C, for some elements 3000.C) where the solvent evaporates and the chemical compounds dissociate into their atomic components. Most atoms remain in the ground state, a fact which is important for atomic absorption.

The light of a so-called hollow cathode lamp (HCL), the cathode of which consists of the element to be determined is led through the flame with the cloud of mostly neutral atoms. The atoms are able to absorb this HCL-light so that the intensity of the light beam is reduced. The absorption is a measure of the number of atoms in the flame which is proportional to the concentration of the element in the solution or at least the sample.

6.9 APPLICATION

All types of material and all elements of interest in cement chemistry except sulphur, fluorine, chlorine and phosphorus.

6.10 TIME OF ANALYSIS

approx. 4 hours for one analysis including Si, Al, Fe, Ca, Mg, K, Na or 6 analyses per day.

6.11 MAINTENANCE AND PERSONNEL

The method has to be introduced by a specialist or chemist. Routine work can be executed by trained personnel.

Service by supplier should be guaranteed.

6.12 SUPPLIERS

Perkin Elmer, Beckman, Varian, Hilger Watts, Jarell Ash, Instrumentation Laboratory, Pye Uni cam (Philips)

6.13 COST OF EQUIPMENT

SFr. 50'000 to 120'000 depending on spectrometer single or double beam, etc.)

7. ATOMIC EMISSION SPECTROMETRY (BY INDUCTIVELY COUPLED PLASMA)

7.1 METHOD

The sample has to be dissolved in acid very often after a fusion procedure. This solution is then sprayed into a very hot argon plasma, a physical flame of more than 6000.C where the solvent evaporates and the chemical compounds dissociate into their atomic components. The atoms are excited by the high temperature of the plasma and are then able to emit light of certain wave lengths which can be measured and which is proportional to the concentration of the elements to be determined.

7.2 APPLICATION

All types of material and all elements of interest in cement chemistry with the exception of the halogens.

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7.3 TIME OF ANALYSIS

approx. 1 hour for determination of 30 to 40 elements and including preparation of the solution.

7.4 MAINTENANCE AND PERSONNEL

The method has to be introduced by a specialist and chemist. Routine work can be executed by very well trained personnel.

Very important it that good and prompt service is guaranteed by the supplier.

7.5 SUPPLIERS

ARL, Jarell-Ash, Instrumentation Laboratory, Jobin-Yvon, Kontron, Philips.

7.6 COST OF EQUIPMENT

SFr. 180'000 to more than 300'000 depending on the instrument: sequential or multi-channel spectrometer.

8. X-RAY FLUORESCENCE (XRF)

8.1 ANALYSIS:

Qualitative and quantitative determination of chemical elements with atomic number higher than 9.

8.2 METHOD

Wave lengths dispersive method or non-dispersive technique (energy dispersive)

8.3 PRINCIPLE

The sample (in form of pressed powder tablet or fused bead) is irradiated by primary X-radiation from an X-ray tube. Atoms and ions of the various elements in the sample are thereby excited and emit their characteristic radiation.

- Wave length dispersive method
  The fluorescence radiation from the excited sample is dispersed on special analysing crystals according to Bragg's law (compare with dispersion of sunlight by a prism into the bands of colour of the visible spectrum). The resolved characteristic radiation of the different elements is analysed individually and measured by appropriate detectors. The intensity of the characteristic radiation is thereby a direct measure of the chemical concentration of the corresponding element.
- Energy dispersive technique
  The fluorescence radiation is measured using a semi-conductor together with an analyser which has an electrical response proportional to the quantum energy of the radiation to be measured.

8.4 APPLICATION

Determination of the chemical composition of raw material, raw meal, clinker and cement. Plant laboratories are mostly equipped with so-called wave length dispersive multi-channel units and make use predominantly of pressed powder tablets; central laboratories often prefer the installation of sequential spectrometers to increase flexibility of analysis.

8.5 TIME OF ANALYSIS

According to the application used: 3 to 20 minutes for 8 elements excluding preparation.

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8.6 LABOUR

1 well trained man

8.7 SUPPLIERS

Philips, Siemens, ARL, Rigaku (Ortec, EDAX)

8.8 COST OF EQUIPMENT

SFr. 250'000 - 400 '000

9. X-RAY DIFFRACTION (XRD)

9.1 ANALYSIS

Qualitative and quantitative determination of crystalline solids

9.2 METHOD

A monochromatic X-ray beam is directed upon a fine powdered sample. By varying the angle between sample and incoming beam, the X-rays are reflected from the various crystallographic planes of the minerals present. The reflected X-rays contain the following information:

Angle: identification of mineral

Intensity: concentration of individual mineral

9.3 APPLICATION

Raw meal: content of quartz, clays, dolomite, etc.

Clinker: content of free lime, periclase, sulphates, aluminate, ferrite and modifications of C_3S, C_2S, C_3A, etc.

9.4 TIME OF ANALYSIS

Quantitative 10 minutes (computerised)
Qualitative 1 hour

9.5 STAFF

Skilled laboratory assistant, mineralogist, chemist

9.6 SUPPLIERS:

Philips (Netherlands), Siemens (W. Germany), Seiffert and Co. (W. Germany), CGR (France), Rigaku (USA, Japan)

9.7 COST OF EQUIPMENT

SFr. 80'000 to 200'000

10. LIGHT MICROSCOPY

10.1 ANALYSIS

Determination of microstructure and mineralogical composition

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10.2 METHOD

Samples are examined on:

- polished sections using reflected light
- thin sections and loose powders using transmitted light

Qualitative information:

- phase association
- texture
- structure

Quantitative information:

- phase composition
- crystallite / grain / granule size

10.3 APPLICATION

Determination of actual clinker composition, e.g. percentage of elite, belite, ferrite, aluminate, free lime, periclase and sulphates

Determination of clinker microstructure to ascertain conditions present during production.

Characterisation of raw mix components (e.g. limestone) Investigation of refractory and concrete damages.

10.4 TIME OF PREPARATION

Preparation: approx. 3 hours:

Interpretation: 1 - 3 hours

10.5 STAFF

Skilled laboratory assistant, mineralogist, chemist

10.6 SUPPLIERS

Wild & Leitz (Switzerland), etc.

10.7 COST OF EQUIPMENT

SFr. 20'000 to 60'000

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11. THERMAL SHOCK TESTING OF BASIC REFRACTORIES

11.1 METHOD

Test prisms (35 x 35 x 200 mm) are cut from the supplied brick(s) using a diamond impregnated saw, such as to give a full hot to cold face section. For dolomite products, the cutting should be performed dry; with magnesite bricks wet cutting is possible but requires immediate drying afterwards.

For shock testing, the prepared prisms are heated at 1350.C in a laboratory furnace, for a period of up to 15 minutes, including heating up time. They are then placed vertically (i.e. end-on) upon a water-cooled copper plate, on which they are allowed to cool for 10 minutes. After this time, the now cooled prisms are checked for spelling and crack development. They are then returned to the furnace for more heating up and subsequent cooling cycles until 20 such treatments have been performed, assuming of course that previous destruction has not taken place.

11.2 APPLICATION

Testing of new refractory types and quality control on already delivered ones, in cases where doubt exists as to the quality supplied.

11.3 TIME OF ANALYSIS

For the cutting of the prisms about 1 hour is required with drying taking place overnight. The actual thermal testing requires about 9 hours, the actual number of samples tested depending principally on the capacity of the 1350°C furnace.

11.4 MAINTENANCE AND PERSONNEL

No special requirements with regard to type of equipment and its maintenance. The work can be performed by semi-skilled personnel after a short training period.

11.5 COST OF EQUIPMENT

approx. SFr 10'000 for diamond saw, high temperature furnace and cooling plate.

12. THERMAL ANALYSIS - DIFFERENTIAL THERMAL ANALYSIS (DTA)

12.1 METHOD

Qualitative and semi quantitative determination of caloric changes due to exo- and endothermic reactions in materials subjected to heating; max. temperature: 1500°C.

12.2 PRINCIPLE

Approx. 100 mg of sample are weighed into a Pt-crucible and placed into the DTA furnace. A second crucible with Al_2O_3 is used as reference. Both, sample and reference are heated in the same furnace according to a selected temperature gradient. The temperature of sample and reference is measured continuously and the difference is recorded.

To relate temperature difference with energy it is necessary to perform a calibration in the temperature range of interest prior to the measurement.

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12.3 APPLICATION

Identification of clay minerals in cement raw materials, investigations on de-carbonisation of carbonates and formation of clinker minerals. Behaviour of circulating elements under different atmospheric conditions.

12.4 TIME OF ANALYSIS

Depending on type of application: 30 minutes to 3 hours.

12.5 PERSONNEL

Skilled laboratory assistant

12.6 SUPPLIERS

Perkin Elmer, Stanton Redcroft, Dupont, Netsch

12.7 COST OF EQUIPMENT

SFr. 70'000 - 130'000

Purchase price depends on whether DTA is combined with DCS and/or TGA.

13. THERMAL ANALYSIS - THERMO-GRAVIMETRIC ANALYSIS (TGA)

13.1 METHOD

Quantitative determination of weight losses or weight gains in materials subjected to heating; max. temperature 1000°C.

13.2 PRINCIPLE

Up to 1g of sample (generally approx. 50 mg) are weighed into a Pt-crucible which hangs on filament of the TGA-balance. The sample is heated in a platinum resistance furnace at a selected heat rate. The weight of the sample is determined and recorded continuously. Weight changes are given in absolute values or in percent of initial weight

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13.3 APPLICATION

Investigation of decomposition, dehydration and oxidation accompanied by weight changes, e.g. determination of degree of hydration and carbonisation in hydrated clinker or cement (open clinker storage, coating in cement mill or silos); determination of crystal water, e.g. of gypsum, hemihydrate etc.; determination of fillers (components) in blended cements; oxidation of sulphides and organic compounds.

13.4 TIME OF ANALYSIS

Depending on type of application: 30 minutes to 3 hours

13.5 PERSONNEL

Skilled laboratory assistant

13.6 SUPPLIERS

Mettler, Perkin Elmer, Dupont, Stanton Redcroft

13.7 COST OF EQUIPMENT

SFr. 60'000 - 80'000 Purchase price may depend on whether TGA is combined with DSC and/or DTA.

14. THERMAL ANALYSIS - DIFFERENTIAL SCANNING CALORIMETRY (DSC)

14.1 METHOD

Qualitative and quantitative determination of caloric changes in materials subjected to iso thermal or non-isothermal heating; max. temperature approx. 600°C.

14.2 PRINCIPLE

Approx. 10 - 100 mg of sample are weighed into a Al or Pt crucible and placed into the sample furnace of the DSC. An empty crucible is used as reference and placed into the reference furnace. Both, sample and reference are subjected to the same temperature program and equal atmospheric conditions. Differences in temperature between sample and reference - due to caloric changes - are compensated by supplying the required amount of electrical energy. The heat of reaction is determined by integration of the energy flow. The kinetics of the reaction may also be determined.

14.3 APPLICATION

Transformation of phases: melting, crystallisation, evaporation, volatilisation, changes in modifications. Chemical reactions: decomposition, oxidation, dehydration, hydration, e.g. determination and optimization of gypsum phases in cement, hydra tion of cement, determination of sulfides in raw materials.

14.4 TIME OF ANALYSIS

Depending on type of application: 30 minutes to 1 day.

14.5 PERSONNEL

Skilled laboratory assistant

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14.6 SUPPLIERS

Perkin Elmer, Mettler, Dupont, Stanton, Redcroft

14.7 COST OF EQUIPMENT

SFr. 60'000 - 150'000

Purchase price depends on whether DSC is combined with DTA and/or TGA

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Quality Assurance

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1. SCOPE OF TESTING

- Production control
- Quality Control (due to standard specifications)
- Material technological investigations
  - project planning
  - application
  - research & development

METHODS OF CHEMICAL CONTROL

Method	Criteria			
	Investment	Information	Times of analysis	Personnel
Titration	Very little	Σ Of carbonates	Minutes	Few, trained
Gravimetry volumetry	Little	Main elements LS, SR, AR; BOGUE	Many hours	Several, trained
Complexo metry colorimetry	~ $10'000	CaO, Al_2O_3, Fe_2O_3, MgO	Approx. 2 - 3 Hrs.	Several, trained
AAS	~ $30'000	All elements except S, Cl, F, P	Approx. 2 - 3 Hrs.	Several + specialist
XRF	~$250' 000	All elements + complete automation (off or online)	Minutes	Few + specialist

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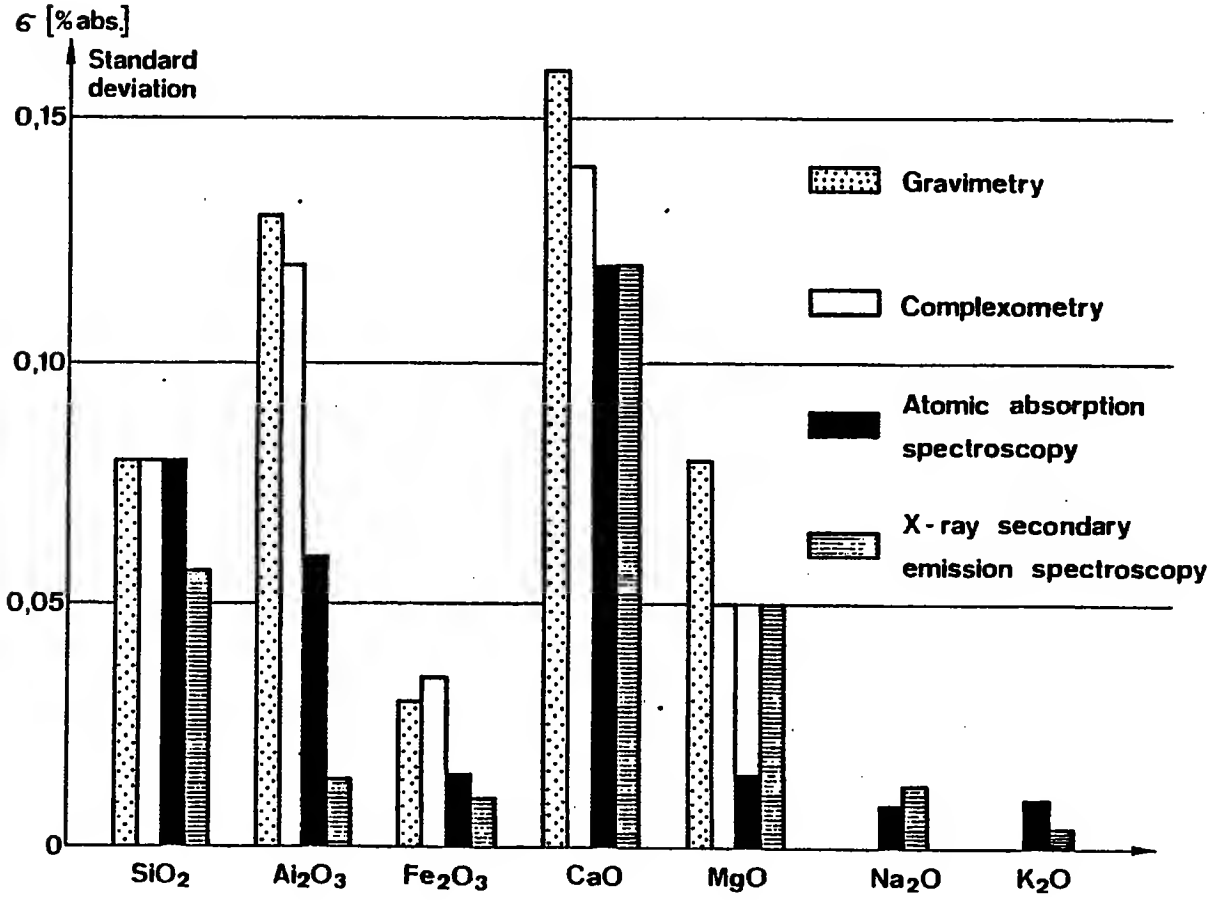
Fig. 1 **Standard deviations for clinker analyses by different analytical techniques.**



2. TYPES OF XRF-SPECTROMETERS

- Wavelength dispersive (WLD)
  - Sequential Mode
  - Simultaneous ('Mull) Channel') mode
- Energy dispersive (ED)
- Low cost (200 W) WLD Systems
- Bench-top XRF analysers
- Continuous ('on-line') XRF analysers

3. X-RAY FLUORESCENCE SPECTROMETERS LOW COST EQUIPMENT (200 W)

- ADVANTAGES:
  - Moderate purchase costs
  - Lower operational cost
  - Simple operation and maintenance

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- DISADVANTAGES:
  - Less sensitive
  - Increased! measuring time
  - Frequent re-calibration
  - No trace element analyses

4. ANALYTICAL TRENDS IN XRF

- Trace element analyses
- Chemical analysis of liquids
- Analysis of carbon
- Sulphide/sulphate analysis

5. ADDITIONAL METHODS

- Sulphur analyser
- Carbon analyser
- Coal analyser
  - calorimeter
  - proximate analysis
- Free lime
  - Conductometric
  - X-ray diffraction

6. ADDITIONAL ANALYTICAL EQUIPMENT

- delivery control of waste
- emission control
- XRF (additional channels or sequential mode)
  or
  AAS → trace elements in liquids/solids
- Ion chromatography or ion-sensitive electrodes
  - halogens (chlorine, fluorine, bromine)
- flash point determination
- viscometer
- gas chromatography
  - organic compounds (PCB's etc.) bomb calorimeter (automatic)

7. ADDITIONAL ANALYTICAL EQUIPMENT

- gas chromatography with mass spectrometer
  - emission control on stack gas (organic compounds, volatile toxic elements etc.)

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Chapter 12

Automation in Quality Control

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Automation in Quality Control

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**First Chemist's Colloquium of
North American "Holderbank" Group Plants
January 17-19, 1990 in St. Louis, MO**

Automation in Quality Control

**by Dr. P. Bürki,
"Holderbank" Management & Consulting Ltd.
Technical Centre**

Summary:

Automated quality control systems find increasing application in the cement industry. The main objectives are to intensify the control in order to achieve more uniform materials properties and to reduce risks associated with inadequate quality and to reduce shift personnel in quality control. Automated systems are widely applied for raw mix blending and clinker control, to a lesser extent so far on cement grinding, and little for physical cement testing.

The paper reviews the concept and main features of such systems on the hand of examples mainly from 'Holderbank' group plants, and briefly discusses the potential and limits of neutron activation analysis.

Report No. MA 89/3702/E

1. INTRODUCTION

There is definitely a trend to increase the automation in quality control procedures in the cement industry, whereby the degree of automation can vary from individual automated testing equipment to complex integrated on-line systems.

The objectives of automation, and possible drawbacks are summarised in the following figure.

Fig. 1 BENEFITS AND DRAWBACKS OF AUTOMATED CONTROL SYSTEMS

BENEFITS:

Increased product quality and reduced operation costs through improved uniformity:

- high sampling frequency
- more sampling points
- round-the-clock control
- optional on-line link to process computer
- high standard of sample preparation and analysis
- elimination of operator related errors

Liberation of personnel from routine work for more demanding duties.

Reduction of personnel.

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DRAWBACKS:

High investment costs

Susceptible to (minor) technical faults

Requirement of back-up methods

Highly qualified personnel required for maintenance

An attempt to classify various concepts of automation is made in Figure 2.

FIG. 2 CONCEPTS OF AUTOMATED CONTROL SYSTEMS

- Centralised:
  - sample transport to central test laboratory
- Decentralised:
  - sample transport to testing equipment installed close to production unit
  - continuous analysis on full or partial, by-passed materials stream

The present paper does not aim at giving a systematic, well balanced review of current automation systems, but to highlight some selected aspects of automation, mainly taken from examples of such systems installed in Holderbank Group plants.

2. CENTRALISED CONTROL SYSTEMS

2.1 The Automated Control System at Obourg (Lit. 1)

2.1.1 Objective

Obourg is one of the few remaining wet plants in western Europe. The tradition of using low-cost ash rich coal has made it possible to produce at costs competitive to dry plants. The lay-out is shown in Figure 3.

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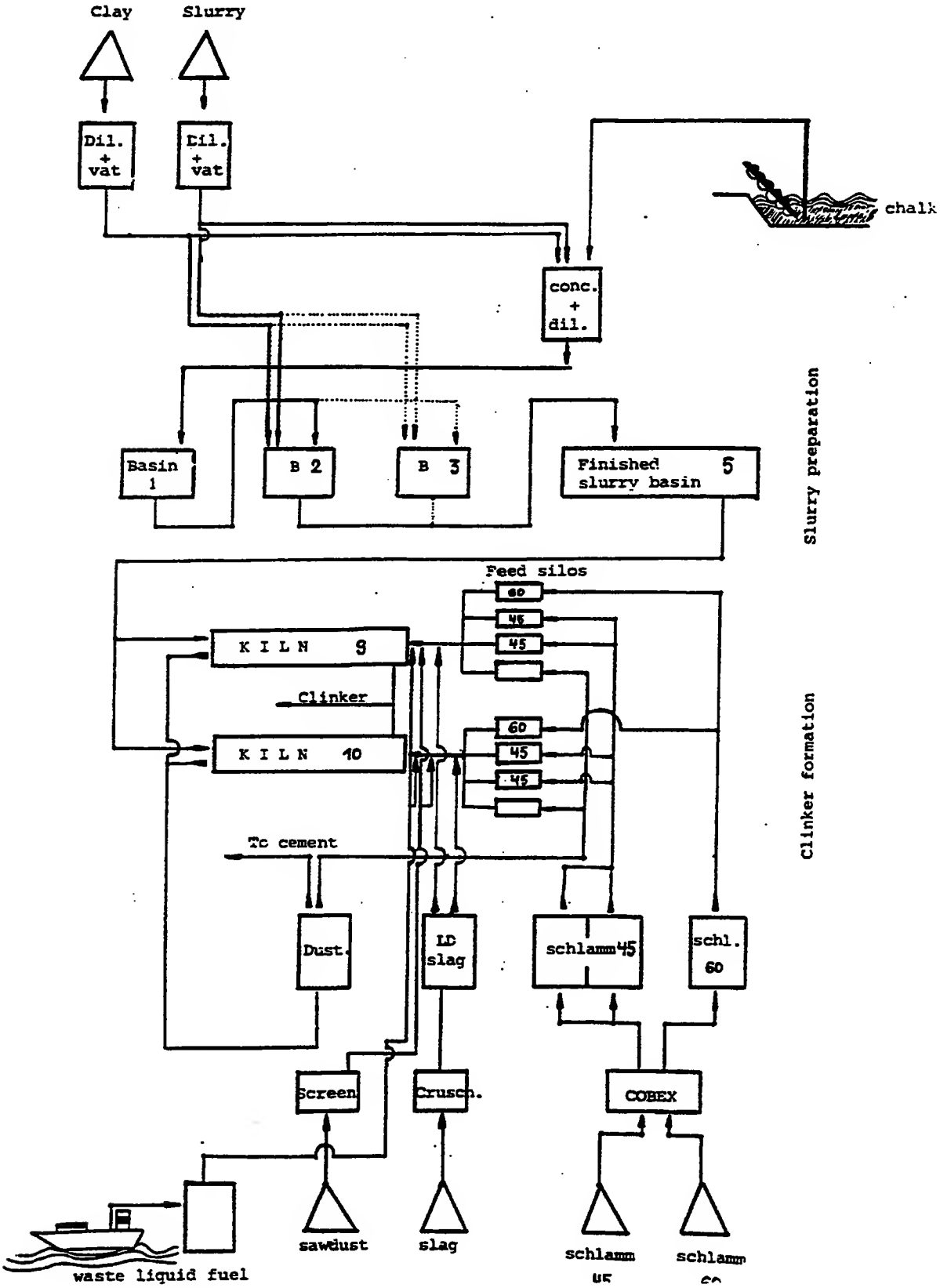
Fig. 3 Preparation of Clinker at Obourg.



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The reasons for investing into a complex automated control system were:

- High variability in chemical composition; occasional quality problems due to excessive C_3A content.
- Intention to increase the use of ash rich coal, aiming at an ash content in clinker of 20% (including non-combustible components injected into the sinter zone), which increases the average C_3A content.
  - Clinker targets:
    LS : 98 ±1
    C_3A : 12 ±1 %
    C_4AF : 11.5 ±1 %

Such an extreme clinker composition does not tolerate significant fluctuations!

- Intention to change the slurry preparation from a batch to a continuous mode, to eliminate frequent bottlenecks in slurry production.

The realisation of these objectives necessitates very close blending control for both slurry and clinker production. After a detailed evaluation, it was decided:

- to automate slurry and clinker control up to the stage of calculation of feed corrections (excluding feeder adjustments)
- to adopt a centralised concept
- to realise the investment as a turnkey project.

A corresponding contract was made with Polysius to supply their POLAB-system, whereby detailed guarantee conditions for system performance and availability were agreed upon.

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2.1.2 Aspects of hardware

Fig. 4 Block Diagram of the Polab system.

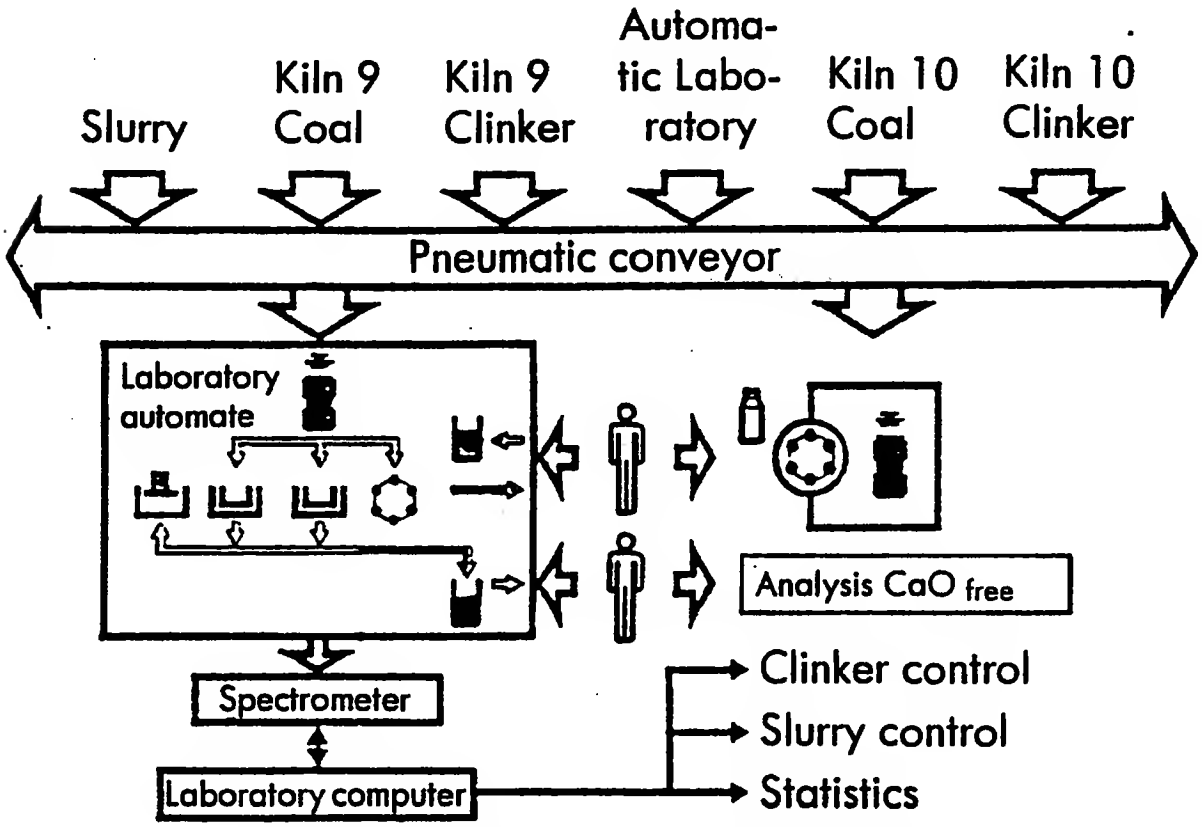

Figure 4 shows the elements of the POLAB system. Features of particular interest are:

- Slurry sampling (Figure 5)
  Emphasis was given to obtaining a representative sample of the slurry, which still contains an appreciable quantity of large grains; hence the complex preparation plant.
- Clinker sampling
  The sampling from the cooler (second screen plate) reduces the dead time of the control loop.
- Laboratory automate
  Sample preparation by grinding and pelletising was selected, one reason being the scarce experience available at that time with automated fused lead preparation.
- XRF-spectrometer
  The eight channel simultaneous spectrometer is exclusively used for the POLAB system.
- Free lime analyser
  The POLAB is complemented with a free lime analyser DIA 2000 (ethylene glycol extraction method), which was formerly installed at the kilns.

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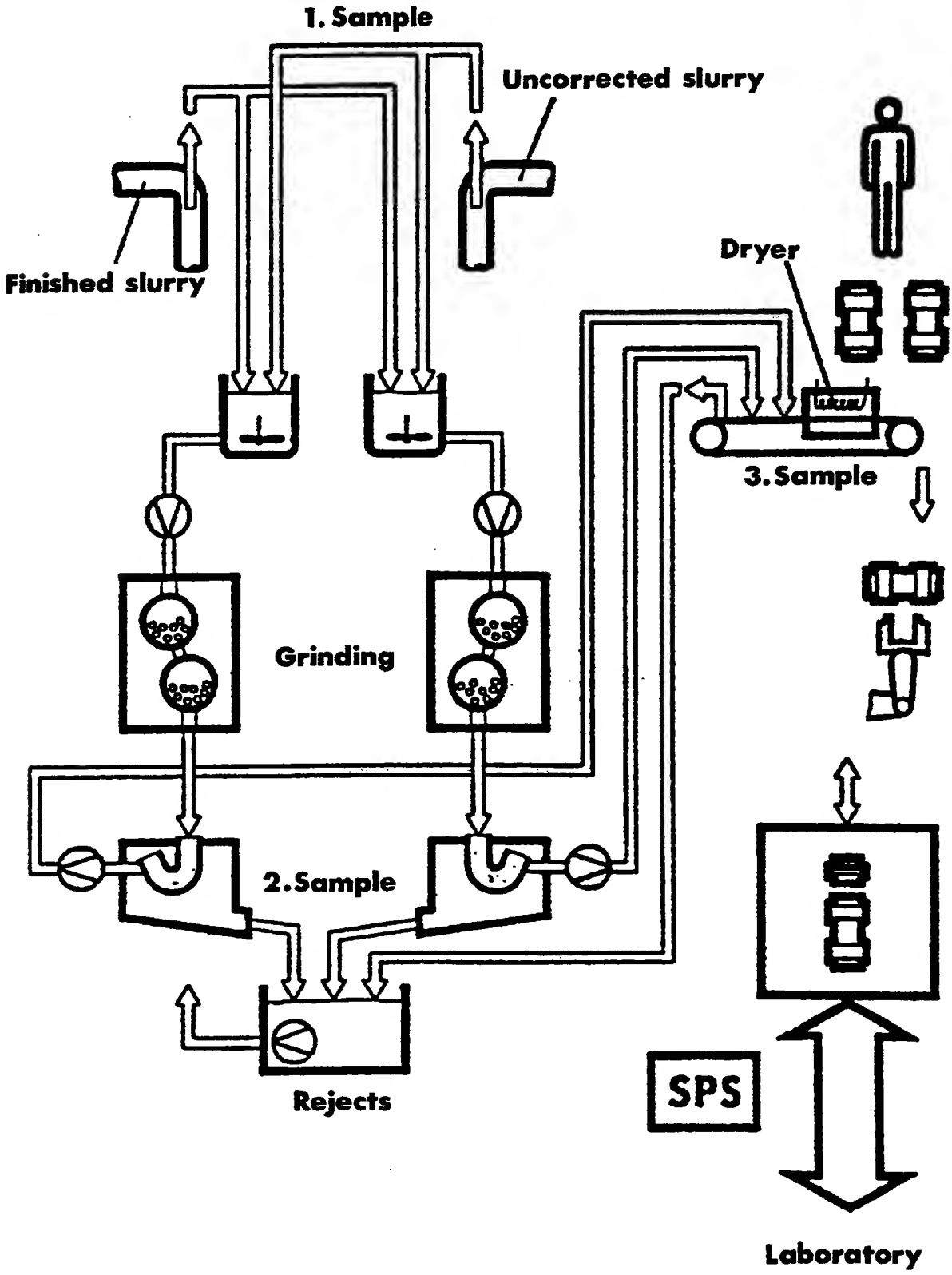
Fig. 5 Slurry sampling and preparation plant.



2.1.3 Aspects of software

The POLAB blending control software is responsible for calculating the components feed rate both for slurry preparation and for clinker production.

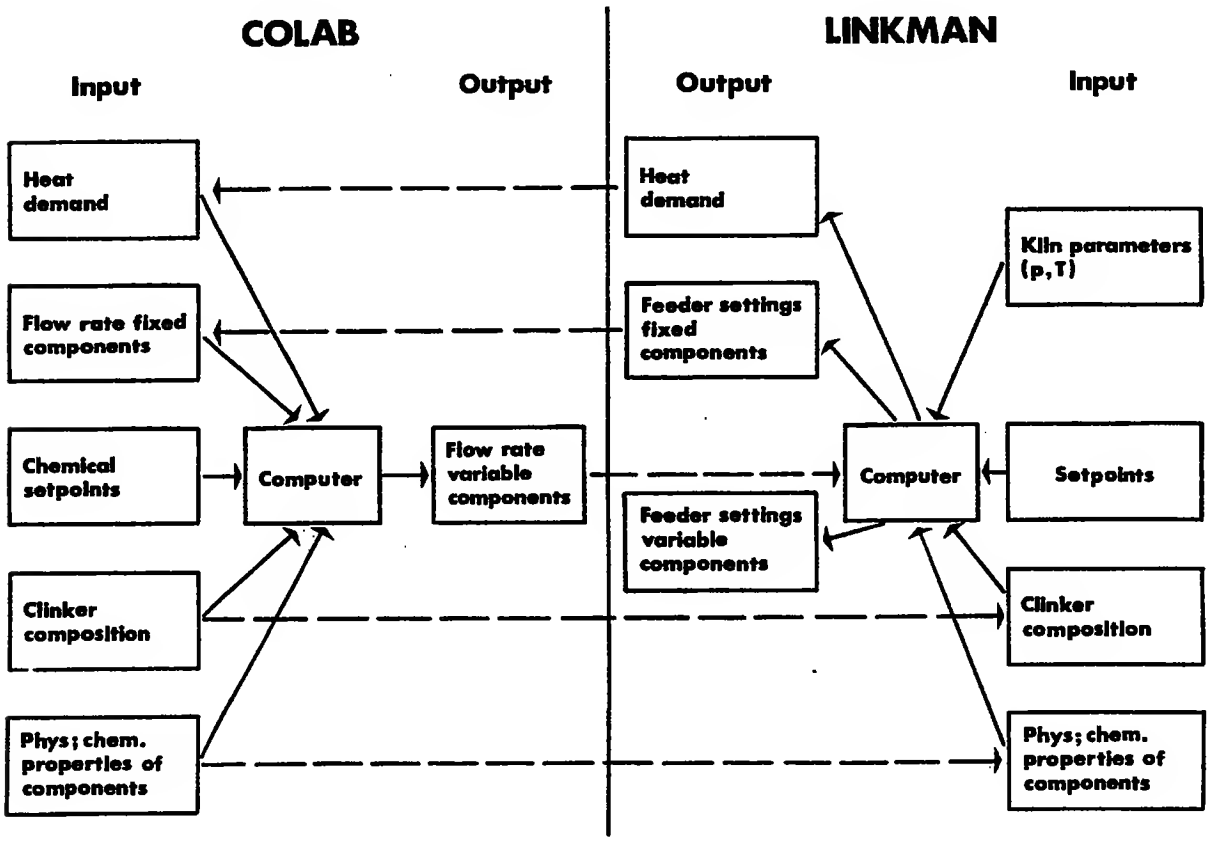
One aspect of the clinker 'blending' control is worth commenting on:

A number of the 4 or more components injected at the kiln hood into the sinter zone affect both the ash and the heat input. The blending control has therefore not only to meet the clinker composition targets, but also the target of the kiln heat requirement. The latter is introduced and continuously adapted by the kiln operator through a terminal in the control room.

2.1.4 Link to kiln automation

Recently, LINKMAN kiln automation systems were implemented for both kilns. The essential tasks of the system are to determine the kiln heat requirement and to control and automatically adjust the feed rates of all materials fed to the kiln. Because the POLAB and the LINKMAN have obviously to fulfill complementary tasks, the two respective suppliers were obliged to establish a link between these systems, with the aim of having an integrated kiln automation, simultaneously fulfilling set points for kiln operation and clinker composition. Figure 6 illustrates, in a simplified way, the concept of data exchange between the two systems.

Fig. 6 Link of COLAB and LINKMAN



The link has been in successful operation since October 1989.

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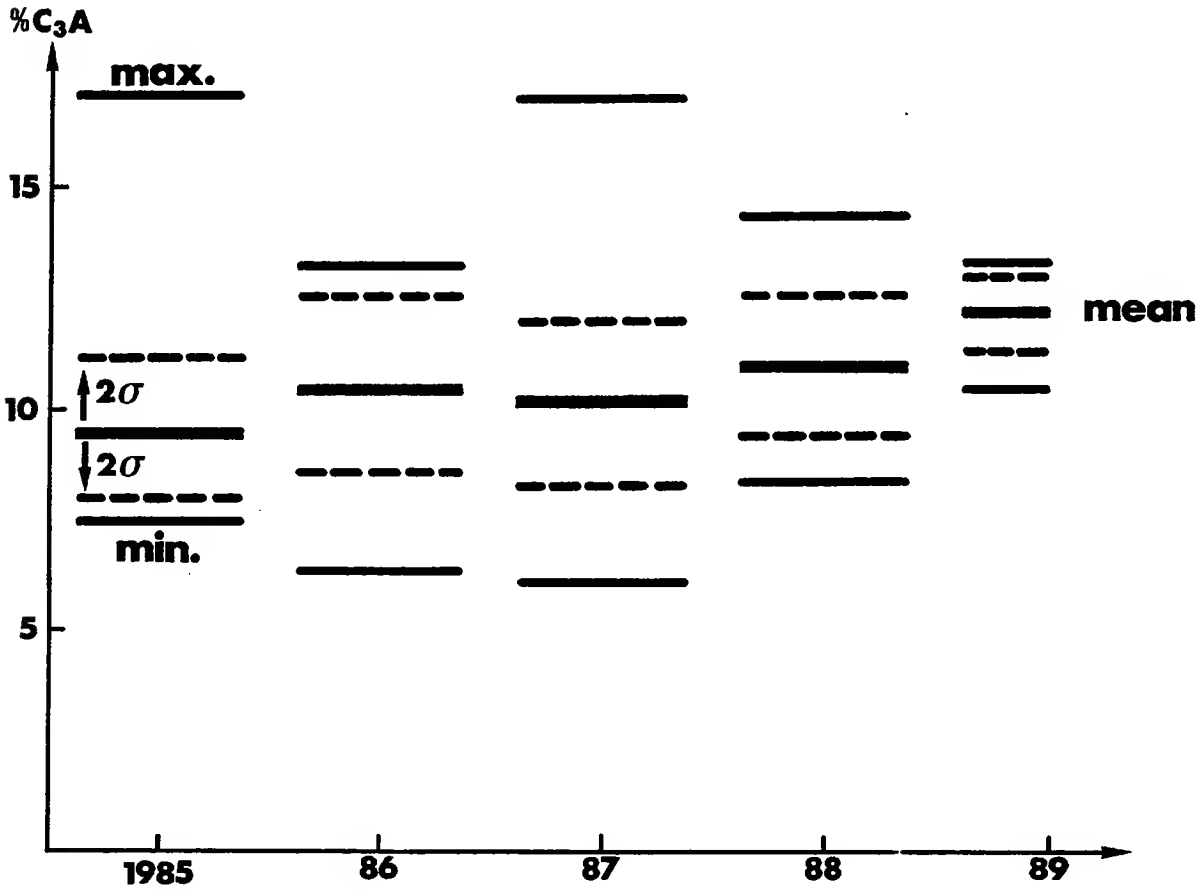
2.1.5 Experiences with the POLAB

Fig. 7 C_3A: Daily Mean Sample.



The experiences can be summarised as follows:

- The ash content in clinker could be raised close to 20%.
- The clinker uniformity could be significantly improved; C3A can be controlled within + 1%, as compared to previous +3% (see Figure 7).
- The slurry preparation and control concept are successful; production bottlenecks are eliminated. On-line control of the components flow rate is anticipated.
- The K2O-content of clinker turned out to be a rapid and significant indicator of the burning degree.
- Technical difficulties, mainly in the sector of slurry sampling and preparation - causing delay in commissioning - could be solved in a co-operative manner.
- The guaranteed systems availability of 95% could eventually be met.
- The formation of personnel to operate and maintain the system became far more demanding and time consuming than envisaged. Preventive maintenance and rapid trouble-shooting are essential to achieve and maintain the required high availability.

2.2 The Control Concept at Gmunden (Lit. 2)

The automation system at the Hans Hatschek cement plant at Gmunden, Austria, was supplied by Siemens (with Pfaff and Herzog as subcontractors). It will integrate the raw meal (already realised), clinker and cement control in three stages.

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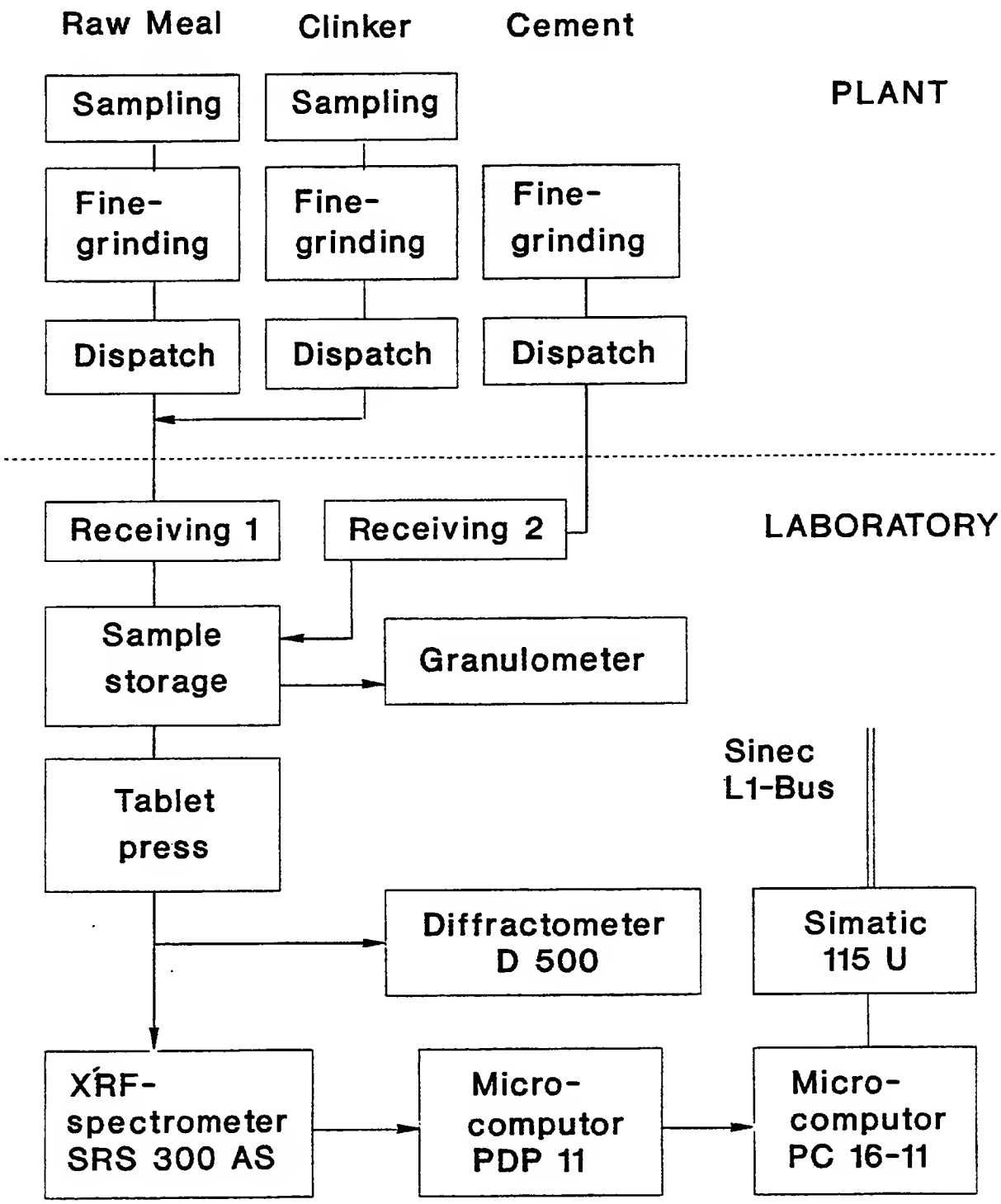
The concept is shown in Figure 8 (simplified)

Fig. 8 Automatic Concept (Gmunder Zementwerke Hans Hatschek)



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In comparison to the system at Obourg, the following aspects are worth commenting.

- The sample fine-grinding is done at the place of sampling. Advantages: no bottlenecks in sample preparation in the central laboratory; less risk of sample contamination. Both untreated (for fineness determination) and fine ground samples are sent to the laboratory.
- The sequential XRF-spectrometer is also used for general analytical purposes. The software ensures that samples arriving automatically are given priority over manually introduced samples.
- The clinker free lime will be determined by X-ray diffraction, on samples prepared in the same way as for XRF-analysis.
- The future automated cement control shall include the following control parameters:
  - Fineness
    Presently evaluated alternatives are:
    * Automated Blaine-metre, AB500 by Pragoinvest, Czechoslovakia
    * Laser-Granulometer, either with wet-dispensing system (e.g. CILAS) or dry dispensing system (e.g. Sympatec). The former would necessitate recuperation of ethanol.
  - A major criteria will be the correlation between possible fineness parameters (Blaine, size fraction) and standard mortar strength.
- Chemical composition by XRF-analysis. In addition to SO3 determination for gypsum dosage, the determination of additional guide-elements for fly-ash and slag proportioning is anticipated.

3. DECENTRALISED CONTROL SYSTEMS

3.1 Raw Meal Control at Altkirch

Until recently, on-line raw meal control was carried out by means of an XEG (X-ray emission gauge) system installed at the exit of the raw mill. The raw meal composition was continuously determined by XRF-analysis directly on a by-passed stream of raw meal.

As the analytical accuracy has always been affected by matrix effects due to the fluctuating quartz content in the raw meal, one requirement for the necessary replacement of the system was to include conventional XRF-spectroscopy on fused beads. For reasons related to the plant layout, the new system should again be installed close to the raw mill, accepting the risks associated with a very limited systems supervision.

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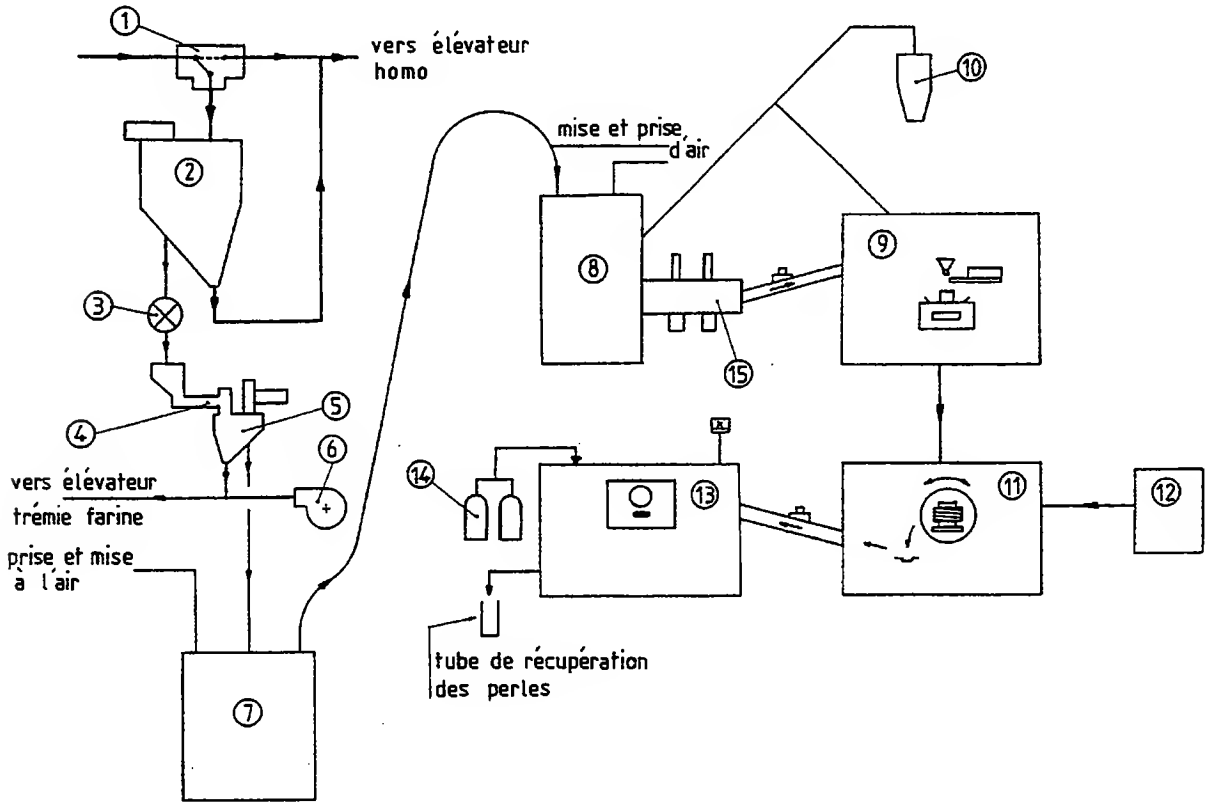
The new on-line control system was supplied as a turnkey project by Phillips France (Figure 9).

Fig. 9 Ciments D'Origny



Rep	Désignation	Fournisseurs
1	Vanne de dérivation	KIVAIR
2	Qualdoris 30	MINEMET
3	Ecluse rotative	Devance Industries
4	Vibrant	AEG
5	Qualminos	MINEMET
6	Ventilateur	Delta neu
7	Transport pneumatique	PFAFF
8	Transport pneumatique	PFAFF
9	Mapp x	Phillips
10	Dépoussiérage	PFAFF
11	Perl x -2-	Philips
12	Refroidisseur SCU 4E	F&R Cooling Ltd.
13	Spectrométre	Philips
14	Argon Méthane	Alphagaz
15	Récupération farine	PFAF

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The following features deserve comment:

- Application of a robot for sample manipulation (MAPP'X):
  Although in this application the task of the robot is limited to control the weighing of sample and fusing agent, the principal advantage of laboratory robots is seen in their flexibility to be programmable for a wide range of conventional laboratory equipment, equipment which can also be manually operated.
- Application of a low-power (200 W) XRF-spectrometer PW1660.
  In contrast to previous instruments, the performance of the latest generation of low-power XRF-spectrometers (PW 1660, Oxford QX) appears to be quite satisfactory, unless special emphasis is laid on the speed of analysis and on analysis of heavy or very light elements.

3.2 Clinker Control

3.2.1 Conductometric free lime determination

Automatic free lime analysers, particularly designed for installation outside the laboratory, e.g. close to the clinker cooler, are offered by two suppliers in France, namely ISYS and ITECA. The layout is shown in Figure 10.

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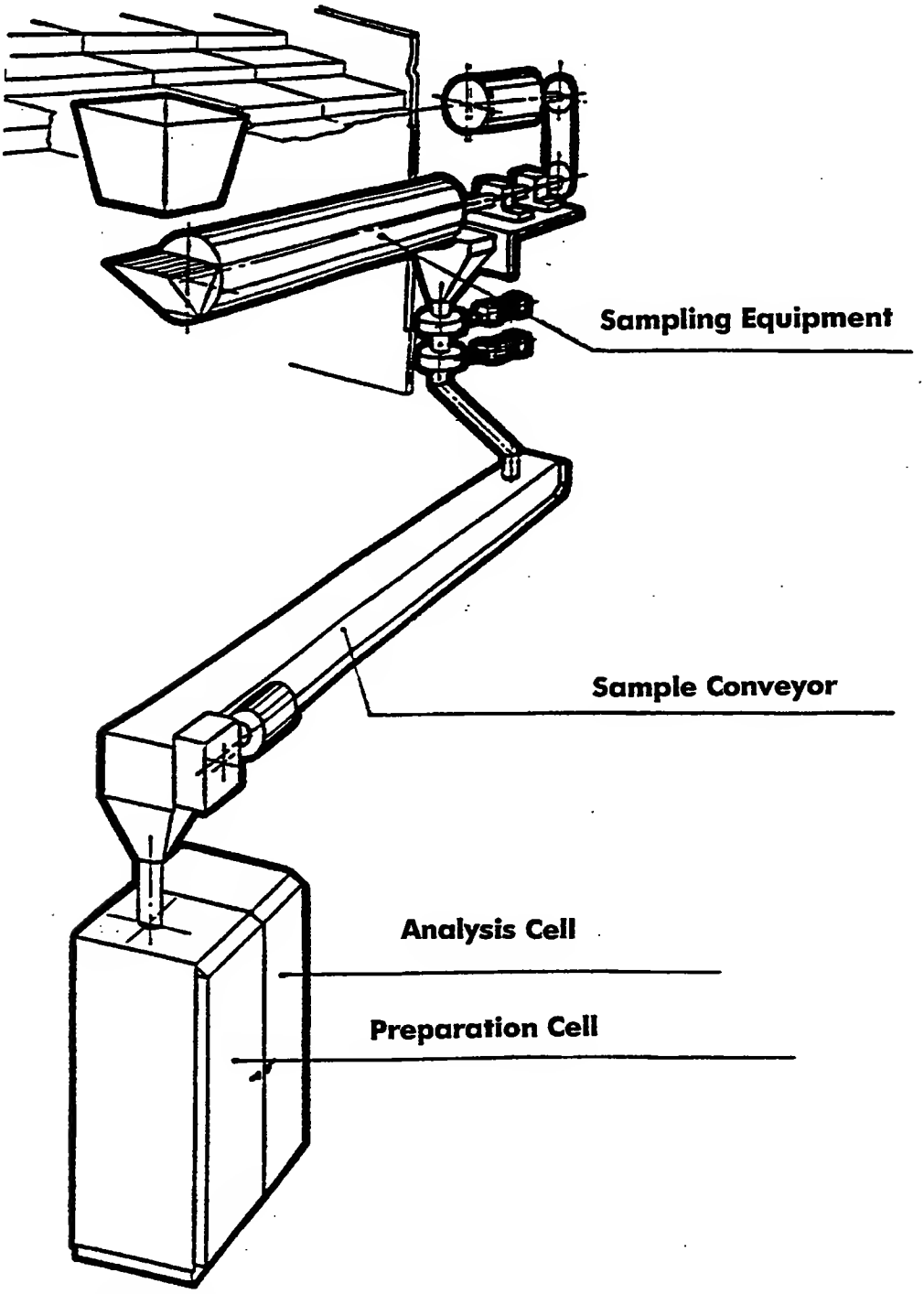
Fig. 10 ITECA Free Lime Analyzer.



In a usual arrangement, clinker samples are taken underneath the cooler grate. Free lime is determined by extraction of the ground clinker with ethylene glycol and measurement of the resulting conductivity of the solution. The results are transmitted to the central control room or laboratory.

Such systems are, with good experience, fairly widely applied in French and Belgium plants, including the plants of Ciments d'Origny and Ciments d'Obourg (Lit. 3).

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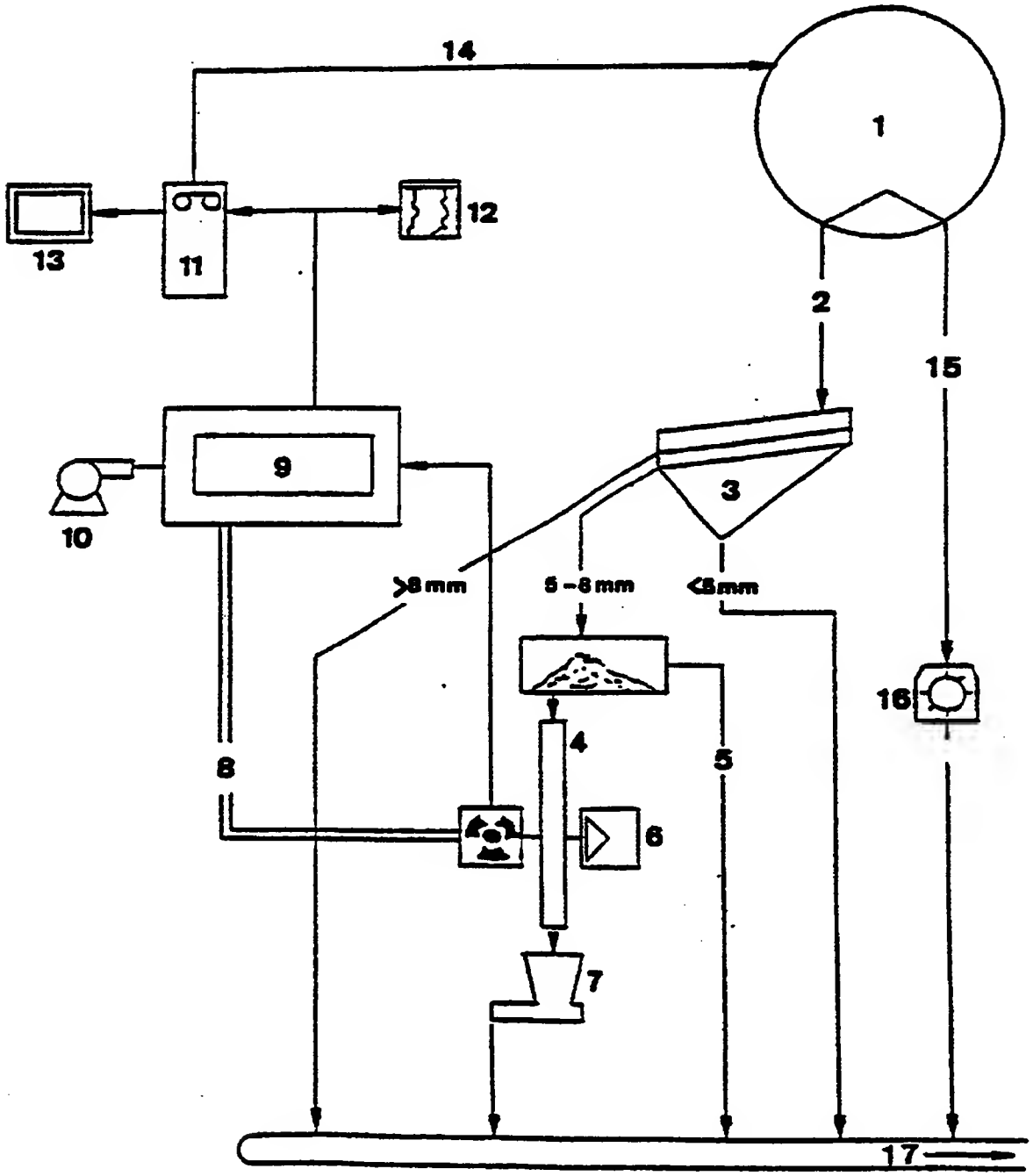
3.2.2 Automatic litre weight determination

At the Eclepens plant of the 'Societe de Ciments et Betons', Switzerland, the manual litre weight determination has successfully been replaced by an automatic on-line system, measuring the apparent clinker density with a nuclear densitometer (Lit. 4). The system is schematically shown in the next section -Figure 11a, along with the correlation of automatic and manual litre-weight measurement (Fig. 11b).

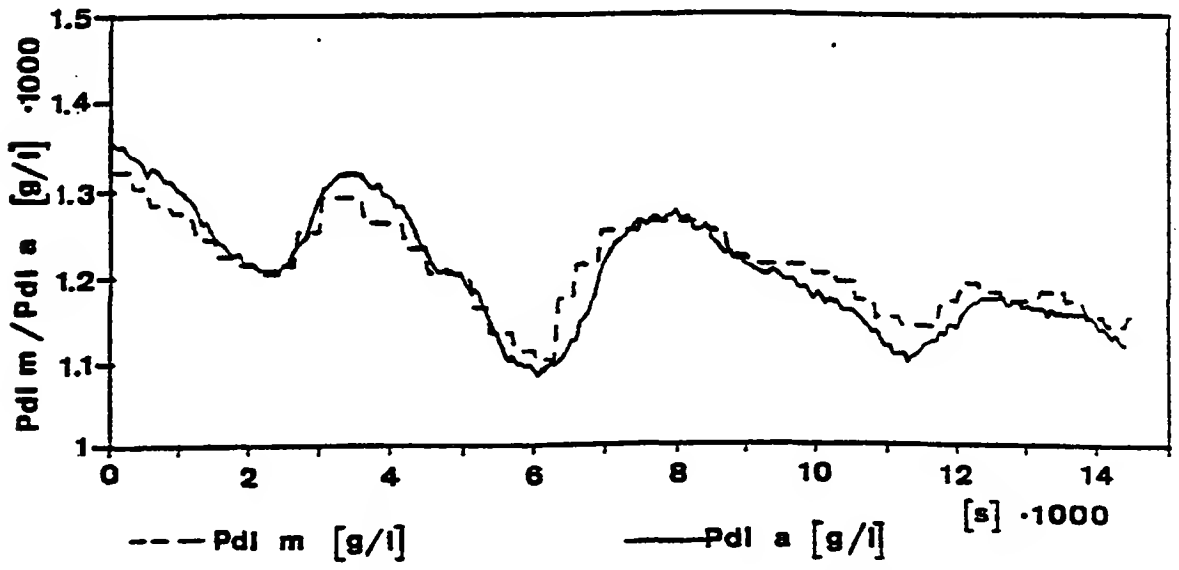
3.2.3 On-line Litre-weight Determination of Eclepens

Fig. 11a **Circuit diagram for continuous measurement of the apparent density of clinker.**



1) Kiln
2) clinker from grate
3) vibrating screen
4) measuring cell
5) overflow
6) gamma-ray source
7) variable vibrating funnel
8) cooling system
9) measuring electronics
10) cooling fan
11) process computer
12) analogue recorder
13) monitor screen
14) kiln control
15) clinker silo
16) clinker crusher
17) clinker to silo

Fig. 11b **Development of manual and automatic measurements of density over a period of 4 hours.**



It is worth noting that the density measurement is used as the primary set-point parameter for the automated kiln control system.

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3.3 Physical Cement Testing

Automated, computer controlled equipment for physical cement testing has been available on the market for several years, in particular from the German company RK TONI TECHNIK. Such equipment comprises individual testing instruments up to integrated testing systems for mortar preparation, VICAT-testing, specimen storage and strength determination. A comprehensive review is given in Lit. 5.

While automated testing instruments such as Blaine-meter, laser granulometer, VICAT-testing apparatus, penetrometer etc. are frequently used in plant laboratories, integrated testing systems are not often applied we are presently not aware of any such installation in 'Holderbank' Group plants. It would appear that the potential for rationalisation and cost-savings is too small with the usual testing frequency in plant laboratories.

4. BULK MATERIAL CONTROL BY NEUTRON ACTIVATION ANALYSIS

The technique of prompt gamma neutron activation analysis (PGNAA) has already existed for a few years for on-line analysis of bulk materials. ~ Commercial equipment is presently available from Gamma Metrics, USA.

The principle of the method, and some characteristics relevant to its application are summarised in Figure 12 to 15.

Fig. 12a How Activation Analysis Works.

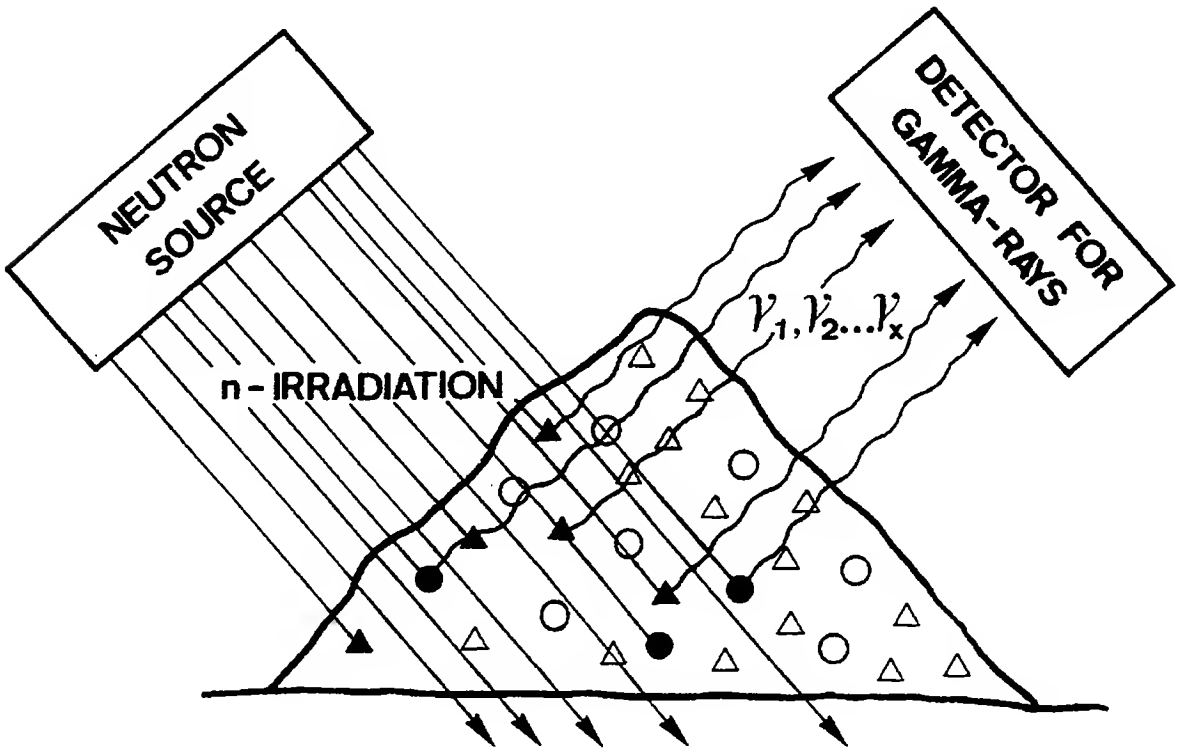

Fig. 12b Principles of Operation.

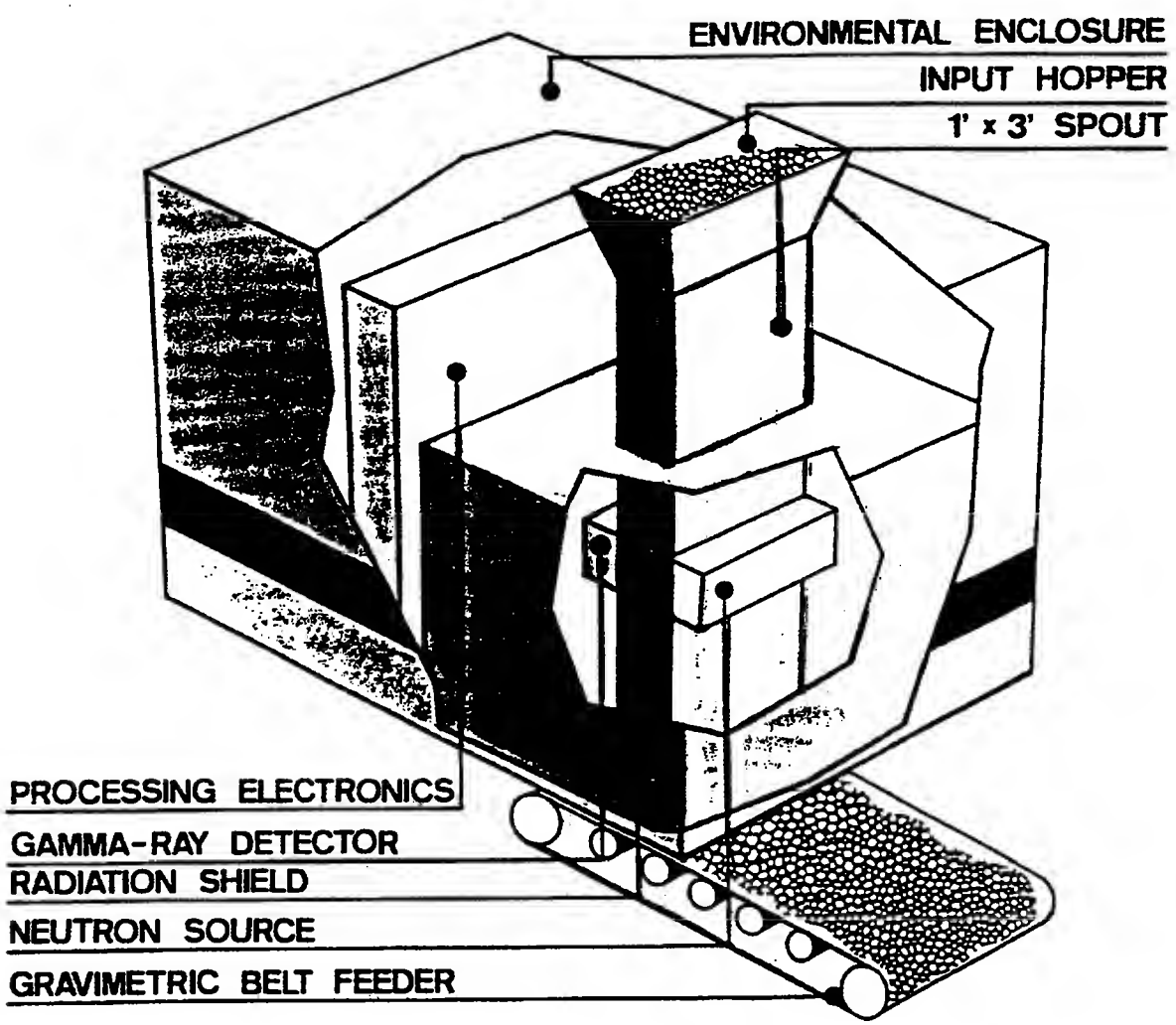

Fig. 13 PROMPT GAMMA NEUTRON ACTIVATION ANALYSIS (PGNAA)

Application:

- On-stream analysis of crushed raw materials or coal
- Blending control upon filling of integrated pre-blending becs
- Blending control on feed to raw mill
- Blending control on raw meal

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Fig. 14 PROMPT GAMMA NEUTRON ACTIVATION ANALYSIS (PGNAA)

Advantages;

- Analysis on entire material stream maximum 1000 t/h
- Analysis on crushed material Ømaximum 100 mm
- Continuous, real-time analyses
- Multi-element analysis, incl. moisture
- Permits rapid reactions in control loops
- Low maintenance requirements
- High availability

Fig. 15 PROMPT GAMMA NEUTRON ACTIVATION ANALYSIS (PGNAA)

Disadvantages:

- High investment costs, excluding installations for material transport
- High service costs
- Problematic material flow with moist and sticky materials
- Relatively high threshold concentrations for certain elements (Na2O, K2O, MgO)
- Low precision at short measuring times
- Subject to regulations regarding handling of radioactive sources

Apart from the Phoenix plant in USA, the longest experience with such equipment has been made at the cement plant of the 'Compagnie des Ciments Belge', in Belgium, where it is installed ahead of the raw mill for raw mix blending control. The experience is apparently very satisfactory, both in achieving a uniform raw meal and a very high systems availability (Lit. 6).

Nevertheless, the still limited industrial experience does not yet permit a definite assessment of the potential and restrictions of the PGNAA application in the cement industry. A thorough assessment for each individual project is, therefore, still necessary.

In the context of assessing the PGNA-analyser for the pre-blending bed control at the future Norte plant of Cementos Apasco, Mexico, HMC has carried out analyser tests with six different plant specific raw material samples at the Gamma Metrics premises.

The tests revealed that:

- a linear calibration is possible over a wide range of concentration (Figure 16a)
- the standard deviation of repeated measurements is - as expected rather poor (Figure 16b)
- the analytical results are not affected by the water content of the material.

As a first conclusion, the analyser is considered to be suitable for the control of the filling of pre-blending beds. On the other hand the consequences of the high scatter of short-term analysis for the application in raw mill control - especially with raw components of fluctuating composition - need to be further clarified.

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Fig. 16a Calibration of SiO_2 with PGNAA

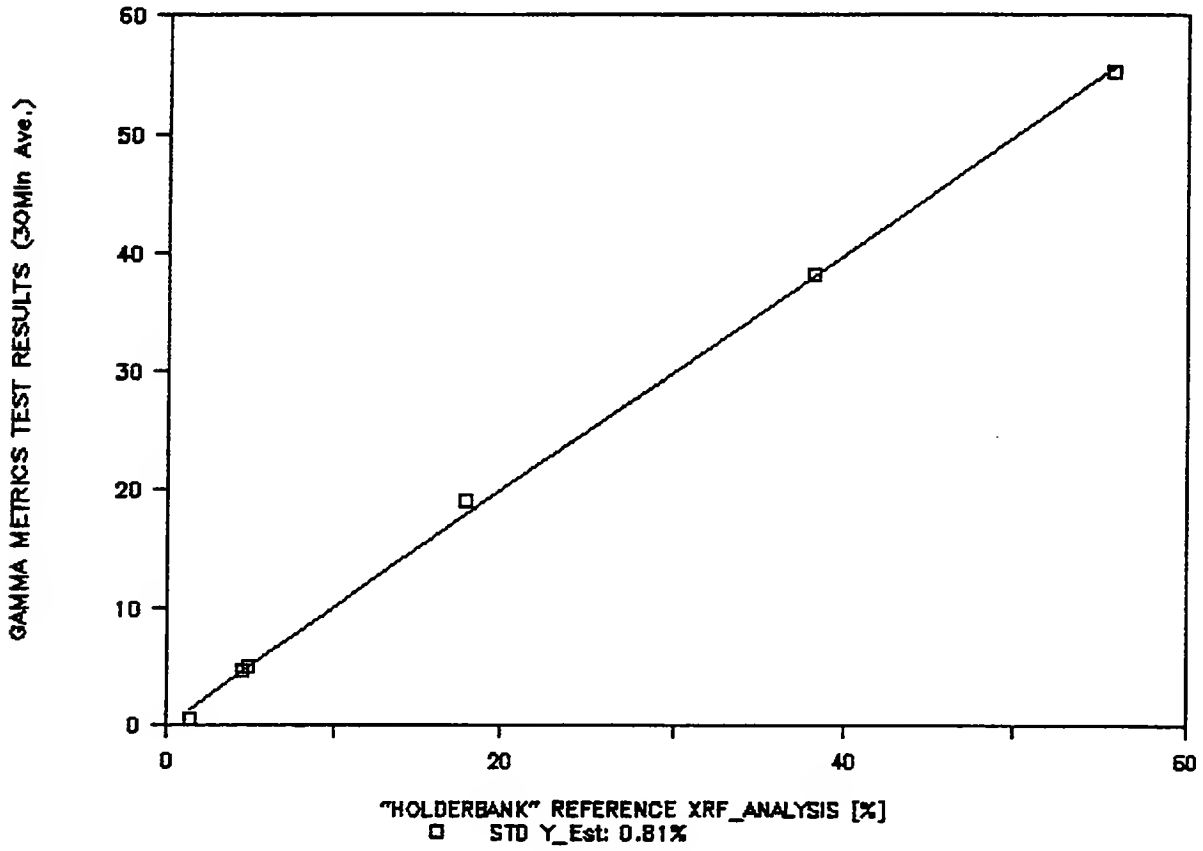

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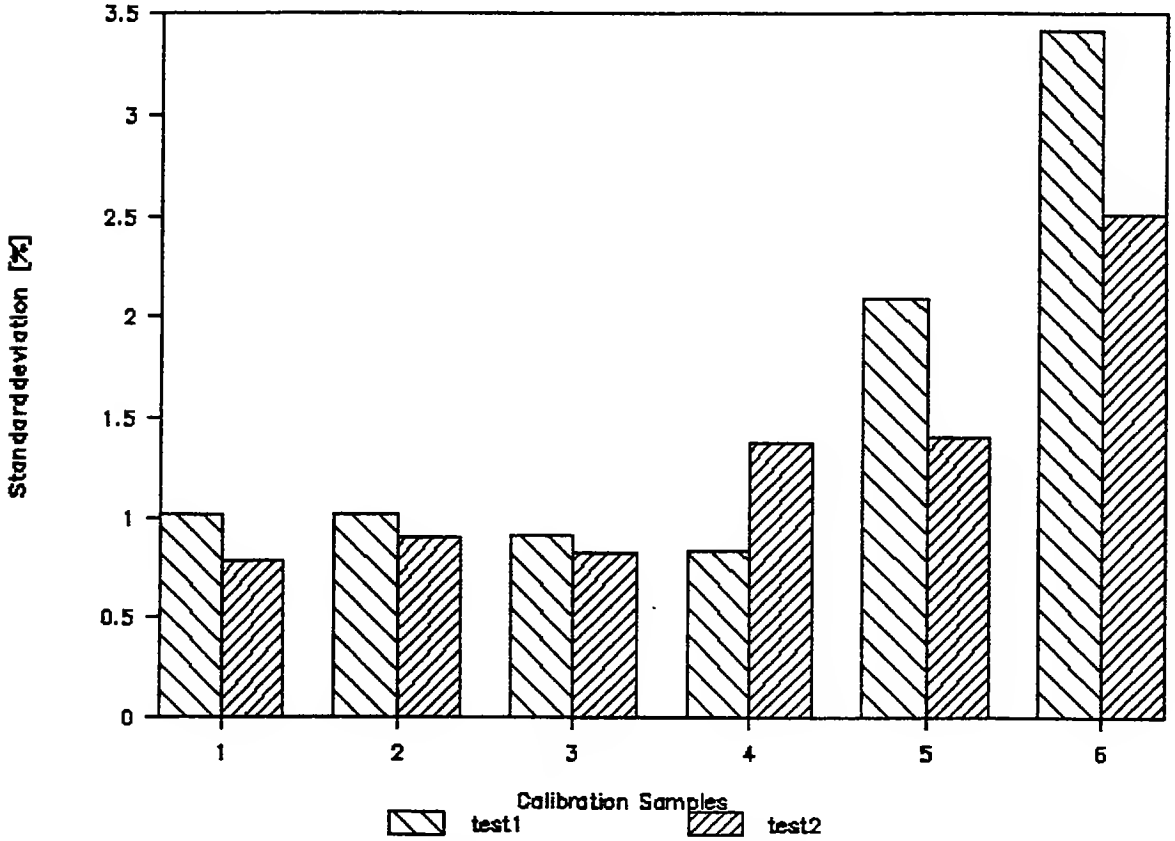
Fig. 16b PGNAA Reproducibility of SiO_2



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Chapter 13

Statistics

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PREFACE

For appropriate process and quality control in the cement and concrete industry, a large number of data are derived. Optimum benefit is, however, only achieved if these are adequately processed and interpreted. Statistics is one of the important means to make best use of the data be it by application of numerical methods and/or by graphical representation.

The present handbook describes the relevant basic definitions, formulae and applications of statistical methods which are useful in the cement industry. Emphasis is put on adequate data description and graphical representation to ensure reasonable processing and interpretation of statistical data. Most of the described procedures are illustrated with practical examples.

Chapters 1 and 2 are concerned with the basic ideas of statistics, the rules for the representation of a given set of data (graphical representation, numerical measures) and the treatment of outliers.

Chapters 3 to 8 present some statistical methods, useful for decision making, experimentation and process control.

Of special practical significance is the Application Section (chapter 5.2), which includes a procedure manual with a general check list and a collection of important test procedures. Chapter 9 gives an outwork to more sophisticated statistics.

Appendix I contains a selection of practical examples to illustrate applicability and interpretation of the demonstrated methods. A useful work sheet to construct frequency tables and to check the data for normality is given in Appendix II. Further Appendices contain: the required statistical tables for the determination of confidence limits and the application of test procedures, a list of recommended literature and an index of examples used in the text.

A subject index (English, German, French) of statistical terms is provided in Appendix VI.

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1. INTRODUCTION

Statistics is concerned with methods for collecting, organising, summarising, presenting and analysing data, obtained by measurement, counting or enumeration.

With descriptive statistics a given set of observations is summarised or presented to get a quick survey of the corresponding phenomena.

In a more sophisticated analysis of representative samples statistical inference (inductive statistics) allows conclusions to be drawn about the entire population.

A sample is considered to be representative only if it is drawn from the population at random. Such a group of n observations is called a sample of size n.

Some main topics of interest in statistical analysis are:

a) Location of the data:
Where are the observations located on the numerical scale? This question leads to the use of central values as the mean or the median.

b) Variability or dispersion:
Problems concerning the degree to which data tend to spread about an average value.

c) Correlation:
Degree of dependence between paired measurements, e.g.: Is there a real dependence of mortar strength on the alkali content in clinker?

d) Regression:
Fitting lines and curves to express the relationship between variables in a mathematical form especially used for prediction and calibration.

e) Splitting variability:
Looking for the importance of several causes to the variability of observations. Variability arises due to different components of random errors. Special experiments must be designed to split these components.

The use of statistical methods and the interpretation of statistical results requires a certain comprehension of variability. If ten pieces of coal from a delivery are analysed as to their water content, the results are not identical with the water content of the full quantity. Rather we have ten different results in a certain range. This variability of data arises not only from the fact that the pieces really have different water contents, but also because the results are influenced by three different types of errors which may occur in every set of observations:

Random errors cannot be avoided. They are due to imprecise measurement, rounding of data, environmental effects and not identically repeatable preparation. The amount of random errors may be expressed by statistical measures of variability.

Systematic errors lead to a bias in location, but not necessarily in dispersion. A bias in dispersion is obtained if several systematic errors are mixed.

Example: Every laboratory assistant has usually his own systematic error. This error may be relevant or not and it may change with time. But we always expect a greater variability of the observations if more than one person have performed the measurements. On the other hand, we recognise in this example that a small variability does not necessarily lead to a better average value. Environmental effects may be systematic too (e.g. air humidity).

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Gross errors are "wrong" values in the set of observations. Experience shows that 5% to 10% of gross errors have to be expected in a data set. Reasons may be: wrong reading of scales, errors in copying data, data not legible, miscalculations, gross error in measurement. Gross errors have a considerable influence on statistical results.

Careful measurement and data handling is, therefore, important. Data have to be inspected for outliers before a statistical analysis is performed.

Conclusion: The deviation of the single results from the true mean of the full quantity originate from real differences between the samples on one hand, and from the occurrence of several types of errors on the other.

It is often stated that "everything can be proved with statistics", this reasoning clearly is wrong; false or often only misunderstood statistics originate from insufficient representations, application of wrong procedures or assumptions, or misleading interpretation of the results. Especially graphical representations can easily be manipulated. It is therefore essential that the applicant of statistical techniques knows what he can and what he cannot do! An amusing booklet dealing with such statistical "lies" is "How to Tell the Liars from the Statisticians".

In our days of growing computer use, a lot of powerful (and sometimes less powerful or even poor) statistical software packages are available, leading to extensive use of statistics procedures by nonstatisticians or people not having enough statistical background. These packages manage nearly any instruction without being able to decide whether the statistical procedure is appropriate. This bears a great danger of use of statistics by statistical amateurs or ignorants. It is therefore absolutely necessary that the user of statistics has a solid statistical education.

2. SIMPLE DATA DESCRIPTION

In this section some descriptive methods are presented to get a quick survey of the data. The methods are illustrated with an example of concrete strength.

Example 1

The following data represent the compressive strength of 90 cubes (20 x 20 x 20 cm) of concrete. The data listed in chronological order of measurement is called set of observations.

35.8	39.2	36.8	32.4	30.7	30.8	23.5	22.8	23.7	31.7	34.6	27.6	29.9	28.4	29.3
33.0	37.6	38.1	33.3	38.9	37.1	33.3	33.4	36.4	44.3	48.9	40.1	43.4	35.4	36.6
32.8	34.1	37.4	27.9	30.2	32.0	45.3	45.8	41.0	26.1	27.9	24.4	35.3	34.5	36.1
30.1	40.2	37.9	25.0	23.0	27.8	33.5	34.2	30.0	29.0	35.2	35.8	23.9	34.9	31.5
35.9	39.7	39.4	32.4	33.6	35.2	32.8	30.2	31.6	28.5	28.5	30.3	31.4	31.8	35.5
27.1	24.5	20.9	24.6	27.2	31.7	32.2	38.6	32.8	37.8	36.8	35.3	41.9	34.4	35.5

Generally a set of n observations is denoted by $x_1, x_2, ..., x_n$ where the index j of x_j corresponds to the number of the observation in the set.

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2.1 Graphical Representation

Although the set of observations gives the complete information about the measurements, it is little informative for the reader. A better survey is obtained by grouping the data in classes of equal length, presented in a frequency table with tally.

Rules for classification:

1) The mid points should be impressive values with a few number of digits.
2) Choose 8 to 20 classes of equal length (approx. $\sqrt{n}$; classes)
3) Class boundaries should not coincide with observed values (if possible)

mid-point	class	tally	absolute frequency	relative frequency
20.0	18.75 - 21.25	I	1	0.011
22.5	21.25 - 23.75	IIII	4	0.044
25.0	23.75 - 26.25	卌 I	6	0.067
27.5	26.25 - 28.75	卌 IIII	9	0.100
30.0	28.75 - 31.25	卌 卌	10	0.111
32.5	31.25 - 33.75	卌 卌 卌 IIII	19	0.212
35.0	33.75 - 36.25	卌 卌 卌 II	17	0.189
37.5	36.25 - 38.75	卌 卌 II	11	0.122
40.0	38.75 - 41.25	卌 II	7	0.078
42.5	41.25 - 43.75	IIII	2	0.022
45.0	43.75 - 46.25	II	3	0.033
47.5	46.25 - 48.75		0	0.000
50.0	48.75 - 51.25	I	1	0.011
Total			90	1.000

In this representation we recognise directly a minimum of 20.0 N/mm^2, a maximum of 50.0 N/mm^2, and an average value between 32.5 and 35.0. The measurements are distributed symmetrically about the average value.

The graphical representation of the tally with rectangles is called a histogram.

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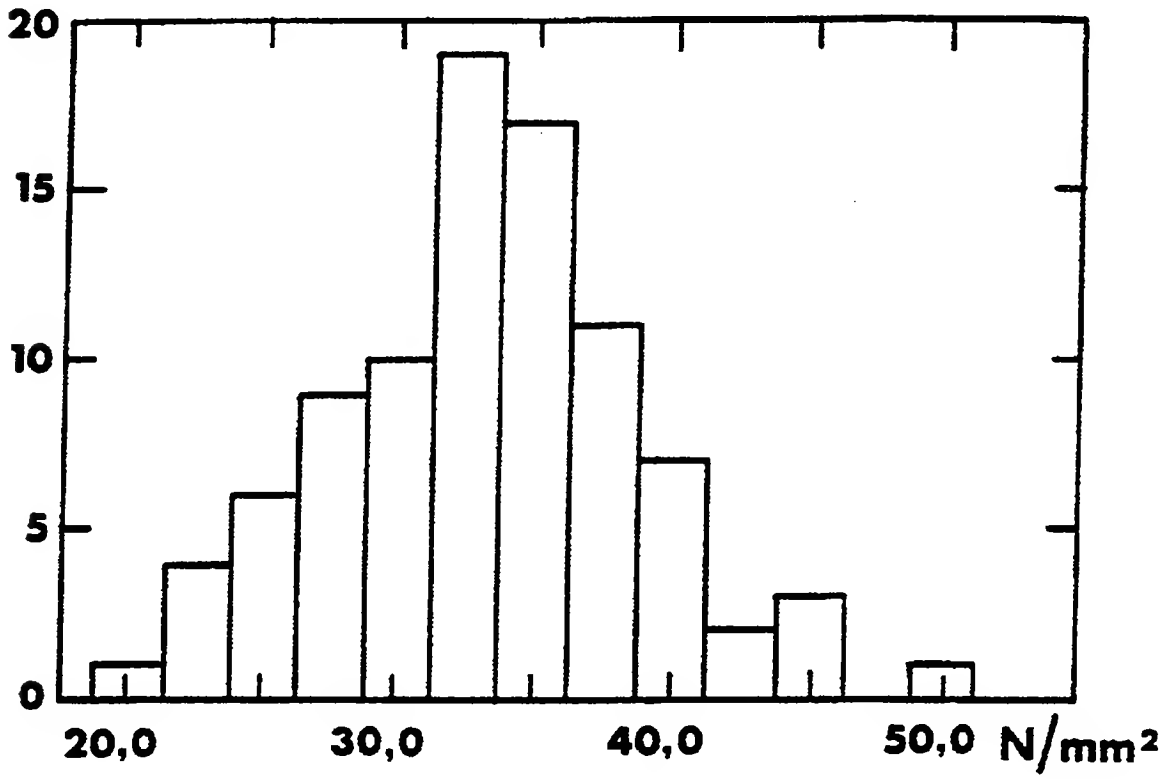
Fig. 1 Histogram.



$$\bar{x} = 33{,}2 \text{ N/mm}^2$$

$$\tilde{x} = 33{,}5 \text{ N/mm}^2$$

The area of the rectangles must be proportional to the tally not the height of the bar, e.g. if two classes contain the same number of observations and the width of class 1 is twice the one of class 2, then the height in class 1 is half the height in class 2. Same number of observations same area of rectangle!

Two further informative graphs are the frequency curve and the cumulative frequency curve. They are constructed as follows:

Frequency curve

- plot the class frequency (absolute or relative) against the class midpoint (again take into account the note for the histogram, same number = same area).

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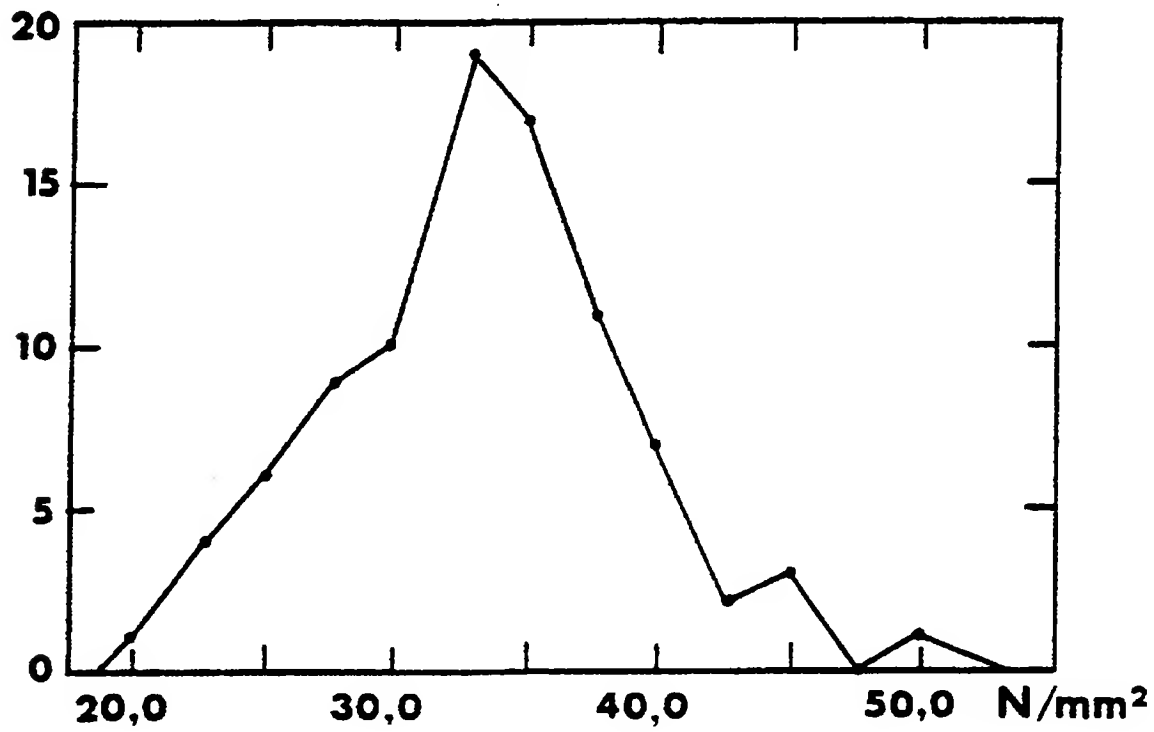
Fig. 2 Frequency Curve.



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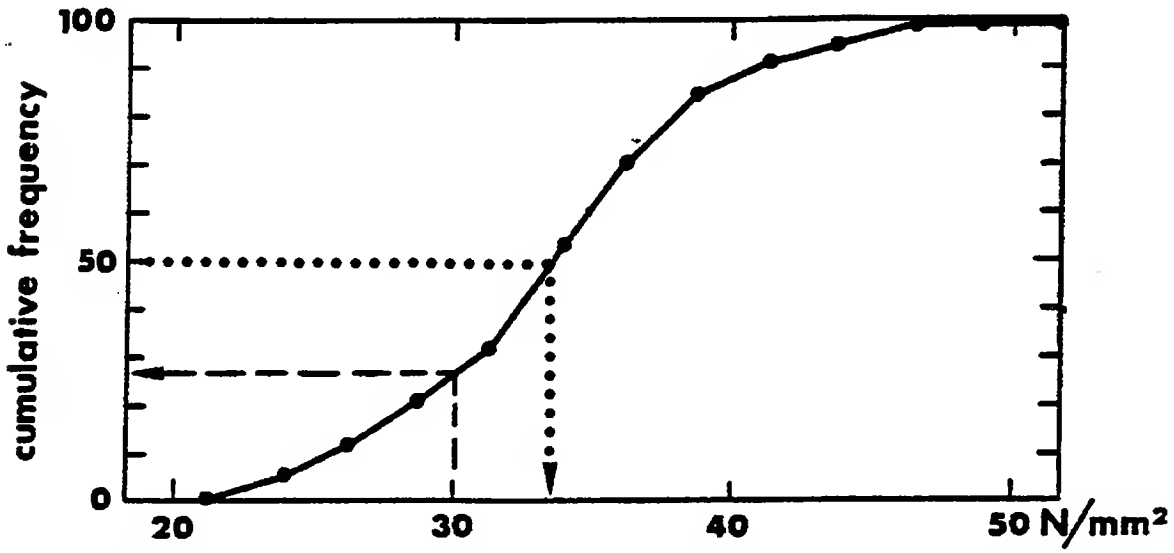
Cumulative frequency curve

- cumulate the absolute frequencies less than the upper class boundary for every class
- plot the cumulative frequency in percent against the upper class boundary.

Fig. 3 Cumulative Frequency Curve.

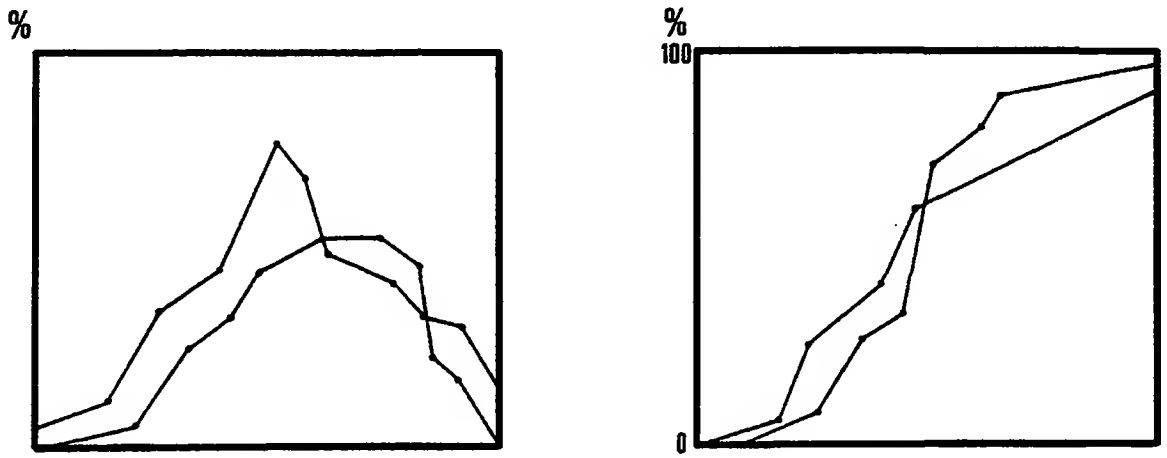


From this graph we can determine the portion of observations smaller than any given strength value x.

Example: Percentage of the observations smaller than 30.0 N/mm^2: 26 %. Inverse problem: Half of the measurements are smaller (resp. greater) than 33.5 N/mm^2.

These two frequency curves are especially suited to compare two or more different distributions with one another.

Fig. 4 Compare Distributions.



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Stem and Leaf Plot

The stem and leaf plot is very similar to the histogram but allows to reconstruct the individual data. [The stem and leaf plot is shown in the next picture, the comment does not belong to it, but shows how the data are reconstructed.]

STEM	LEAF	COMMENT
20	9	20.9
21		no observation
22	8	22.8
23	0579	23.0, 23.5, 23.7, 23.9
24	456	24.4, 24.5, 24.6
25	0	
26	1	etc.
27	126899	
28	455	
29	039	
30	0122378	
31	456778	
32	0244888	
33	033456	
34	124569	
35	2233455889	
36	14688	
37	14689	
38	169	
39	247	
40	1	
41	029	
42		
43	4	
44	3	
45	38	
46		
47	9	
48		
49		
50		

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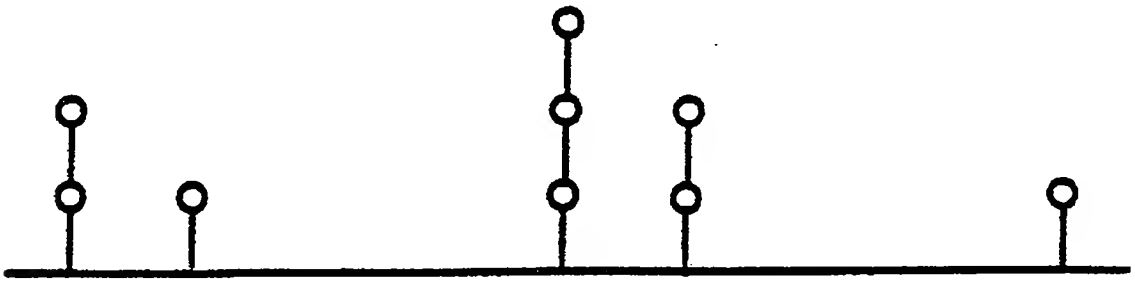
Small Samples

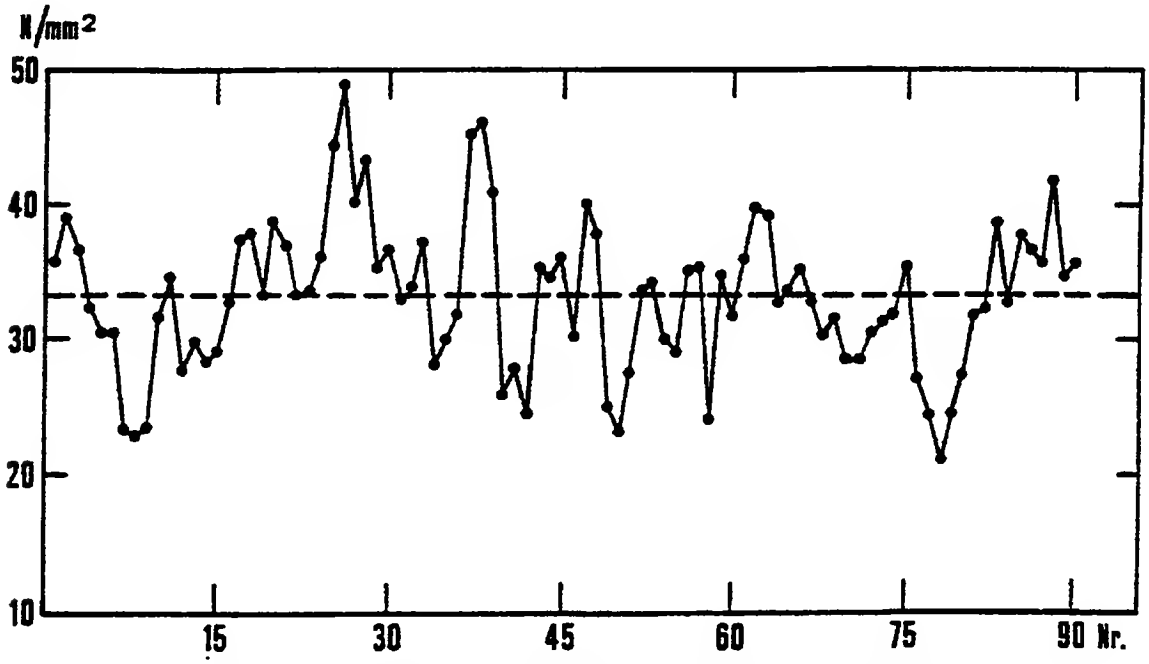
Individual values are marked on the scale

Fig. 5 Small Sample Table.



A further attractive possibility to describe graphically the distribution of a variable, the Box Plot, will be given in the next section. Often it is appropriate (especially in quality control) to plot the observations in chronological order to show a possible change of the level during the experiment.

Fig. 6 Chronological Order.



If only few data are available, the single values may be represented as points on the measurement scale (cf. example A1, Appendix I).

The scatter diagram or scatter plot is used to represent the relationship between two variables (paired observations concerning the same individual sample). The following diagrams show the dependence of concrete strength in N/mm^2 from cement/water ratio after 2 days and 28 days. Different symbols can be used to discriminate groups of observations. In the present case two groups are considered: Portland cement (PC) and blended Portland cement (BPC).

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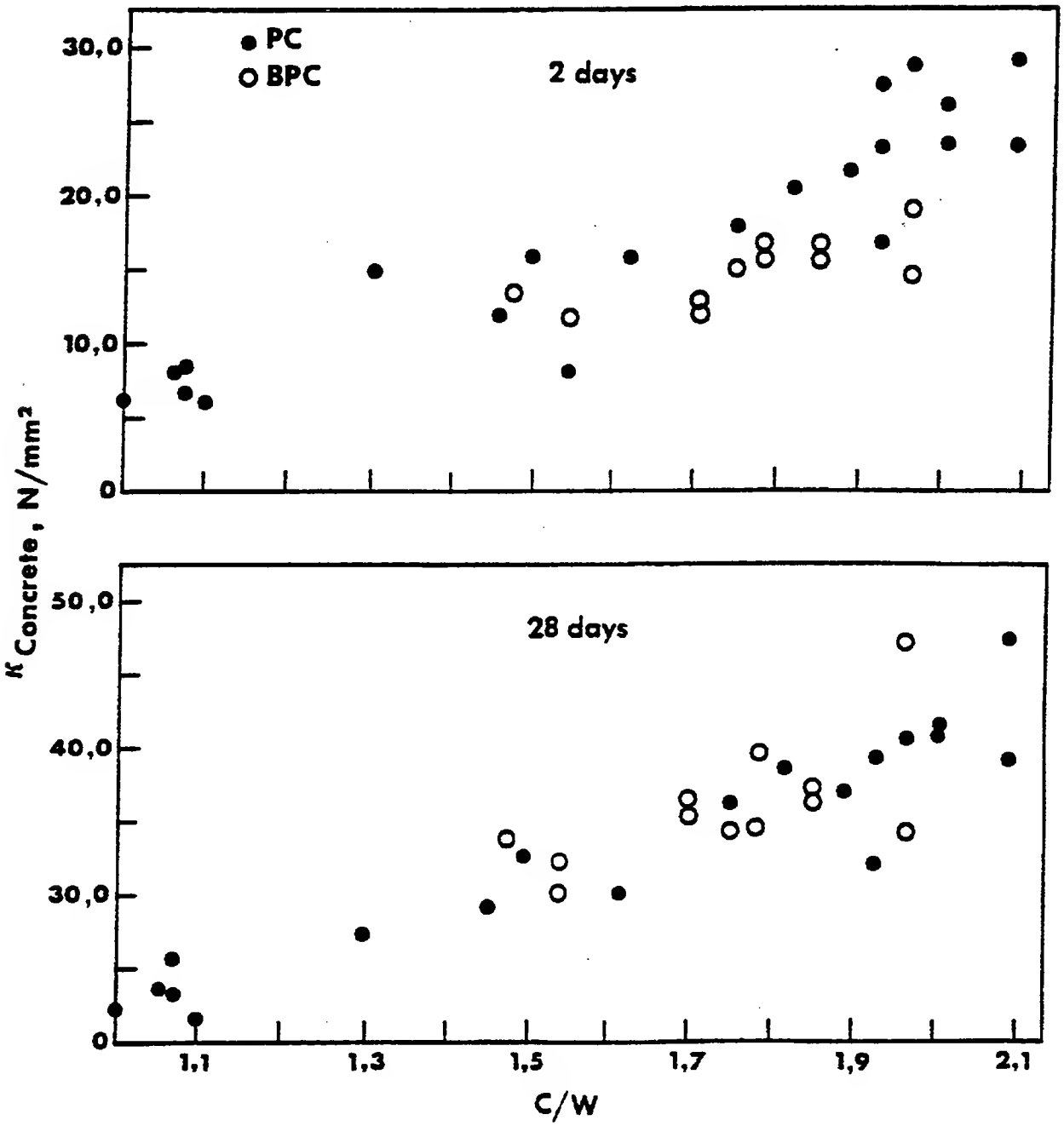
Fig. 7 Scatter Diagram.



If we are dealing with more than 2 variables a suggested graphical representation is the draftmansplot, which plots all pairwise scattergrams in one picture.

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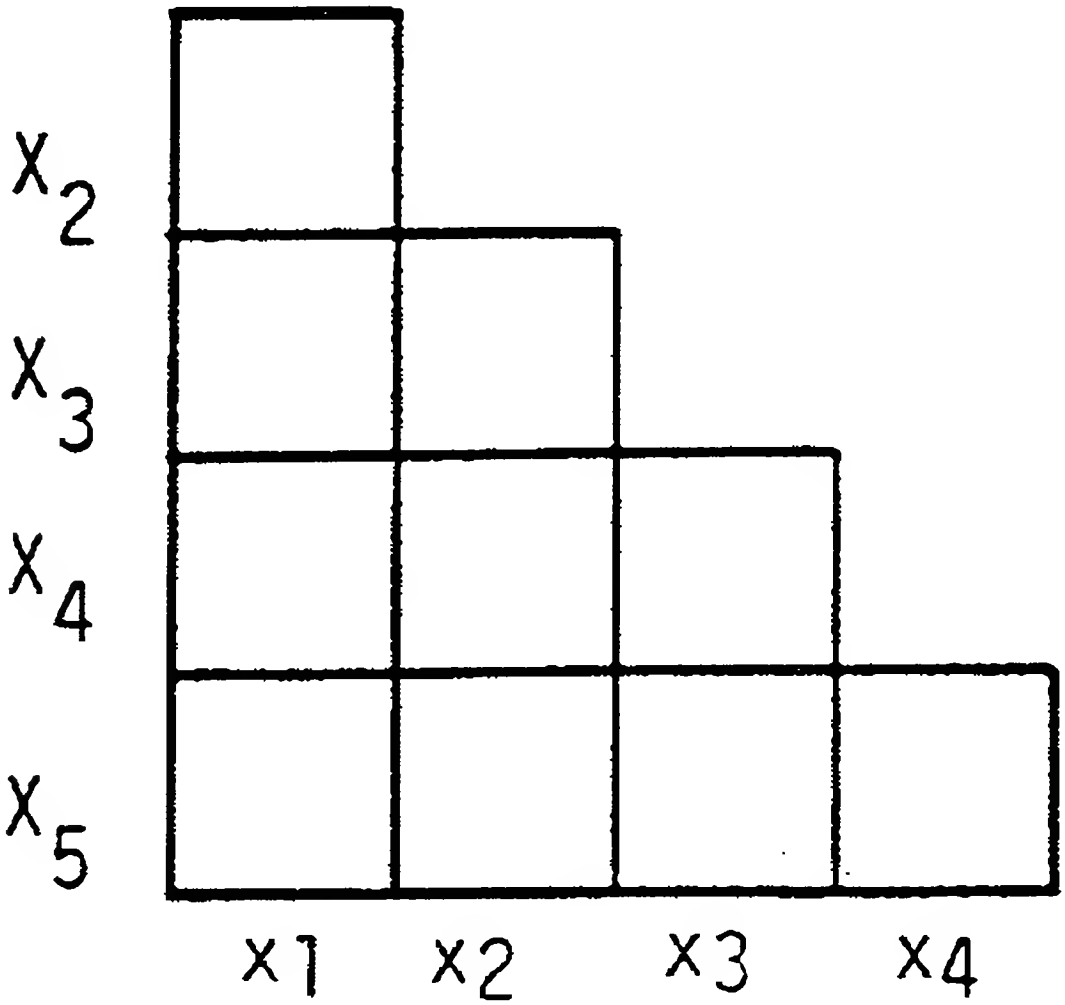

X2
X3
X4
X5
x1
x2
x3
x4

2.2 Statistical measures

As statistics deals with the behaviour of random variables, i.e. variables which take certain values with certain probabilities, we are interested in describing this behaviour of the random variable by a few characteristic measures. The first measures we will be considering are the p quantiles (or p fractiles), which are roughly said those values for which 100.p % ($0 <= p <= 1$) of the data are smaller or equal to. The p quantiles allow a full description of the data, similar e.g. to the cumulative frequency curve from Section 2.1. The use of the p quantiles gives way to construct a simple and attractive graphical representation of the data, the box plot.

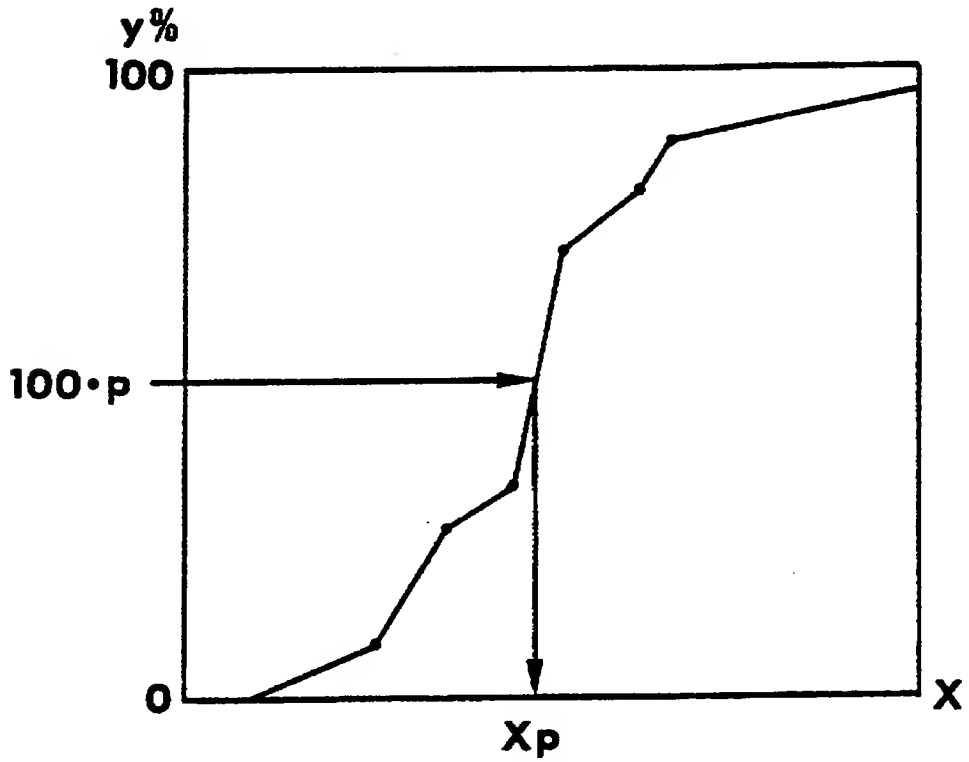
2.2.1 The p-quantile

x_p is called the p-quantile (or p-fractile) if

$100 \bullet p\%(0 <= p <= 1)$

of the values of the random variable are smaller or equal to x_p.

Based on a set of observations from a random variable x, a first method to estimate these quantiles is given by the cumulative frequency curve.



Given a sample x(i), i=1,..,n the empirical cumulative distribution function (cdf) is defined as

F(x): = (number of the x(i)'s smaller or equal to x) / n

and the following relations between the p quantile y(p) and the cdf F(x) hold

a) F(y(p)) = p

b) Y(p) = min {x(i) : F(x(i)) >= p}

The p quantile y(p) is roughly said the value for which 100*p% of the observed data are smaller or equal to.

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Estimation of P quantile v(P)

Denote by z(i), i=l, ..,n the ordered sequence of the x(i)'s, e.g.

z(1) <= z(2) <= z(3) <= .... '= z(n 1) '= z(n)

and let p(i) = (i 0.5)/n.

Then the P quantile can be computed as follows:

1) If P equals a p(i), then z(i) is the P quantile

Y(P) = z(i)

2) Otherwise compute

j(P) = nP + 0.5

and split this value j(P) into a whole number j and the remaining part B (e.g. 1.347 is split into j=1 and B=0.347). The P quantile is then

$$y(P)=(1-B)z(j)+Bz(j+1)$$

An example

Consider the following ordered sample

i	Z(I)	p(i)
1	156	0.0417
2	158	0.1250
3	159	0.2083
4	160	0.2917
5	161	0.3750
6	161	0.4583
7	163	0.5417
8	166	0.6250
9	166	0.7083
10	168	0.7917
11	172	0.8750
12	174	0.9583

12.5 % quantile (P=0.125)

As P=p(2), our 12.5 %-quantile y(.125) equals the second ordered observation, thus y(.125) = 158

10 % quantile (P=O.1)

Compute j(P)=nP + 0.5 = 12*0.1 + 0.5 = 1.7

As this yields no whole number our 10 % quantile is (j=1 and B=0.7)

y(.1) = (1 B) z(1) + B z(2)

= 0.3*156 + 0.7*158 = 157.4

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25 %-quantile (P=0.25)

Compute j(P)=nP + 0.5 = 12*0.25 + 0.5 = 3.5

As this yields no whole number our 10 % quantile is (j=3 and B=0.5)

$$y(.25) = 0.5*159 + 0.5*160 = 159.5$$

50 % quantile (P=0.5)

Compute j(P)=nP + 0.5 = 12*0.5 + 0.5 = 6.5

As this yields no whole number our 10 % quantile is (j=6 and B=0.5)

$$y(.5) = 0.5*161 + 0.5*163 = 162$$

Some of the quantiles have special names, e.g.

50 % quantile	median (M)
25 % quantile	lower quantile (LQ)
75 % quantile	upper quantile (UP)
10 % quantile	lower decile (LD)
90 % quantile	upper decile (UD)
OQ UQ	Inter quantile range (IQ)

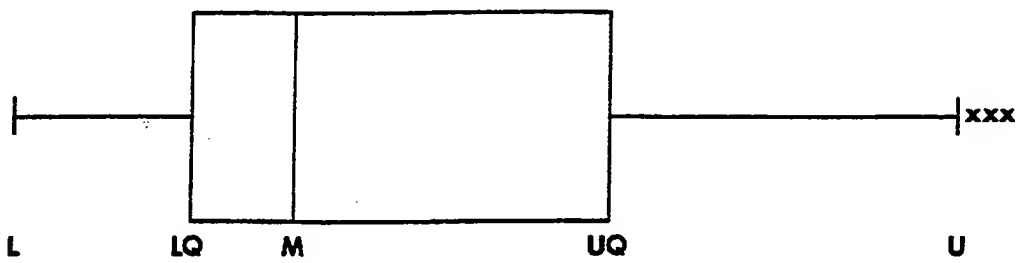
2.2.2 The Box Plot

Today the box-plot is perhaps the most frequently used graphical representation for univariate data. It offers a condensed picture of the data's distribution and shows location, variability and extremes of the data.

In order to construct the box plot we need three quartiles:

- the median
- the lower quantile
- the upper quantile

The construction is very simple:



where L = max (LQ 1.5 x IQ, min x_i)

U = min (UQ + 1.5 x IQ, max x_i)

IQ = UQ IQ

x denotes extreme values.

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This construction is based on the fact that for a normal distribution approximately 1 % of the data lie outside the interval (L,U).

An alternative method also frequently used, would be not to use L and U but instead use the lower and upper decile. Especially for non normal distributions this representation seems preferable.

2.2.3 Measures of location

The arithmetic mean is the mean of all observations and corresponds to the centre of gravity in physics.

$$\bar{x} = \frac{x_1 + x_2 + \ldots x_n}{n} = \frac{1}{n}\sum_{i=1}^{n} x_i$$

$$\bar{x} = \frac{35.8 + 39.2 + 36.8 + \ldots + 34.4 + 35.5}{90} = 33.199 N/mm^2$$

The median is the central value. Half of the observations are smaller, respectively greater than the median.

Place the measured values in an ordered array starting with the smallest value

$$X_{(1)} \leq X_{(2)} \leq \ldots \leq X_{(n)}$$

From this array we obtain the median by

$$\tilde{x} = x_{\frac{(n+1)}{2}} \qquad \text{if n is odd}$$

$$\tilde{x} = \frac{1}{2}\left(x_{(\frac{n}{2})} + x_{(\frac{n}{2}+1)} \right) \quad \text{if n is even}$$

For a great number of observations the ordering is very laborious. In this case the median may be determined graphically from the cumulative frequency curve by looking for the point on the x axis corresponding to a cumulative frequency of 50 %.

Mean or Median?

If the histogram is symmetrical, mean and median are approximately the same. In this case the mean is a better estimate of central tendency if the distribution is not too long tailed.

If the histogram is skew, mean and median are different.

- Use the mean if you are interested in the centre of gravity or the sum of all observations
- Use the median if you are interested in the centre, i.e. with equal probability a future observation will be smaller, resp. greater than the median.

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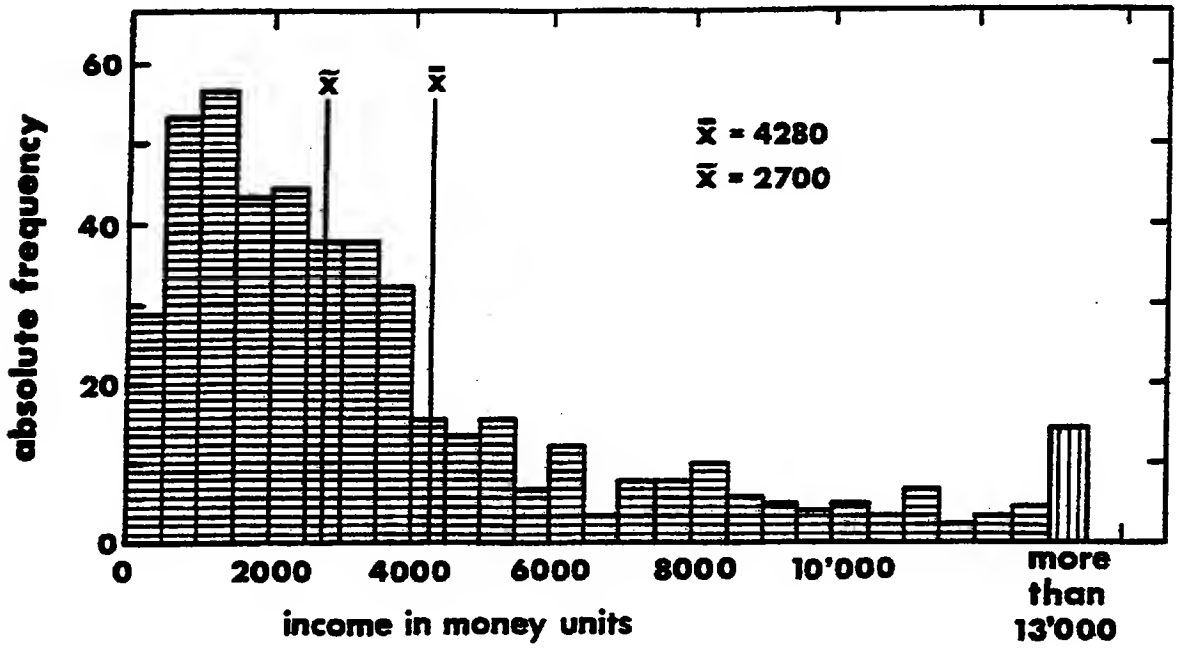
Example 2

The following histogram represents the income of 479 persons



In this extremely skew distribution the arithmetic mean is quite different from the median. Whereas the median represents a typical income, the mean can be used in a projection to estimate the total income of the population if the sample is drawn at random.

The trimmed mean is used to estimate the mean of a symmetrical distribution if gross errors in the data are suspected.

To calculate the α-trimmed mean the α-percent largest and smallest values are deleted. The trimmed mean is then the arithmetic mean of the remaining observations. Notation for the 5%-trimmed mean: $x_{\text{trimmed } 0.05}$

Usual values for α: 5 % to 10 %

The weighted mean is used if certain weighting factors w_i are associated with the observation x_i. Reasons may be:

- samples of unequal weights

- observation are measured with unequal precision

$$\bar{x}_w = \frac{\sum w_i x_i}{\sum w_i}$$

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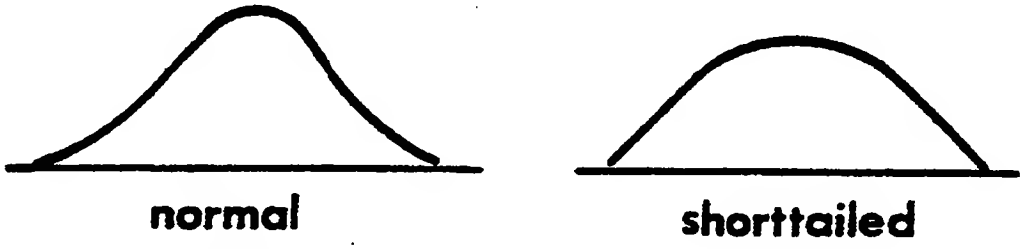
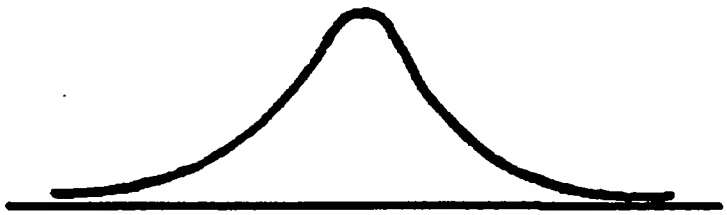
Schematic survey on the use of measures of location:

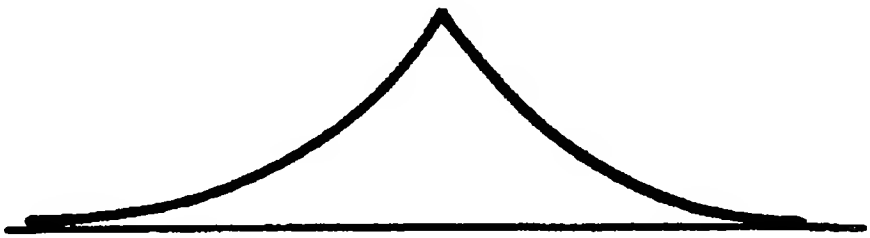
Type of distribution



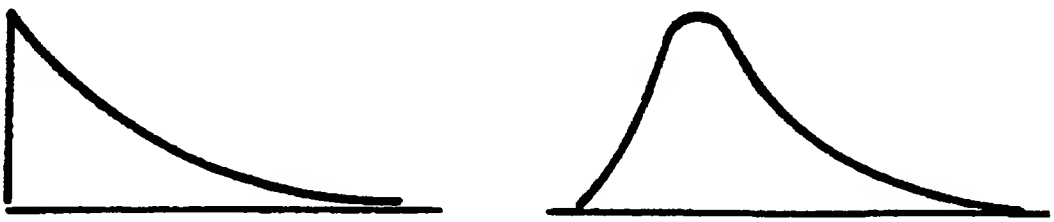
Measure to be used: arithmetic mean $\bar{x}$



Measure to be used: trimmed mean $\bar{x}_{trimmed}$



Measure to be used: median $\tilde{x}$



Measure to be used: median or arithmetic mean. The choice depends on the interest of the user.

Remark: The median is equal to the 50%-trimmed mean.

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2.2.4 Measures of Variability

The variance is the mean square deviation of single observations from the mean.

$$s^2 = \frac{1}{n-1}\sum_{i=1}^{n}(x_i - \bar{x})^2$$

For practical computations use

$$s^2 = \frac{1}{n-1}\left(\sum_i x^2 - \frac{1}{n}(\sum x_i)^2\right)$$

The standard deviation is the positive square root of the variance.

$$s = +\sqrt{s^2}$$

In our example:

$$s = \sqrt{\frac{1}{n-1}\sum_{i=1}^{n}(x_i - \bar{x})^2} = 55.596 N/mm^2$$

The coefficient of variation is a relative measure of the variability (relative to the mean).

$$v = \frac{s}{\bar{x}}$$

v is often used for comparing variabilities. It is especially useful if s increases proportionally with the mean $\bar{x}$, i.e. v = constant.

Example of compressive strength:

$$v = \frac{5.596}{33.199} = 0.169 = 16.9\%$$

The range is the difference between the largest and the smallest value.

$$R = X_{(n)} - x_{(1)} = X_{max} - X_{min}$$

Useful with small sample sizes.

Note: The range increases in general with increasing sample size n (greater probability to get extreme values).

If the variables under consideration are assumed to be normally distributed then

the range can be used to provide an estimate of the standard deviation, according to

$s = R/k$ where $k \cong \sqrt{n}$ for $3 \le n \le 10$

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For sample size n >10 divide the set of observations in random sub samples of size n', calculate the arithmetic mean $\overline{R}$ of the range in these sub-samples, and estimate s according to

$$s \sim \overline{R}/k$$

using a k-value corresponding to n'. . For a first rough estimation the following k values may be used:

$k \sim 4$ for n = 20
$k \sim 4.5$ for n = 50
$k \sim 5$ for n = 100

The interquartile range is the difference between the upper and lower quartiles.

$Q = x_{.75} - x_{.25}$

If the distribution is symmetric $s \approx Q.\frac{4}{3}$

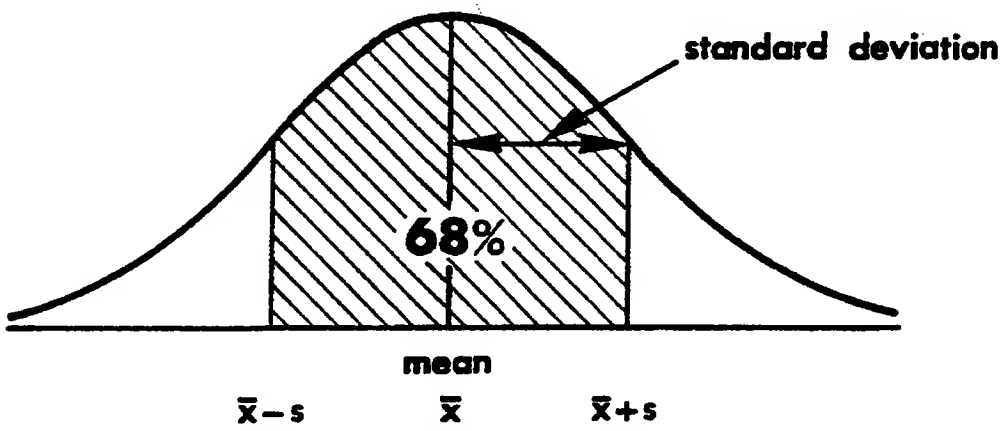
2.2.5 Statistical program packages

There are a lot of statistical programs on PCs, microcomputers and mainframes that offer the possibility to compute these statistical measures including quartiles and even to plot the box plots. The perhaps most prominent under many others are SAS, SYSTAT/SYGRAPH, SPSS, BMDP, STATGRAPHICS.

2.2.6 Interpretation of the standard deviation

In the histogram of example 1 (chapter 2.1) frequency of observations decreases on both sides of the mean. The distribution law seems to be symmetrical. In the present case we consider the 90 measurements to be a random sample of a specific distribution model, the normal distribution (or Gaussian distribution). This type of distribution is represented by a symmetrical, bell shaped curve, and often observed in practical applications.

The normal distribution is determined by the mean and the standard deviation.



For this distribution about 68 % of the observation are expected in the interval mean ±1 standard deviation.

If the distribution is not normal, the standard deviation allows no direct interpretation. In this case we can determine the interval that contains a certain portion of the observations graphically from the cumulative frequency curve.

Another possibility to get an interpretation of the standard deviation in a skew distribution is "normalising" of the distribution by transformations. In the case of example 2 (chapter 2.2.3), the logarithms of incomes show approximately a normal distribution. So called variance-stabilising transformations are especially used to fulfil normality conditions in higher statistical analysis.

For further information consult e.g. Natrella (1963).

Further properties of the normal distribution are given in section 3.

2.2.7 Outliers

As mentioned in the introduction we have to expect about 5 % to 10 % gross errors in a set of observations. Most of them may not be recognisable in the region of all other values. Some may be extremely outlying values with an important influence on statistical results.

In modern statistics robust methods are studied, which are not sensitive to a certain portion of gross errors, as for example the median or the trimmed mean. More sophisticated robust procedures are in general rather complicated.

If classical measures as the standard deviation and the arithmetic mean are used, we have to check the data for outliers and to eliminate them from the set of observations (cf. example A2, Appendix I).

Note: If outliers are eliminated, they must be recorded separately in the report.

To detect outliers, check the data plot (tally, histogram or chronological order) for suspicious values. If outliers are suspected and the data show a normal distribution, use the Dixon criterion for rejecting observations ($n<26$).

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The Dixon Criterion

Procedure

1) Choose a, the probability or risk we are willing to take of rejecting an observation that really belongs in the group.

2) If

$3 \le n \le 7$	Compute r_{10}
$8 \le n \le 10$	Compute r_{11}
$11 \le n \le 13$	Compute r_{21}
$14 \le n \le 25$	Compute r_{22}

where r_{ij} is computed as follows

r_{ij}	If $X_{(n)}$ is suspect	If $X_{(1)}$ is suspect
r_{10}	$(X_{(n)} - X_{(n-1)})/(X_{(n)} - X_{(1)})$	$(X_{(2)} - X_{(1)})/(X_{(n)} - X_{(1)})$
r_{11}	$(X_{(n)} - X_{(n-1)})/(X_{(n)} - X_{(2)})$	$(X_{(2)} - X_{(1)})/(X_{(n-1)} - X_{(1)})$
r_{21}	$(X_{(n)} - X_{(n-2)})/(X_{(n)} - X_{(2)})$	$(X_{(3)} - X_{(1)})/(X_{(n-1)} - X_{(1)})$
r_{22}	$(X_{(n)} - X_{(n-2)})/(X_{(n)} - X_{(3)})$	$(X_{(3)} - X_{(1)})/(X_{(n-2)} - X_{(1)})$

3) Look up $r_{1-\alpha/2}$ for the r_{ij} from Step (2), in Table A 2

4) If $r_{ij} > r_{i-\alpha/2}$ reject the suspect observation; otherwise, retain it.

In the case of a sample size n>25 use the following procedure:

1) Choose α, the probability or risk we are willing to take of rejecting an observation that really belongs to the group

2) Calculate $Z_B = \dfrac{X_{(n)} - X_{(1)}}{s}$

3) Look for $Z_{T,1-\alpha}$ in Table A 3, Appendix III

(4) If $Z_b > Z_{T,1-\alpha}$ reject the suspect observation, otherwise retain it.

Note: The presented outlier rejecting rules are only valid in normal distributions. In a skew distribution, the elimination of outliers is very dangerous and should be avoided. In this case the reason for the extreme observation must be known.

A check for outliers is not necessary if the trimmed mean or the median is used and if we are only interested in a location measure.

For further tests on outliers cf. "Wissenschaftliche Tabellen Geigy, Statistik".

The standard deviation is extremely sensitive to outliers.

A check is therefore important, because it is not allowed to calculate a standard deviation from a trimmed set of observations.

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3. THE NORMAL DISTRIBUTION (ND)

As mentioned in section 2, the normal distribution is a theoretical model of a statistical universe (population), defined by the mean μ and the standard deviation σ. Graphically it is represented by a smooth, symmetric mean x and the standard deviation s of the samples are estimated for the true, but unknown values μ and σ.

By the standardisation formula

$$Z = \frac{x-\mu}{\sigma} \text{ (respectively } \frac{x-x}{s} \text{ if n is large)}$$

the ND is transformed in a normalised form with mean 0 and standard deviation 1.

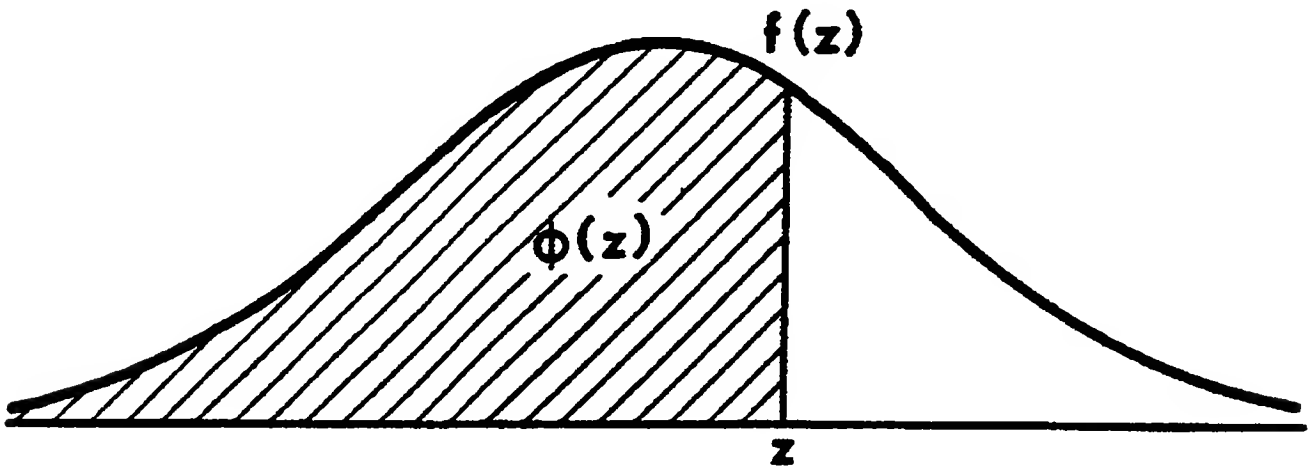
Formulae for the standard normal distribution:

Density function (bell shaped curve):

$$f(z) = \frac{1}{\sqrt{2\pi}} \exp(-\frac{z^2}{2}) (-\infty < z < \infty)$$

Especially of interest is the area under the curve (distribution function), which corresponds for every given value z to the probability of an observation to be smaller than z.

$$\phi(z) = \frac{1}{\sqrt{2\pi}} \int_{-\infty}^{z} \exp(-\frac{z^2}{2}) dz, \phi(-z) = 1 - \phi(z)$$



Numerical values for z and $\phi(z)$ are given in Table A 1, Appendix III. The probability to observe a measured value between two given limits T_1 and T_2 can be calculated as follows:

1) transform the limits T_1 and T_2 in a standardised form
2) calculate the probability with help of Table A 1 by
   $F(z_1, z_2) = \phi(z_2) - \phi(z_1)$

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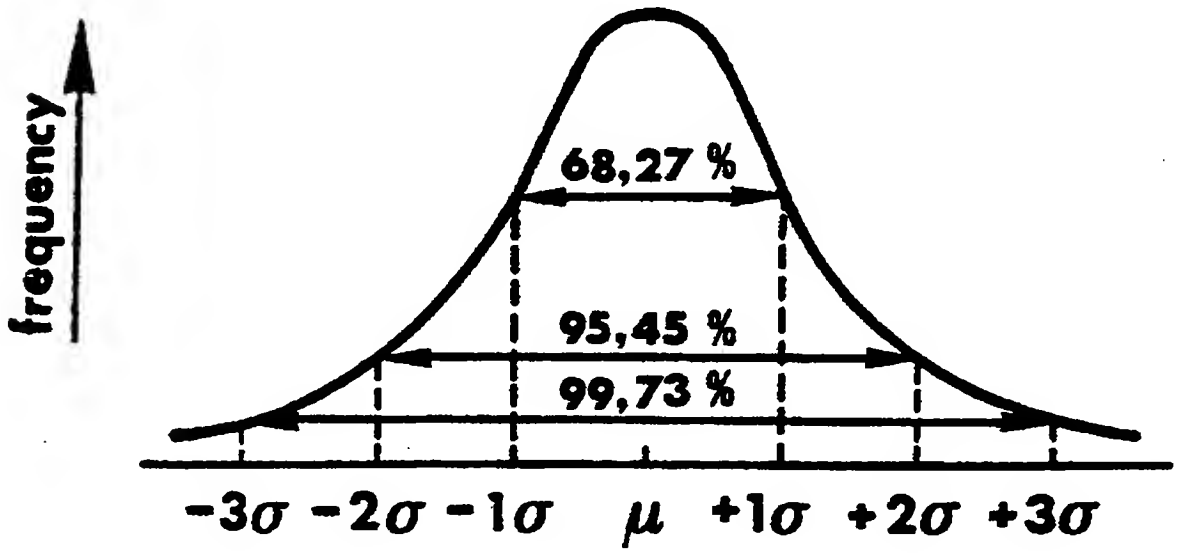
Interpretation of the standard deviation in a normal distribution:

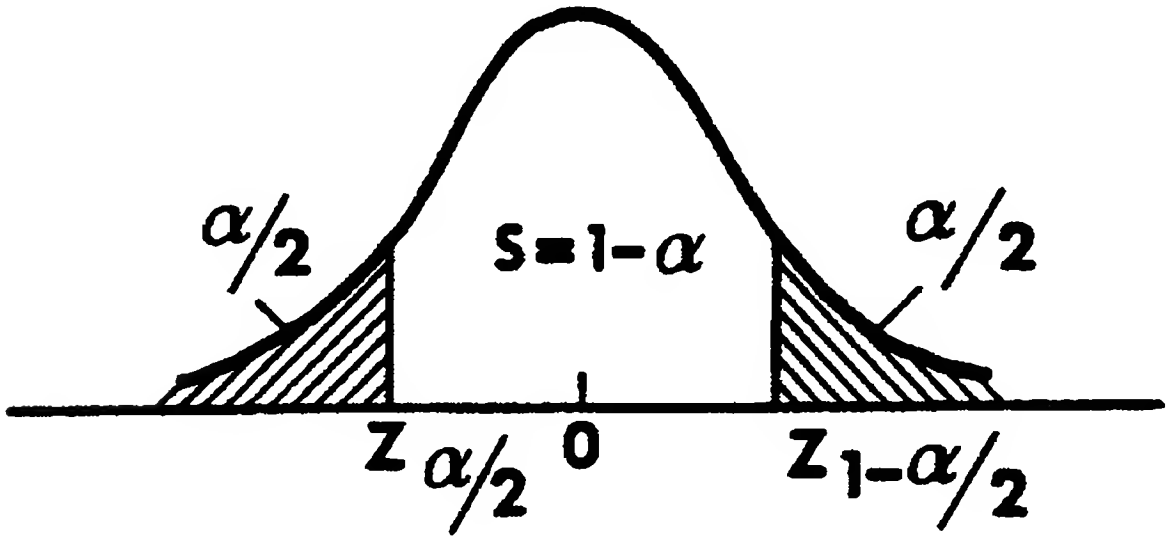


We expect about 95 % of the observation between $\mu-2\sigma$ and $\mu+2\sigma$. More general we have:

The statement that the values (of the population) lie between $\mu+z_{1-\alpha/2}$. and $\mu-z_{1-\alpha/2}$. is right with the probability S = 1 - α and wrong with probability α. One sided or two sided regions may be considered. The corresponding z-values are taken from a table of the standard normal distribution.

Usual percentiles for the statistical confidence S:

a) Two sided, $S=1-\alpha; z_{\alpha 2}, z_{1-(\alpha/2)}$



S(%)	90	95	99	99.9
-z α/2 = z_1 .- α/2	1.64	1.96	2.58	3.29

b) One sided, $S=1-\alpha; z_{\alpha}, z_{1-\alpha)}$

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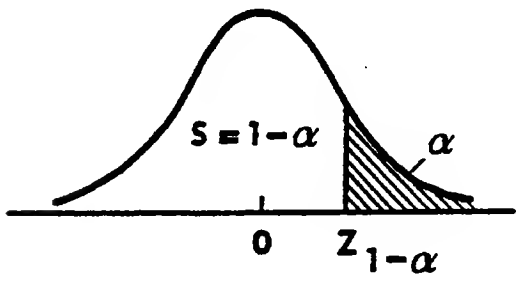

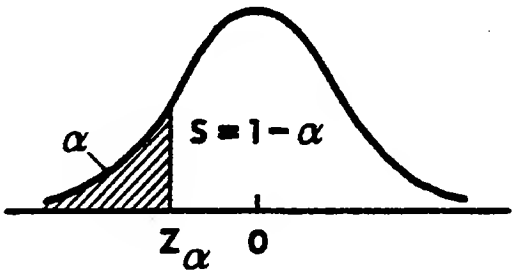

S(%)	90	95	99	99.9
$-z\,\alpha = z_{1.-}\,\alpha$	1.28	1.64	2.33	3.09

How to check normality?

Before using the characteristics of the ND the validity of the model has to be checked.

Method: Draw the cumulative frequency curve in the normal probability paper (Appendix II). Normality can be assumed if the resulting curve is approximately a straight line between 5% and 95%.

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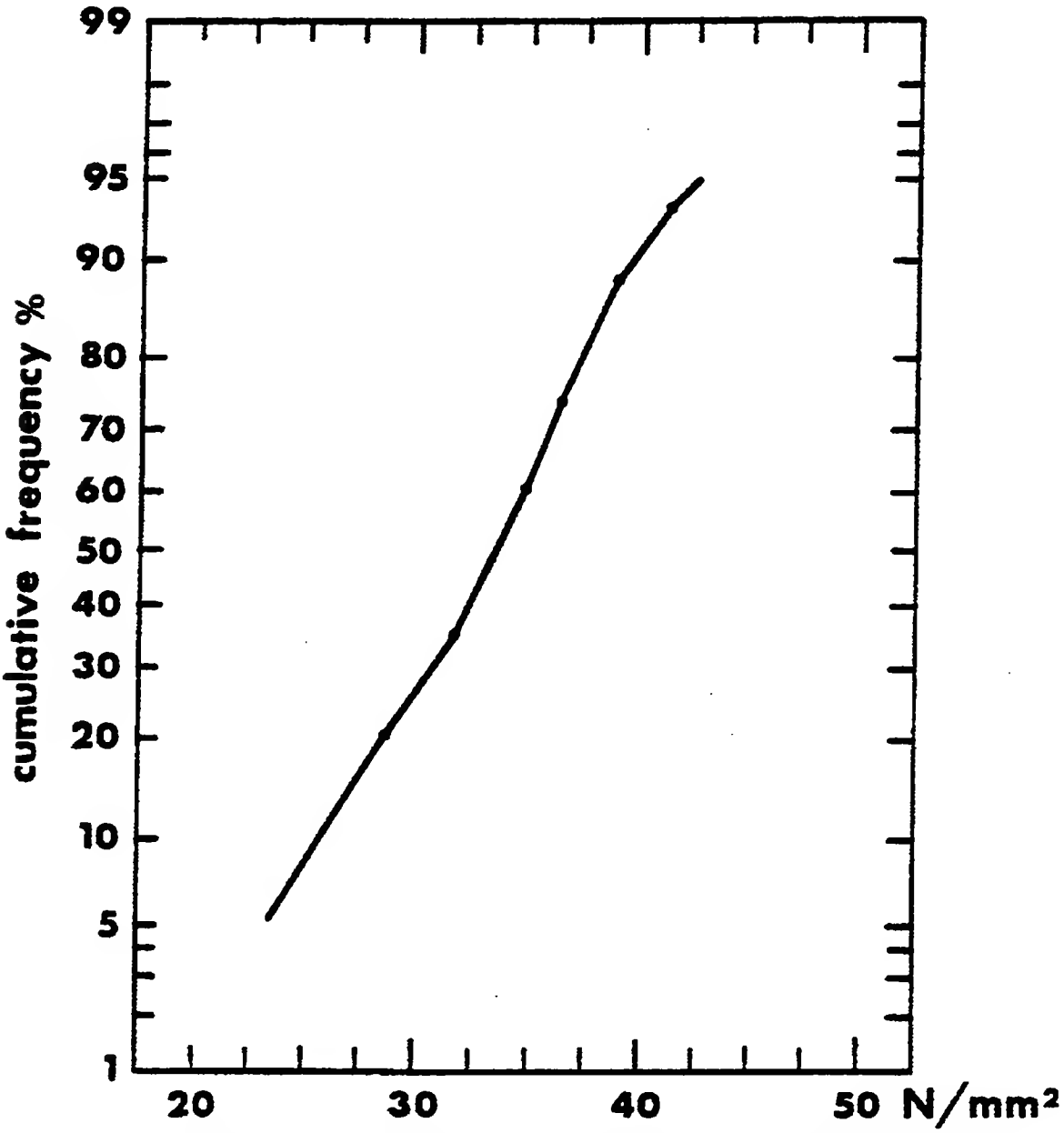
Example: Normal probability plot of compressive strength data



Conclusion: The observations of compressive strength follow approximately a normal distribution.

Short cut rule for rejecting normality: If no negative values are allowed in the observations and the coefficient of variation is greater than 30 %, the distribution is not normal.

4. CONFIDENCE LIMITS

4.1 Confidence limits for the mean

The arithmetic mean is an estimate for the true, unknown mean of the population of all possible observations. We may be interested in the precision of this estimation. Therefore we calculate a confidence interval that contains the true value with high probability (confidence level).

a) If the distribution is normal:

Two sided confidence interval at confidence level 1-α

$$\bar{x} \pm t_{(1-\alpha/2;f)} \cdot \frac{s}{\sqrt{n}}$$

with

f = n-1 degrees of freedom

$t_{1-\alpha/2;f}$ given in Table A 4, Appendix III (Student t-Distribution)

Interpretation:

With probability 1-α the true mean μ lies between the two confidence limits.

For n>50 $t_{1-\alpha/2;f}$ may be replaced by $z_{1-\alpha/2}$ given on page 15 (corresponding to normal deviates of Table A 1, Appendix III). In our example of compressive strength we calculate the 95% confidence interval by replacing the t- by the z- value:

$$\bar{x} \pm t_{0.975;89} \cdot \frac{s}{\sqrt{n}} \cong \bar{x} \pm z_{0.975} \frac{s}{\sqrt{n}}$$

$$\rightarrow 33.2 \pm 1.96 \frac{5.6}{90} = 33.2 \pm 1.16$$

b) If the distribution is not normal:

Approximate confidence intervals can be obtained by

$$\bar{x} \pm z_{1-\alpha/2} \cdot \frac{s}{\sqrt{n}}$$

This approximation is derived from the central limit theorem and can be used if

$$n > \frac{10 s^2}{\bar{x}^2}$$

c) Range method:

In the case of a normal distribution, confidence limits may be calculated by use of the range instead of the standard deviation (often used in quality control for small sample sizes n).

$$\bar{x} \pm \lambda_{1-\alpha/2} R$$

Values for are given in Table A 6, Appendix III.

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4.2 Confidence limits for the median

Confidence limits for the median can be determined directly from the ordered array of observations.

$$x_{(1)} \le x_{(1)} \le .... \le x_{(n-1)} \le x_{(n)}$$

(1-α)-confidence interval:

$$x_{(k)} < \text{Median} < x_{(n-k+1)}$$

with $k = \frac{n-1}{2} - \frac{z_{1-\alpha/2}}{2}\sqrt{n-1}$ rounded to the next lower integer value

$z_{1-\alpha/2}$ is given in Table A 1, Appendix III.

4.3 Confidence limits for the standard deviation

If the population follows a normal distribution, two sided confidence limits are given by

$$s\frac{f}{\chi^2_{1-\alpha/2;f}} \le \sigma \le s\frac{f}{\chi^2_{\alpha/2;f}}$$

with f = n 1 degrees of freedom

$\chi^2_{1-\alpha/2;f}, \chi^2_{\alpha/2;f}$ critical values of the chi-square distribution given in Table A 5, Appendix III

Confidence intervals are reduced with increasing sample size n, i.e. the more observations available, the better is the estimation.

The degree of improvement for the arithmetic mean can be derived from the fundamental central limit theorem. If several samples of size n are drawn from the same population, the arithmetic means of these samples are approximately normal with mean μ and standard deviation s/√n, the so called standard error of the mean (SEM).

For increasing n to infinity, the standard error of the mean tends to zero, i.e. the estimation tends to be absolutely precise if no systematic errors are present.

4.4 Other methods for the construction of confidence limits

Sometimes it happens that we have to deal with very complicated functions of random variables, for which we can't derive or know the underlying distribution function, but we would like to have confidence limits for the values of this complicated function. A newer method, the Bootshap method (B. Efron, 1983), allows to obtain such results, but is rather intensive in calculation.

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5. STANDARD TESTS

5.1 General Test Idea

Tests are used to make decisions in the case of incomplete information. We want to know if some observed difference is significant or only an effect of random errors.

Significant:

We decide that a difference really exists. The probability of this decision to be false is known and can be chosen by the decision maker. A usual choice of this error probability is 5%.

Not significant:

The observed difference may be realized by chance alone. The data give no argument to suppose the existence of a real difference. If such a difference really exists the sample size n is too small to detect it.

Example 3:

At a cement plant A, 618 titration samples gave the mean value $\bar{x}_1 = 82.67$ and the calculated standard deviation $s_1 = 4.13$.

At a later stage a further series of 525 samples were taken and after processing, yielded the information of calculated mean value $x_2 = 85.58$ and calculated standard deviation $s_2 = 3.79$

Your decision problem is:

State whether a significant change has occurred under plant conditions!

Example 4:

The production rate of a cement mill in tons/hr was measured as:

28.3 27.2 29.3 26.7 29.9 24.6 25.0 30.0 26.3 27.8

After modification, the production rate of the same mill in tons/hr was measured as:

28.0 30.0 30.5 26.0 31.0 30.3 24.6 25.4 26.7 29.3

Your decision problem is:

Has the modification made a significant improvement?

There are many problems in which we are interested in whether the mean (or another parameter value) exceeds a given number, is less than a given number, falls into a certain interval, etc.

Instead of estimating exactly the value of the mean (or another parameter), we thus want to decide whether a statement concerning the mean (or other parameter value) is true or false, i.e. we want to test a hypothesis H_0 about the mean (or another value).

In example 3 the hypothesis H_0 could be
"No significant change has occurred in the plant conditions"

In example 4 the hypothesis H_0 could be
"No significant improvement has been made by the modification"

We will solve the two given decision problems later in section 5.2 (test procedures).

Now we will consider another specific example introducing at the same time the important parts of all similar decision problems.

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Example 5:

In the manufacture of safety razor blades the width is obviously important. Some variation in dimension must be expected due to a large number of s mall causes affecting the production process. But even so the average width should meet a certain specification. Suppose that the production process for a particular brand of razor blades has been geared to produce a mean width of 0.700 inches. Production has been underway for some time since the cutting and honing machines were set for the last time, and the production manager wishes to know whether the mean width is still 0.700 inches, as intended.

We call set of all blades coming from the production line in a certain time interval (t, t+h) the statistical population to be studied. For example t = January 5th, 0^{oo} and t+h = January 6th, 0^{oo}, if the statistical population we are interested in is the set of all blades produced on January 5th.

If the production process was initially set up on that day to give a mean width of 0.700 inches, we can say that the hypothesis H_0 "the produced mean width of the regarded population is 0.700" should be tested. In symbols this is: $\mu_0 = 0.700$ = hypothesized mean.

Accepting the Hypothesis H_0:

Suppose we draw a simple random sample of 100 blades from the production line. We measure each of these carefully and find the mean width of the sample to be 0.7005 inches. The standard deviation in the sample turns out to be 0.010 inches. That is,

n = 100

$\bar{x}$ = 0.7005 inches

s = 0.010 inches

For the hypothesis $\mu_0 = 0.700$ to be true, the sample mean $\bar{x}$ = 0.7005 inches would have to be drawn from the sampling distribution of all possible sample means whose overall mean is 0.700 inches.

Now the important question arises: If the true mean of the population really were 0.700 inches, how likely is it that we would draw a random sample of 100 blades and find their mean width to be as far away as 0.7005 inches or farther? In other words, what is the probability that a value could differ by 0.0005 inches or more from the population mean by chance alone?

If this is a high probability, we can accept the hypothesis that true mean is 0.700 inches, because it is very easy to get it (high probability).

If the probability is low, however, the truth of the hypothesis becomes questionable because the sample we got is in reality very seldom.

To get at this question, compute the standard error of the mean from the sample:

$$s_{\bar{x}} = \frac{s}{\sqrt{n}} = \frac{0.010}{\sqrt{100}} = 0.001 inches$$

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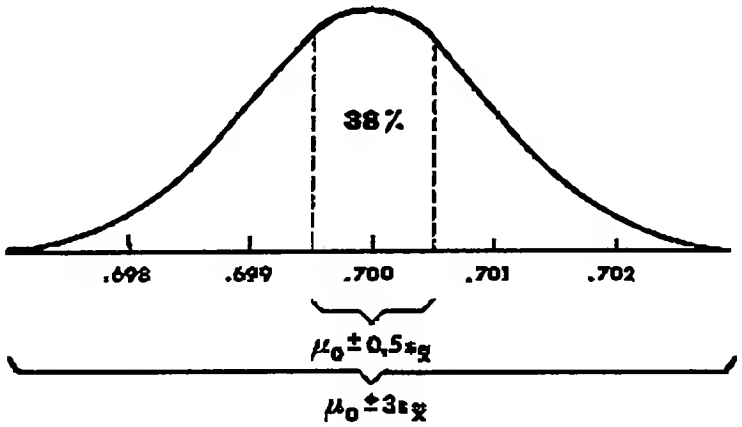
Since the difference between the hypothetical mean and the observed sample mean is 0.0005 inches and the standard error of the mean is 0.001 inches, the difference equals 0.5 standard errors. By consulting Table A-1 ($z_p = 0.5$)* we find that the area within this interval around the mean of a normal curve is 38%, so that 100 - 38 = 62 % of the total area falls outside this interval (cf. dashed lines below).If 0.700 inches were the true mean, therefore, we should nevertheless expect to find that about 62% of all such possible means would, by chance alone, fall as far away as 0.5 s_x or farther.

Therefore, the probability is 62% that our particular sample mean could fall this far away. This is a substantial reason to accept the hypothesis and attribute to mere chance the appearance of a 0.7005 inches mean in a single random sample of 100 blades.

Naturally we reject in the same time the contrary of H_0 namely that:

$$\mu_0 : \mu \neq 0.700$$



In case H_0 can not be rejected, avoid saying "it is proved that H_0 is correct". or "there are no differences in the means", say "there is no evidence that H0 is not true" or "there is no evidence H_0 should be rejected".

Rejecting the Hypothesis H_0:

Later, after production has gone on for some time, the query again arises:

Is it reasonable to believe that the true mean width of blades produced remains 0.700 inches? Since the process was adjusted to yield that figure the hypothesis still seems reasonable. We could then test it by taking another random sample of 100 blades.

This time the standard deviation is still 0.010 inches, so the standard error of the mean is still 0.001 inches, but the mean is now 0.703 inches:

In order to test the hypothesis that the true mean of the population is 0.700 inches, we again go through the same line of reasoning. If the true population mean really were 0.700 inches, how likely is it that we should draw a random sample of 100 blades and find their sample mean to be as far away as 0.703 inches?

Since the difference between the hypothetical mean of 0.700 inches and the actual sample mean of 0.703 inches is 0.003 inches, and the standard error of the mean is 0.001 inches, the difference is equal to three standard errors of the mean (i.e. 0.003/0.001 = 3).

* $z_p = 0.5$ p = 0.69 = 69%. Therefore 31% fall outside to the right. The same portional fall outside to the left of $-z_p$

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Now if 0.700 inches really were the population mean, we know from Table A-1, Appendix III that 99.7% of all possible sample means, for random samples of 100, would fall within three standard errors around 0.700 inches. Hence, the probability is only 0.3% that we would get a sample mean falling as far away as ours does.

We have two choices:

1) We may continue to accept the hypothesis (i.e. leave the production process alone), and attribute the deviation of the sample mean to chance.
2) We may reject the hypothesis as being inconsistent with the evidence found in the sample (hence, correct the production process).

Either of two things is true and we have to make a decision between them:

1) the hypothesis is correct, and an exceedingly unlikely event has occurred by chance alone (one which would be expected to happen only 3 out of 1000 times); or
2) the hypothesis is wrong

Type I and Type II Errors

Understandably, the question can be raised: What critical value should we select for the probability of getting the observed difference $(x - \mu_0)$ by chance, above which we should accept the hypothesis H0 and below which we should reject it? This value is called the

critical probability or level of significance

The answer to this question is not simple, but to explore it will throw further light on the nature and logic of statistical decision making. Let's study the following example:

in H0 is true H0 is false

In reality We decide	H_0 is true	H_0 is false
Accept H_0	3 right decision accept a true hypothesis	2 error II accept a false hypothesis
Reject H_0	1 error I reject a true hypothesis	4 error II accept a false hypothesis

Example:

H_0: the parliament building burns

Decision maker is the commander of the fire brigade.

Another expression for error I is: error of first kind

Another expression for error II is: error of second kind

If we ask here, what is worse

- error I or error II,

then error I naturally costs a lot of money because the fire will destroy the whole parliament building. Error II only moves the fire-brigade.

In a long run of cases which the hypothesis is in fact true (although we do not know it is true, for otherwise there would be no need to test it), we will necessarily either be wrong as in 1 or right as in 3.

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That is to say, if we make an error it will be of Type I.

Suppose we should adopt 5% as the critical probability. accepting the hypothesis when the probability of getting the observed difference by chance exceeds 5% and rejecting the hypothesis when this probability proves to be less than 5%. This amounts to the decision to accept the hypothesis when the discrepancy of the sample mean is less than 1.96 standard deviations, and to reject the hypothesis when the discrepancy is more than 1.96 standard deviations.

If we take 1% instead of 5%, as above, we will get as limit 2.58 standard deviations.

In fact, the percentage of cases in which we would expect to make an error of the first kind is precisely equal to the critical probability adopted.

(The probability of error I will be abbreviated quite often by α).

Just significant probability level:

In many studies the critical probability is used to describe the statistical significance of a sample result. For example, an economist collects some data on, say, interest rates and the demand for money. He hypothesizes some relationship and wishes to see if the data support his thesis. He tests the hypothesis to rule out the alternative that the observed relationship occurred by pure chance. He reports his sample results as "significant at the 1 percent level". Such a statement is a report to the reader that has the following meaning:

1) if we were to set up a statistical hypothesis
2) if we were to test this hypothesis using a critical probability of 1%
3) then we would reject the hypothesis and rule out a chance relationship

Significance levels of 10%, 5%, 1%, 0.1% are often used in reporting sample data. The smallest of these probability values is chosen at which the hypothesis can be rejected.

So we see now what is basic for every statistical test:

1) we need a clear hypothesis H_0
2) we have to know by which statistic we want to test H_0 (in our example it was $\bar{x}$)
3) we need a criterion C for decision making in the following form:
   reject H_0 if C applies
   accept H_0 if C does not apply

The criterion C is usually given in form of a critical limit, called significance limit, which should not be exceeded by the calculated test statistic.

To perform such a statistical test, it is necessary to select the risk to commit a type I error, i.e. the significance level α (see section 5.2, p. 28).

Often a relevant difference has to be detected with a certain probability, i.e. with a predetermined type II error β. This is only possible with a certain sample size n, as is further outlined in section 5.3.

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If we reject a hypothesis when it should be accepted, we say that a Type I error has been made. If, on the other hand, we accept a hypothesis when it should be rejected, we say that a Type II error has been made. In either case a wrong decision or error in judgement has occurred.

In order for any tests of hypotheses or rules of decision to be good, they must be designed so as to minimize errors of decision. This is not a simple matter since, for a given sample size, an attempt to decrease one type of error increases the other type. In practice one type of error may be more serious than the other, and so a compromise should be reached in favour of a limitation of the more serious error. The only way to reduce both types of error is to increase the sample size, which may or may not be possible.

Note that from the philosophy of testing there is always the possibility to reject H_0 although H_0 effectively is true (the probability for this is α, the type I error probability). So if you are testing say 100 "correct" datasets (i.e. for which H_0 holds) to a significance level $\alpha = 5\%$ you will expect about five results that reject H_0, although it holds. This has to be considered if a lot of (statistical) tests are made on the same data material (see e.g. Multiple Testing ..... ."Simultaneous Statistical Inference", Miller, 1981).

One-sided and two-sided tests

In example 5 we are interested in values of significantly higher or smaller than μ_0. Any such test which takes account of departures from the null hypothesis H_0 in both directions is called a two-sided test (or two-tailed test $H_0 : \mu = \mu_0, A : \mu \neq \mu_0$)

However, other situations exist in which departures form H_0 in only one direction is of interest. In example 4 we are interested only in an improvement of production rate due to the modification of the cement mill and so a one-sided test is appropriate. It is important that the decision maker should decide if a one-tailed or two-tailed test is required before the observations are taken.

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5.2 Test Procedures

Performance of a statistical test requires in advance the recognition and formulation of the present decision problem (test situation). Use the following checklist for applications of statistical tests:

CHECKLIST

1) Is the decision problem concerned with the mean. median. standard deviation. correlation coefficient or other?
   Formulate a clear hypothesis H_0.
2) Is it a one-sample or a two-sample problem?
   One-sample problem:
   Mean, median or standard deviation of a given sample of size n is compared with a hypothetical value (example 5).
   Two-sample problem:
   The decision problem is concerned with differences between two given samples.
3) For the two-sample problem: Are the observations independent or paired? Observations are paired if every value in the first sample can be attached definitely to a value in the other sample. Paired comparisons are in general considerably more efficient than a comparison of independent samples.
4) Is the sample drawn from a normal distribution?
5) Decide between one-sided or two-sided test.
   One-sided tests are used if differences only in one direction may occur or are of interest.
   Two-sided tests are used if no preliminary information is available, in what direction the sample may differ.
6) Choose the test procedure in the table "TEST CONCERNED WITH" (sample sizes $n<50$ are considered to be small).
7) Choose the significance level a. Usual choices are $\alpha = 5\%$ or $\alpha = 1\%$ (error of type I).
8) Calculate the test statistic T of the chosen test (cf. the formulae of the following pages).
9) Look for the significance limit T_p for the test statistic in the corresponding table with $p = 1-\alpha/2$ (two-sided test) or $p = 1-\alpha$ (one-sided test).
10) Decision: If the calculated test statistic exceeds the significance limit, reject the hypothesis H_0 (the observed difference is significant at the level α). Otherwise there is no reason to reject the hypothesis H_0 and the difference is considered to be not significant.

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TEST CONCERNED WITH

test situation	median	mean general	normal distribution	mean standard deviation
One-sample problem				
small n (n<50)	sign test	signed-rank-test	t-test	x^2-test
large n (n≥50)	sign test	z-test	z-test	x^2-test
Two-sample problem independent				
small n	median test	Wilcoxon-test	t-test	F-test
large n	median test	z-test	z-test	F-test
paired				
small n	sign test	signed-rank-test	t-test	-
large n	sign test	z-test	z-test	-
More than two samples	x^2-test	Krukskal- Wallis-test	Analysis of variance (ANOVA) Friedmann-test	Bartlett-test
(These tests are not treated i n this paper)				

Selected test statistics

5.2.1 z-Test

Used to test means with large sample size n.

a) One-sample-problem:
Given is a sample of size n with arithmetic mean $\bar{x}$ and standard deviation s, we test the hypothesis that the mean (estimated by $\bar{x}$) is equal to a given or target value μ_0

Hypothesis H_0 $\mu = \mu_0$

Test statistic $$z = \frac{(\bar{x} - \mu_0)\sqrt{n}}{\sigma}$$

In general σ is not known. It can be replaced by s for large sample sizes.

b) Two independent samples:
Given are two samples of size n_1 and n_2 with arithmetic means $\bar{x}_1, \bar{x}_2$ and standard deviations s_1, s_2.

Hypothesis H_0 $\mu_1 = \mu_2$

Test statistic $$z = \frac{\bar{x}_1 - \bar{x}_2}{\sqrt{\frac{s_1^2}{n_1} + \frac{s_2^2}{n_2}}}$$

c) Paired comparison
Given is a sample of n paired observations x_i, y_j. Calculate the arithmetic mean $\bar{d}$ of all the differences $d_i = y_i - x_i$ and the standard deviation s_d.

Hypothesis H_0 $\mu_x = \mu_y$ respectively $\mu_d = 0$

Test statistic $$z = \frac{\bar{d}\sqrt{n}}{s_d}$$

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Significance limits

Often used significance limits for $z_{1-\alpha}$ (one-sided test) and $z_{1-\alpha/2}$ (two-sided test) are

$z_{0\,95} = 1.645$ $z_{0.975} = 1.960$

$z_{0\,99} = 2.326$ $z_{0\,995} = 2.576$

For other significance levels α look for $z_{1-\alpha}$ in Table A-1.

Decision

The difference is significant if

$$|z| \geq z_{1-\alpha/2} \quad \text{two sided test}$$

$$\left.\begin{array}{l} |z| \geq z_{1-\alpha} \\ |z| \leq -z_{1-\alpha} \end{array}\right\} \quad \text{one-sided test}$$

Example

a) In the example 1 we test the two-sided hypothesis H_0:
$\mu = 35.0\ N/mm^2 = \mu_0$ (one-sample problem).
$\bar{x} = 33.2$
$s = 5.6$
$n = 90$
$\alpha = 0.05$

$$z = \frac{(33.2 - 35.0)\sqrt{90}}{5.6} = -3.05$$

|z| = 3.05 ' 1.96 = Z0.975
Decision: The sample mean differs significantly from the standard strength 35.0 N/mm^2.

b) in example 3 (chapter 5.1) we test the hypothesis whether a change in plant conditions has occurred or not (two independent samples):
H_0: $\mu_1 = \mu_2$
$n_1 = 618$, $n_2 = 525$
$x_1 = 82.67$, $x_2 = 85.58$
$s_1 = 4.13$, $s_2 = 3.79$
$\alpha = 0.05$ (two-sided test)
85.58-82.67

$$z = \frac{85.58 - 82.67}{\sqrt{\frac{(4.13)^2}{618} + \frac{(3.79)^2}{525}}}$$

$$z = 12.41 > 1.96 = z_{0.975}$$

Decision: Highly significant difference between the two samples of titration values.

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5.2.2 Sign Test

Test for the median in one-sample problems and paired comparisons.

a) One-sample problem: Given is a sample of size n.
We test the hypothesis that the median of the population is equal to a given value m_0
H_0: median = m_0
Test statistic:
Count the number of observations smaller than m_0 and those larger than m_0. The test statistic k is then the smaller of the two numbers.

b) Paired comparison: Given are n paired observations x_i, y_j.
We test the hypothesis that x_i and y_j have the same median.
Test statistic:
For each pair of observations x_i, y_j record the sign of the difference $y_i - x_i$.
The test statistic k is then the number of occurrences of the less frequent sign.

Significance limits

Calculate $k_{\alpha/2} = \frac{n-1}{2} - \frac{z_{1-\alpha/2}}{2}\sqrt{n-1}$ (two-sided test).

For the one-sided test replace $\alpha/2$ by α. Z_p is the significance limit of the z-test

$$z_{0.95} = 1.645 \qquad z_{0.975} = 1.960$$

Decision

If k is less than $k_{\alpha/2}$ (resp. k_α) conclude that the medians are different, otherwise, there is no reason to believe that the medians differ. 44

Example

Test the hypothesis that 50% of the population has an income of more than 3000 (example 2).

H_0: $m_0 = 3000$
k = 217
n = 479
α = 0.05 two-sided test

$$k_{0.025} = \frac{478}{2} - \frac{1.96}{2}\sqrt{478} = 217.6$$

The test is just significant at the 5% level. Because the sample median is 2700, we conclude that less than 50% of the population has an income of 3000.

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5.2.3 Signed-rank test

Test for the mean in symmetrical distributions with small sample sizes n (one-sample or paired comparison).

a) One-sample problem: Given is a sample of size n from a symmetrical distribution.
Hypothesis $H_0 : \mu = \mu_0$

Disregarding signs, rank the d_i according to their numerical value, i.e., assign rank 1 to the smallest observation, assign the rank of 2 to the d_i which is next smallest, etc. In case of ties, assign the average of the ranks which would have been assigned if the d_i's had differed only slightly. (If more than 20% of the observations are involved in ties, this procedure should not be used).

To the assigned ranks 1, 2, 3, etc., prefix a + or a - sign, according to whether the corresponding d_i is positive or negative. The test statistic T is then the sum of these signed ranks.

a) Paired comparisons: Given are n paired observations x_i, y_i.
The hypothesis is tested that both have the same mean.
Test statistic: Compute $d_i = y_i - x_i$ for each pair of observation and continue as in a).

Significance limits

Look for $T_{0.95}$ or $T_{0.975}$ in Table A-7. If the number m of differences d_i exceeds 20 perform a z-test with

$$z_T = \frac{T}{\sqrt{\frac{m(m+1)(2m+1)}{6}}}$$

Decision

Conclude that the means differ if

$\lvert T \rvert \geq T_{1-\alpha/2}$	$(or \lvert z_T \rvert \geq z_{1-\alpha/2})$	two-sided test
$T \geq T_{1-\alpha}$	$(or z_T \geq z_{1-\alpha})$	one-sided test
$T < -T_{1-\alpha}$	$(or z_T < -z_{1-\alpha})$	one-sided test

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5.2.4 Wilcoxon-Test

Test for a comparison of means in two independent samples:

Given are two independent samples of size n_1 and n_2.

We test the hypothesis that both samples have the same mean

$$H_0: \mu_1 = \mu_2$$

Test statistic:
Combine the observations from the two samples, and rank them in order of increasing size from smallest to largest. Assign the rank of 1 to the lowest, a rank of 2 to the next lowest, etc. (Use algebraic size, i.e., the lowest rank is assigned to the largest negative number, if there are negative numbers). In case of ties, assign to each the average of the ranks which would have been assigned if the tied observations had differed only slightly. (If more than 20% of the observations are involved in ties, this procedure should not be used).

Let n_1 = smaller sample
n_2 = larger sample
$n = n_1 + n_2$

Compute R, the sum of the ranks for the smaller sample. (If the two samples are equal in size, use the sum of the ranks for either sample).

Compute $W = 2R - n_1(n + 1)$

Significance limits

Look for $W_{0.95}$ or $W_{0.975}$ in Table A-8. For sample sizes not mentioned in the table perform a z-test with

$$Z_W = \frac{W}{\sqrt{\frac{n_1 n_2 (n+1)}{3}}}$$

Decision

Conclude that the means differ if

$|W| \geq W_{1-\alpha/2}$ $(or |z_w| \geq z_{1-\alpha/2})$ two-sided

$W \geq W_{1-\alpha}$ $(or z_w \geq z_{1-\alpha})$ one-sided

$W < -W_{1-\alpha}$ $(or z_w < -z_{1-\alpha})$ one-sided

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Example

In example 4 (chapter 5.1) we are interested in an improvement of production rate after a modification of the cement mill.

$\alpha = 0.05$ (one-sided test)

Combined sample (values in italics correspond to the sample after modification):

tons/hr	24.6	*24.6*	*25.0*	25.4	26.0	*26.3*	26.7	26.7	*27.2*	*27.8*
rank	1.5	*1.5*	*3*	4	5	*6*	7.5	7.5	*9*	*10*

tons/hr	28.0	*28.3*	29.3	*29.3*	*29.9*	30.0	30.0	30.3	30.5	31.0
rank	11	*12*	13	*14*	*15*	16.5	16.5	18	19	20

The sum R of ranks in the second sample is
$R = 1.5 + 4 + 5 + 7.5 + 11 + 13 + 16.5 + 18 + 19 + 20 = 115.5$
$W = 2 \cdot 115.5 - 10 \cdot 21 = 21$
$W = 21 < 46 = W_{0.95}$

Decision

The improvement is not significant. We have no reason to assume a real improvement due to the modification.

5.2.5 t-Test

This test is used for comparisons of means in normal distribution with small sample sizes n. If the distribution is not known to be normal, prefer rank tests (Wilcoxon, signed ranks).

The test situation is the same as in 5.2.1 (z-test) refer for remarks on hypothesis and decision. Instead of $z_{1-\alpha}$ look for significance limits $t_{1-\alpha}$ in Table A-4, with n-1 degrees of freedom (df) in the one-sample case or for paired comparison, and with n1 + n2 - 2 degrees of freedom (df) for the two-sample case respectively.

One-sample problem: Given is a sample of size n drawn from a normal distribution.

$$t = \frac{(\bar{x} - \mu_0)\sqrt{n}}{s}$$

Two independent samples: Given are two independent samples of size n_1 and n_2 from two normal distributions with means μ_1, μ_2 and equal standard deviation σ.

$$t = \frac{\bar{x}_2 - \bar{x}_1}{\sqrt{\frac{(n_1-1)s_1^2 + (n_2-1)s_2^2}{(n_1+n_2-2)}}} \sqrt{\frac{n_1 n_2}{n_1+n_2}}$$

Paired comparison: Given are n pairs of observations x_i, y_i, when x and y follow a normal distribution.

$$t = \frac{\bar{d}}{s_d}\sqrt{n}$$

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5.2.6 Median-Tests

Median tests for two independent samples are not given here in an explicit form. For small sample sizes n Fisher's test for 2 x 2 contingency tables and for large n x^2-test for contingency tables may be used (Natrella 1963, Noether 1971).

In a comparison of two independent samples we are often not interested in differences only of means or medians, but generally in location differences of the two samples.

In this case use the tests for means, which are more efficient than median tests (Wilcoxon-test or z-test).

5.2.7 X^2-Test

Several X^2-tests are available for different test situations. The test we explain here is used to compare the standard deviation s of a sample with hypothetical value σ_0.

X^2-tests are also used for

- Goodness of fit test
- Independence test
- Loglinear models

(see e.g. Haberman, 1978)

Given is a sample of size n drawn from a normal distribution with mean μ and standard deviation σ. We test the hypothesis that the standard deviation σ, estimated by s from the sample is equal to a hypothetical standard deviation σ_0.

$$H_0: \sigma = \sigma_0 \quad \text{Test statistic: } x^2 = \frac{(n-1)s^2}{\sigma_0^2}$$

Significance limits

Look for significance limit $X^2_{p,m,}$ in Table A-6 with m = n-1 degrees of freedom (df).

Decision

If	$X^2 \geq X^2_{1-\alpha/2;m}$	or $X^2 \leq X^2_{\alpha/2;m}$	conclude: $\sigma = \sigma_0$	(two-sided)
If	$X^2 \geq X^2_{1-\alpha;m}$		conclude: $\sigma > \sigma_0$	(one-sided)
	$X^2 \leq X^2_{\alpha;m}$		conclude: $\sigma < \sigma_0$	(one-sided)

Otherwise we have no reason to believe that σ differs from σ_0.

5.2.8 F-Test

Comparison of the standard deviation in two independent samples.

Given are two independent samples of size n_1 and n_2 with standard deviations s_1 and s_2. Both samples are drawn from a normal distribution. We test the hypothesis that both populations have the same standard deviation.

$$H_0: \sigma_1 = \sigma_2$$

Test statistic:

Let be $s_1 > s_2$, then compute

$$F = \frac{s_1^2}{s_2^2}$$

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Significance limits

Look for $F_{1-\alpha;m1,m2}$ (one-sided) or $F_{1-\alpha/2;m1,m2}$ (two-sided) in Table A-9 with $m_1 = n_1-1$ and $m_2 = n_2-1$ degrees of freedom.

Decision

Conclude $\sigma_1 \neq \sigma_2$ if $F \geq F_{1-\alpha/2;m1,m2}$ (two-sided)

or $\sigma_1 > \sigma_2$ if $F \geq F_{1-\alpha;m1,m2}$ (one-sided)

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5.3 Sample Size Determination

As stated at the end of chapter 5.1, the probability β of making a type II error in a given test with significance level α (type I error) depends upon the sample size n.

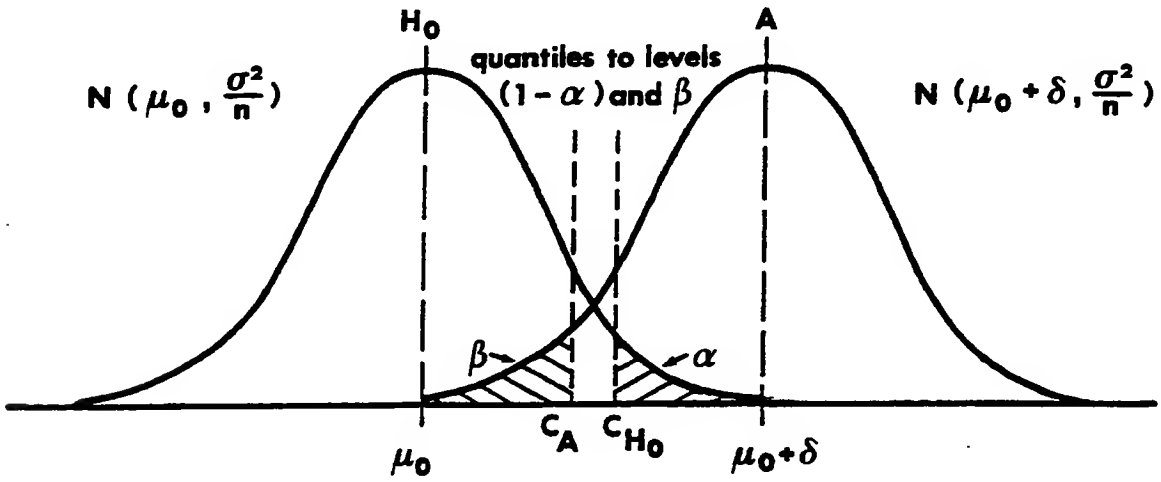
How can we determine the sample size that is necessary to hold the type II error within certain boundaries (probability β)? Or in other words: What sample size n is necessary to detect a relevant difference with great probability (1-β)?

Let us first examine a one-sided one-sample-test for testing a mean:

Suppose we are interested in the mean μ of a random variable X. We want to test the hypothesis H_0: $\mu = \mu_0$ against the alternative hypothesis A: $\mu > \mu_0$ with a z-test. The error-type-one shall be α.

The probability to detect a deviation of Δ from μ_0 shall be at least (1 - β), if this deviation exceeds a preselected relevant difference δ (β is now an upper bound for the type II error).

The following figure shows the distribution of the sample mean $\bar{x}$ under H_0 and under the (special) alternative hypothesis A: $\mu = \mu_0 + \delta$:



Interpretation of the graph:

- C_{Ho} is the criteria (significance limit) of the test:
  The hypothesis H_0 is rejected with probability α even if it is true.
- On the other hand, the probability to accept H_0: $\mu = \mu_0$, even if the true mean is $\mu = \mu_0 + \delta$ (type II error), is greater than β.
- The relevant difference δ may only be detected with probability (1-β), if $C_{Ho} \leq C_A$; in this case the risk to commit a type II error is ≤ β.

The variance $\frac{\sigma^2}{n}$ of the two distributions becomes smaller with increasing n and hence C_A moves to the right and C_{Ho} to the left. Therefore, we can find an n such that $C_{Ho} \leq C_A$:

$$C_{H_0} \leq C_A$$

$$z_{1-\alpha} \cdot \frac{\sigma}{\sqrt{n}} + \mu_0 \leq z_\beta \frac{\sigma}{\sqrt{n}} + \mu_0 + \delta = -z_{1-\beta} \frac{6}{\sqrt{n}} + \mu_0 + \delta \qquad (1)$$

$$\frac{\sigma}{\delta}(z_{1-\alpha}+z_{1-\beta}) \le \sqrt{n}$$

$$\text{or } n \ge \frac{\sigma^2}{\delta^2}(z_{1-\alpha}+z_{1-\beta})^2 \qquad (2)$$

n corresponds with the required sample size to detect a desired difference δ with probability $(1-\beta)$ by a z-test with significance level α.

For the two-sided one-sample-test we only have to replace α by $\alpha/2$, and we get:

$$n \ge \frac{\sigma^2}{\delta^2}(z_{1-\alpha/2}+z_{1-\beta})^2 \qquad (3)$$

Note: β is not replaced by $\beta/2$!

In a similar (but somewhat more complicated) way we find lower bounds for the sample sizes n_1 and n_2 satisfying our conditions in the one-sided two-sample-test:

$$n_2 \ge \frac{\sigma^2\sigma_1+\sigma_2^2}{\delta^2}(z_{1-\beta}+z_{1-\alpha})^2 \qquad (4)$$

$$n_1 \ge \frac{\sigma_1}{\sigma_2} n_2$$

For equal standard-deviations ($\sigma_1 = \sigma_2 = \sigma$) we obtain from (4):

$$n_1 = n_2 \ge \frac{2\sigma^2}{\delta^2}(z_{1-\beta}+z_{1-\alpha})^2 \qquad (5)$$

In the two-sided two-sample-test we simply have to replace α by $\alpha/2$ in (4) or (5).

The following table shows short rules for determining sample sizes when $\alpha = \beta = 5\%$ and $\sigma_1 = \sigma_2 = \sigma$ in the two-sample-case. The numbers in parentheses refer to the formulae from which the rules are derived.

	one sample	two samples
one-sided	$n \ge \frac{11\sigma^2}{\delta^2}$ (2)	$n_1 = n_2 \ge \frac{11\sigma^2}{\delta^2}$ (5; for $\sigma_1 \ne \sigma_2$ see (4))
two-sided	$n \ge \frac{13\sigma^2}{\delta^2}$ (3)	$n_1 = n_2 \ge \frac{26\sigma^2}{\delta^2}$ (5)

with δ = relevant difference to be detected with probability $(1-\beta)$

$\alpha = \beta = 5\%$

Note:

- The variance σ^2 is usually not known, but often some knowledge about σ^2 is available from former experiments (standard deviation). Otherwise σ^2 may be estimated in a pilot study.
- In the case of small sample sizes (t-test, Wilcoxon-test), add 5% to the calculated n.

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6. DATA PRESENTATION AND INTERPRETATION

(Chambers, Cleveland, Kleiner and Mikey, 1983).

The problem of data presentation and interpretation is common to cement manufacturers and users. In every stage of cement, aggregate, concrete production and application, data concerning materials, equipment, energy consumption, market situation, costs, etc. are generated. All this information must increase the knowledge about what happens in the process and market, thus providing the basis of decision. Moreover, communication between supplier and consumer has to rely on this information. We consider it, therefore, vital not only for optimum manufacture of cement and cement based products, but also with respect to the mutual relation between manufacturer and user that appropriate attention is given to data handling, i.e. presentation and evaluation.

We may differentiate between three levels of information:

- data needed for direct process and quality control (routine decision, off- or on-line)
- data required for day-to-day management decisions based on quality reports
- additional data allowing long-term improvements and developments.

With respect to presentation and interpretation of test results, the main problems for plant management are:

- how to organize the required information flow so that decision can be made at all levels of competence
- to reproduce and evaluate the corresponding data in a specific situation.

The relevant data of all domains cannot be made available without a functional information system. But availability alone does not necessarily result in a rational decision based on the data. To achieve this, two further aspects must be considered: first, the data should be presented in an intelligible form, i.e. a high transparency of the results enables the management to recognize certain relationships or critical results in time. Secondly, the decision maker must be aware of accuracy, significance and relevance of the considered data in order to provide a realistic interpretation.

Availability of data

A quick availability of data does not necessarily require a fully integrated data based system with electronic data processing, but is rather a matter of organization. Important data (e.g. for process and quality control) should circulate with little loss of time. Consequently, an appropriate reporting system (organization) must be established. Additional data should not disappear in some drawer where a later retrieval is impossible or at least very inconvenient. To avoid such a disorder, all data are recorded in a similar way, including a note on the circumstances of measurement and provenance of samples and test results. Remarks about circumstances and provenance are used to judge the comparability of different sets of observations. A standardised recording procedure makes data surveying easy and simplifies a later change to electronic data processing.

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6.1 Intelligible Presentation

The main aim of data presentation is transparency rather than secrecy, i.e. graphs should be employed instead of tables. A graphical presentation gives a quick survey on relevant information such as changes in time, extreme values, relationship between variables. A short survey on frequently used graphs can be found in section 6.4. It is recommended to produce the graphs directly at the source of the data in the form of tallies and/or control charts. A control chart may already be included in the laboratory journal and reports, directly behind the columns for sample identification and test results. The possibility of obtaining a quick survey by consulting a well presented graph will not only help the management in its decision making, but transparency will also improve interest and motivation of the personnel at any level of competence.

6.2 Interpretation

First of all. we must know how much we can rely on the data.

Test results are never "true" values, but are rather subject to errors of three types (see also section 1):

- random errors caused by sampling, imprecise measurement, environmental effects, etc.
- systematic errors caused by bias in sampling or process measurement, by the use of inadequate experimental design
- gross errors caused by recording wrong or not comparable values.

Usually, we expect 5 to 10% gross errors in a set of observations. Random errors may be denoted as reproducibility and expressed as the corresponding standard deviation. In general, these errors are underestimated. Reproducibility should be known for every important analyzing method. Systematic and gross errors are difficult to characterize. They should be minimized by careful experimentation and data handling. A periodic check for systematic errors should be done (calibration, comparison with a standard, inter-laboratory test).

How to compare several data sets or groups of data?

Before any comparison is made, we have to seriously check the comparability of data. Often, data sets differ in provenance of samples or circumstance of measurement, so that a comparison ma y be impossible. Even the fact that sample 1 was measured by laboratory assistant Miller and sample 2 by Brown will lead to a biased comparison if there is any relevant systematic error between the two persons. If the data sets are comparable and we observe a certain difference, we have to ask the following questions before taking any action:

a) Is the difference significant?

b) Is the difference relevant?

The problem of significance is answered by a statistical test. If it is not significant, we have no reason to take any action because the observed difference may occur by chance alone. If the test indicates a significant difference, it is not necessarily relevant for the problem we are concerned with. Of course, the decision whether or not it is relevant is not a statistical problem. Perhaps a decision maker is alarmed when an observed difference, considered to be relevant in the present problem, does not lead to a significant test result. In this case, the sample size used may be too small or the testing procedures may not be sufficiently sensitive to solve the given problem.

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Consequently, experience in product manufacture and application is necessary to assess the relevance of difference between target quality and experimental values. However, to judge whether it is significant requires a training in statistical technique, particularly test procedures. Decisions based simply on either practical experience of many years (relevance) or on statistical technique and procedures (significance) will lead to too frequent and unnecessary actions in both the manufacturing process and product application.

6.3 Data Interpretation related to Problems in Cement Application

If confronted with the problem of finding the reasons for poor quality of cement related products (concrete, asbestos cement, etc.), the cement is often suspected to be the cause. This can be explained by the fact that cement is the binding agent, and in the majority of cases constitutes the most expensive component, although it is known that the effect of other components, proportioning and curing conditions etc. are of great importance. A further reason to inspect the cement first is the difficulty to specify the other effects while cement is well defined by its chemical and mineralogical composition and its fineness.

To find out the causes for inferiority or changes in quality, a collaboration of consumer, producer and statistician is indispensable: the statistician may be omitted in uncomplicated problems if delegates of consumer or producer are well trained in statistics. In a first retrospective analysis of available data, parallel changes of parameters and quality are studied. This is done by drawing scatter diagrams and performing a regression analysis. The disadvantage of retrospective studies is the difficulty to find out causal relationships. The effects of several variables are mixed and cannot be separated due to their causal origin. On the other hand, an observed correlation indicates a possible causal effect. The decision about what cause is really responsible must be made by an experienced specialist and not by a statistical test. Often a decision is not possible because dependencies are too complex. In this case, a special experiment has to be planned and performed with a controlled variation of suspected variables and careful elimination of interfering effects (prospective analysis). In contrast to the retrospective analysis, an adequately designed experiment renders it possible to evaluate and judge causal effects with statistical methods. A short survey on the use and interpretation in regression/correlation analysis and designed experiments is given in chapter 7.

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Survey on Graphical Data Representation

Representation of one sample - documentation

Frequency table / Tally

Given is a sample of large size n. Observations are grouped into classes of equal length and marked in the tally.

Example: 90 values of concrete strength

mid-point	class	tally	absolute frequency	relative frequency
20.0	18.75 - 21.25	I	1	0.011
22.5	21.25 - 23.75	IIII	4	0.044
25.0	23.75 - 26.25	IIII I	6	0.067
27.5	26.25 - 28.75	IIII IIII	9	0.100
30.0	28.75 - 31.25	IIII IIII	10	0.111
32.5	31.25 - 33.75	IIII IIII IIII	19	0.212
35.0	33.75 - 36.25	IIII IIII IIII	17	0.189
37.5	36.25 - 38.75	IIII IIII I	11	0.122
40.0	38.75 - 41.25	IIII II	7	0.078
42.5	41.25 - 43.75	II	2	0.022
45.0	43.75 - 46.25	III	3	0.033
47.5	46.25 - 48.75		0	0.000
50.0	48.75 - 51.25	I	1	0.011
Total			90	1.000

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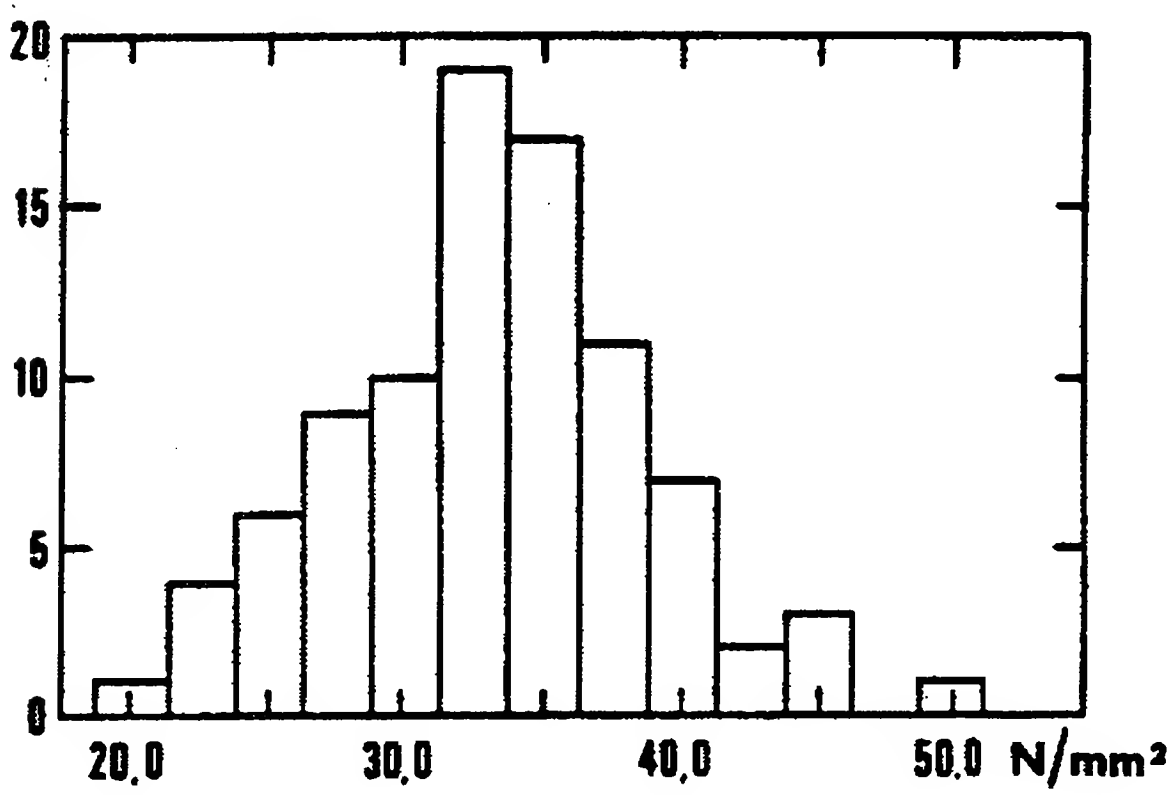
Histogram

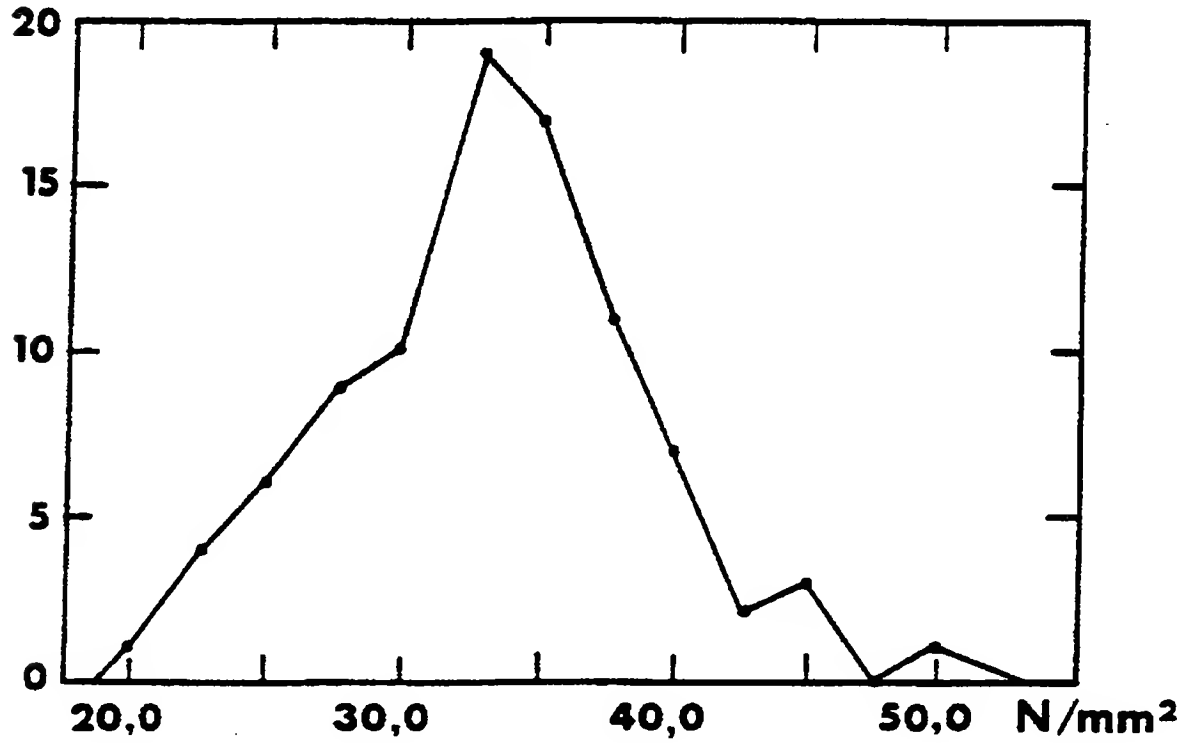
Graphical representation of the tally. Visualization of minimum, maximum, center and shape of the distribution.

Area of rectangles corresponds to absolute or relative frequencies in the classes.



Frequency curve

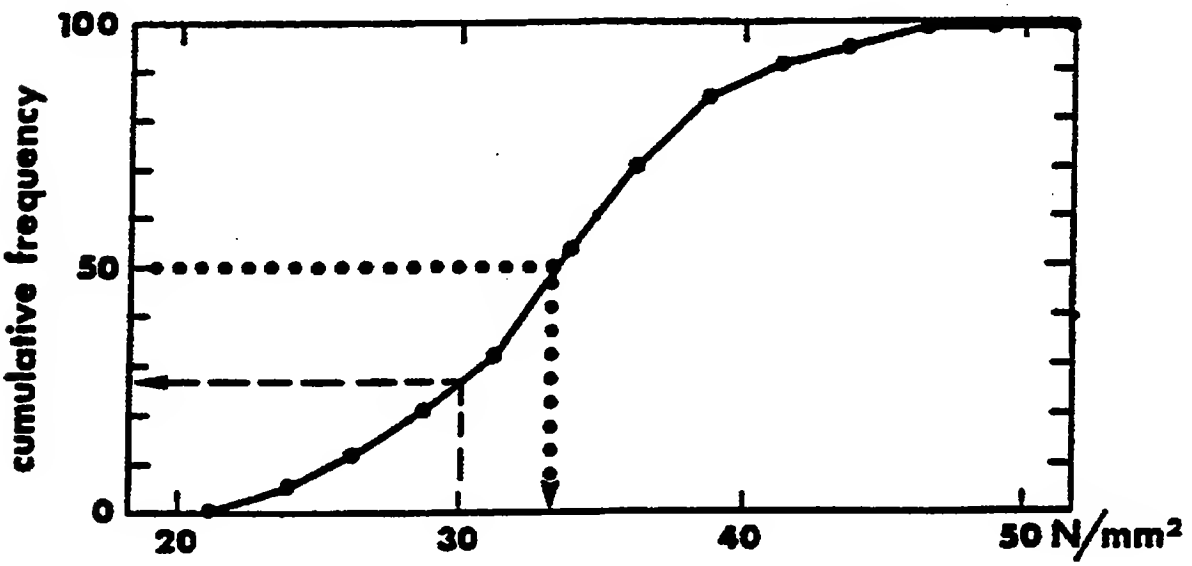
Relative (or absolute) frequencies plotted against class mid point. Every point on the curve corresponds to the relative (or absolute) frequency of observations falling in this class.



Cumulative frequency curve

Cumulated relative frequencies plotted against upper class boundaries. Every point on the curve corresponds to the portion of values which are smaller than any given strength x.

Example: Half of the measurements are smaller (respectively greater) than 33.5 N/mm^2



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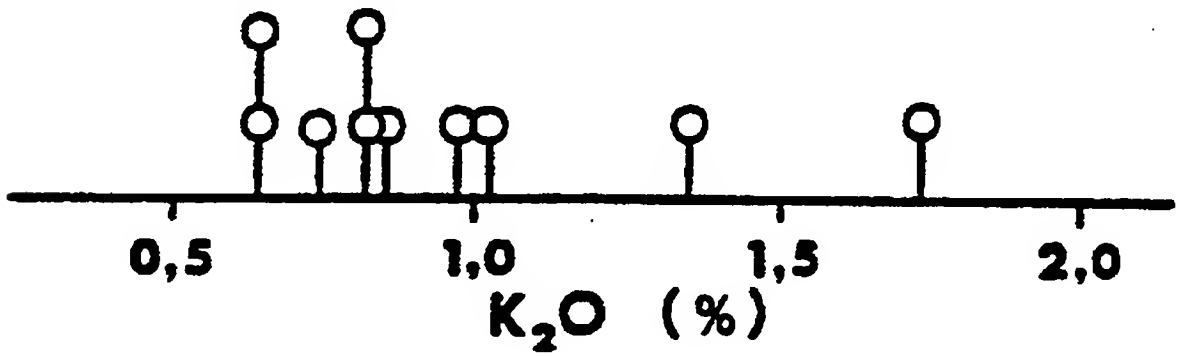
Stem-and-leaf Diagram

Similar to the histogram, but allows to see the individual data. Each observation is a leaf of a stem.

```
20    9
21
22    8
23    0579
24    456
25    0
26    1
27    126899
28    455
29H   039
30    0122378
31    456778
32    0244888
33M   033456
34    124569
35    2233455889
36H   14688
37    14689
38    169
39    247
40    12
41    09
42
43    4
44    3
45    38
      ***OUTSIDE VALUES***
48    9
```

Representation of small samples

For small sample sizes n the individual values are plotted directly on the measurement scale. Suitable to detect outliers and skewed distribution.



Example: K_2O-content of 10 cements

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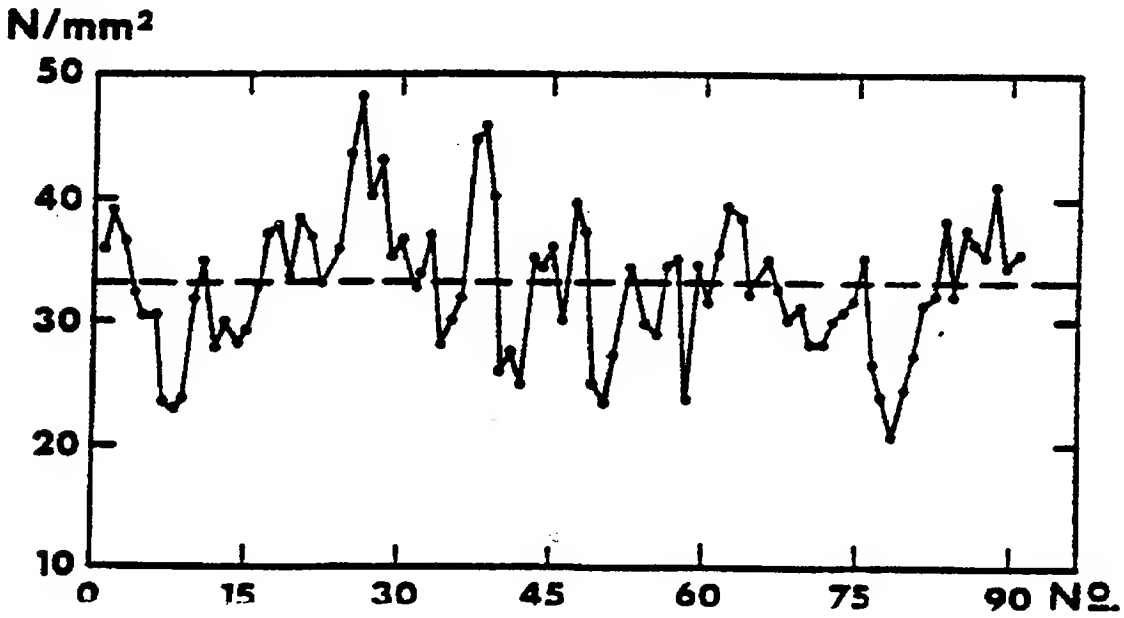
Time-plot

Often it is appropriate to draw the observations in chronological order to show a possible change of the level in time.

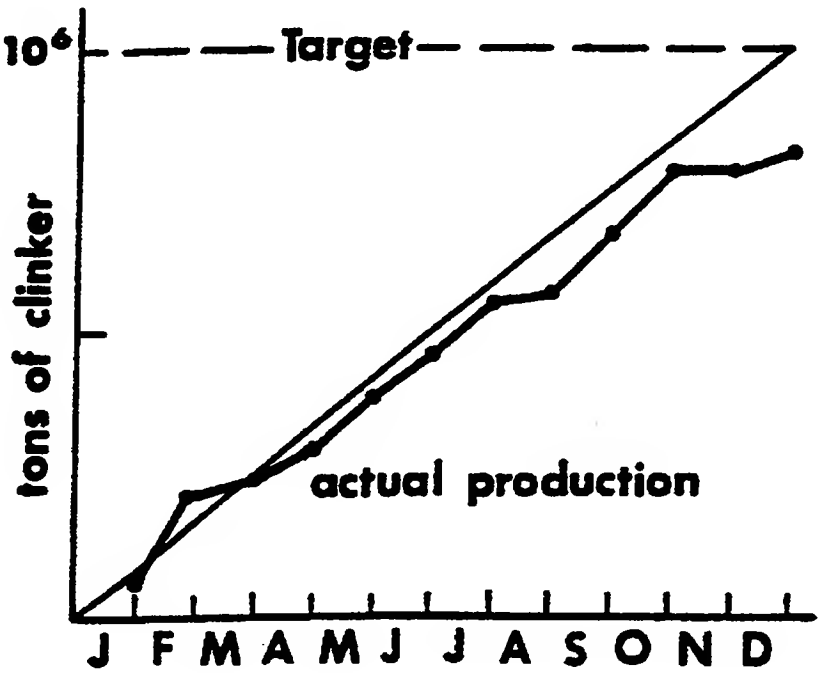
Useful in annual reports.



Cumulative time-plot

Cumulated values are plotted against time. Used to show deviations from a cumulative target.

Example: Actual clinker production is cumulated every month and compared with a target.



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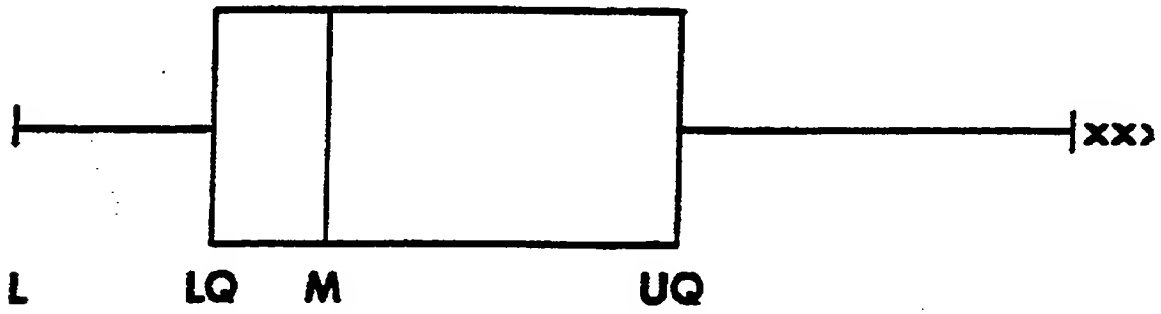
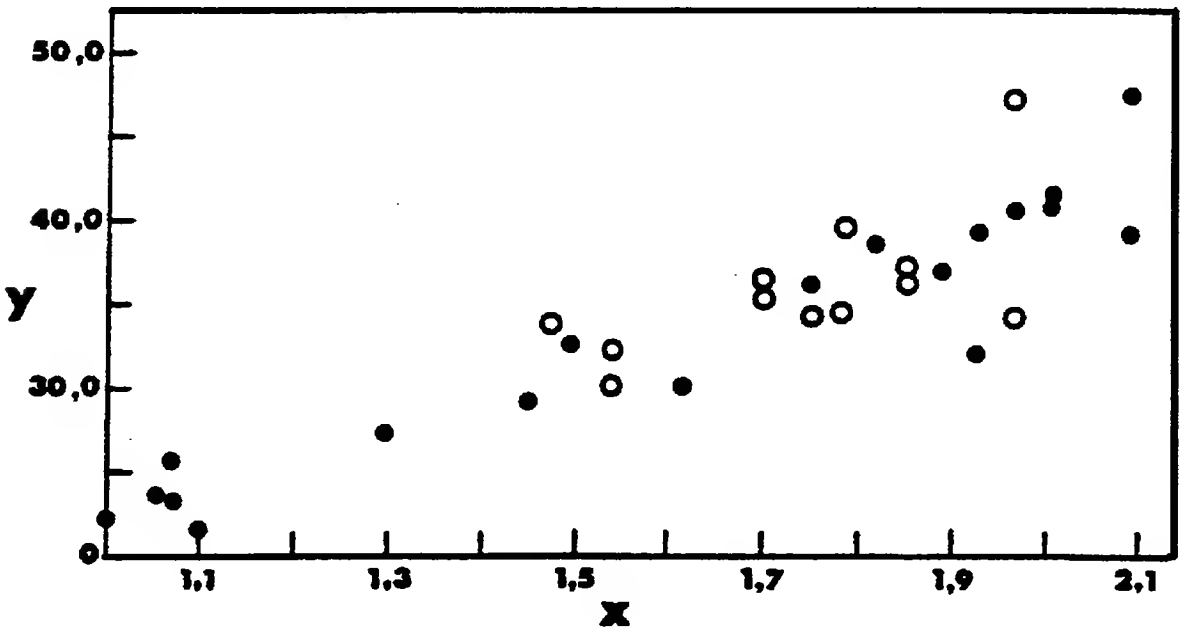
Box-Plot (Box-and Whisker-Plot)

Special and frequently used plot to numerize data. Shows median, lower and upper quartile (forming the box) and two lines (the whiskers) extending to the extremes of the data. If unusual extreme values occur, the whiskers extend only to those points that are within 1.5 times the inter-quartile range.



Scatter-plot or x-y plot

Plots two variables against each other.



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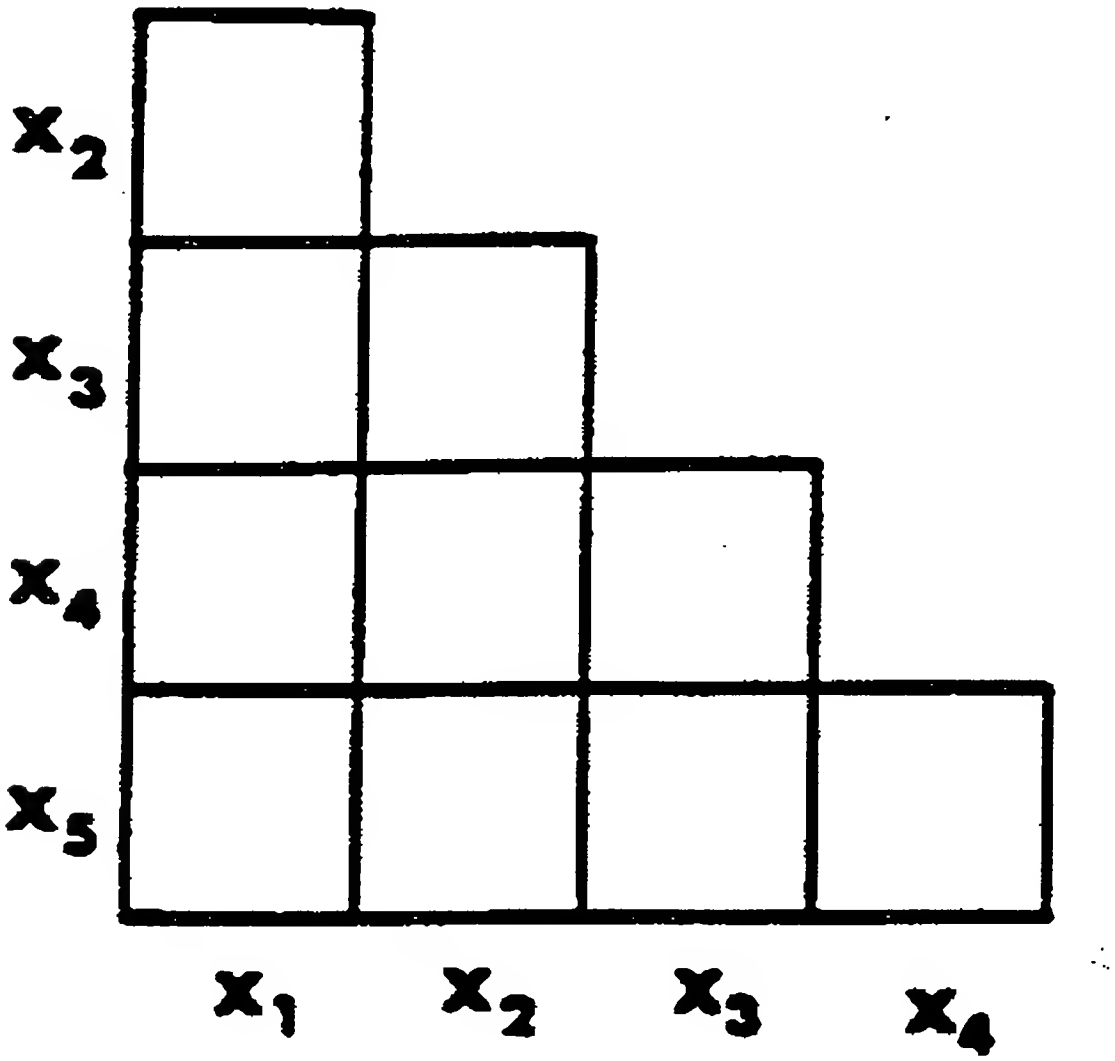
Draftmensplot

Plots two or more variables against each other!



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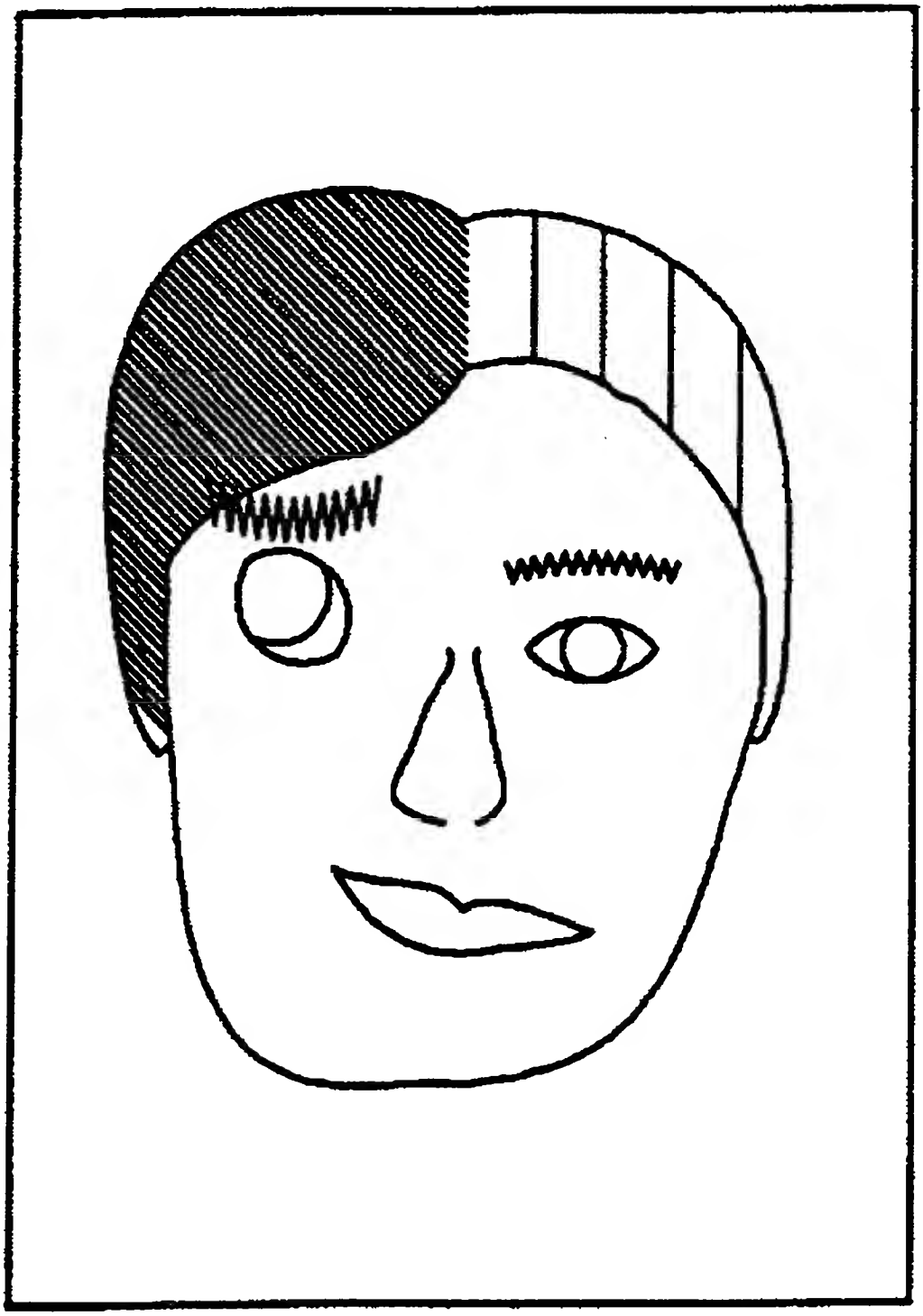
Flury-Riedevyl faces and Chemoff faces

Faces are a method to show multivariate data graphically. Each variable is mapped to one or more of the face parameters. Other methods to represent multivariate data are the star symbol plot, the sun ray plot, castles and trees.



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6.4 Control Charts

Control charts are special time-plots to show a possible change of the characteristics in a production process. The essence of a control chart is its clearness and intelligibility. In a quick survey it is possible to recognize natural subgroups, within which variation is likely to be random but among which assignable causes of one sort or another may cause non-random variation.

Summarized statistical errors (such as means or standard deviations) and confidence limits determined from overall sample data can be very misleading, if the data are not free from the effects of assignable causes. In cement industry annual means and standard deviations are mostly misleading due to a nonstationarity of the process, i.e. the occurrence of systematic changes at the production level. (Example: mean and standard deviation of the lime saturation of raw meal and/or clinker in annual reports. Changes in the target value required by the process or the product lead to a high non-interpretable standard deviation.)

If possible, the data should originally be collected with this subgrouping in mind; in such cases, the analyses can be simplified by arranging for an equal number of observations in each subgroup. If the data must be analyzed as they come, subgrouping may still be possible with knowledge of the data's source, i.e. obvious changes in the process conditions must be registered together with the data.

Control limits

To judge a significant change at the production level, the charts are completed with control limits. As a general rule it can be assumed that such a change is present, if a sample point falls outside of 3 σ-limits, where σ is the standard deviation of a homogeneous production phase.

This engineering rule has been found to work well. No exact probability is given for a chance variation beyond 3 σ-limits, but in general it is very s mall, in the order of perhaps 0.3%.

It may be of advantage to use two limits, such as a 2 σ-warning-limit and a 3 σ-action-limit. In this case, an action should already take place when two subsequent values lie between the two limits.

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The use of 3 σ-limits bases on pure statistical considerations. It may occur that these limits are in conflict with the required tolerances of an external standard (e.g. process, standard or market requirements). If the required tolerances are smaller than the statistical ones, a modification of the process is necessary to improve its precision. In the opposite case, the statistical limits may be used in order to detect changes as early as possible.

Control Charts

Several types of charts may be used to detect different types of changes in the process:

	Control chart			
Cause of Change	**Mean X**	**Range R**	**Standard deviations**	**Cumulative**
Gross error (blunder)	1	2	-	3
Shift in average	2-	-	1	
Shift in variability	-	1	2	-
Slow fluctuation (trend)	2	-	-	1
Rapid fluctuation (cycle)	-1	2	-	

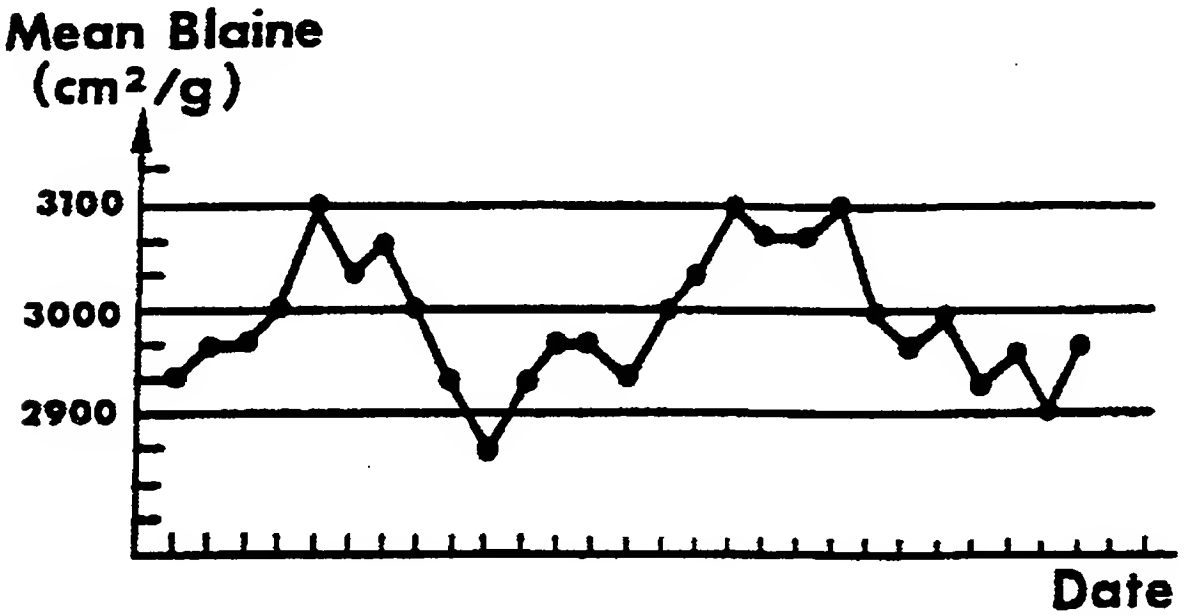
- = not appropriate / 3 = least useful

2 = useful / 1 = most useful

$\bar{x}$ – chart

Samples of fixed size n are taken from time to time. The arithmetic mean of the samples are plotted versus time.

Control limits (for warning and/or action) can be computed with the help of special procedures.



R-chart of s-chart

Given are small samples of fixed size n. The range or the standard deviation of the sample is plotted.

Values exceeding a certain control limit indicate an increase of variability.

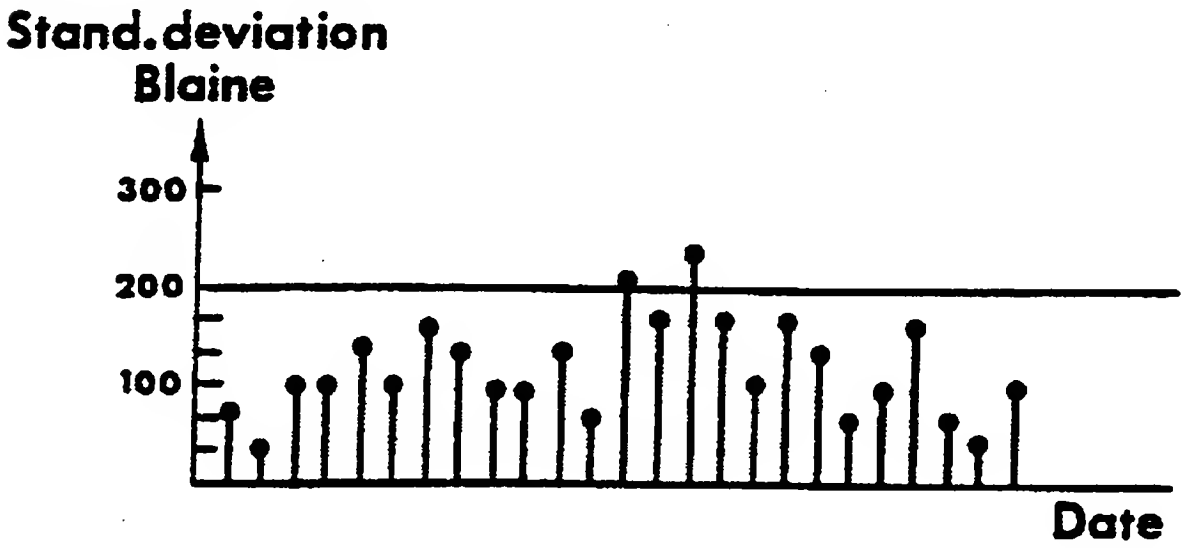
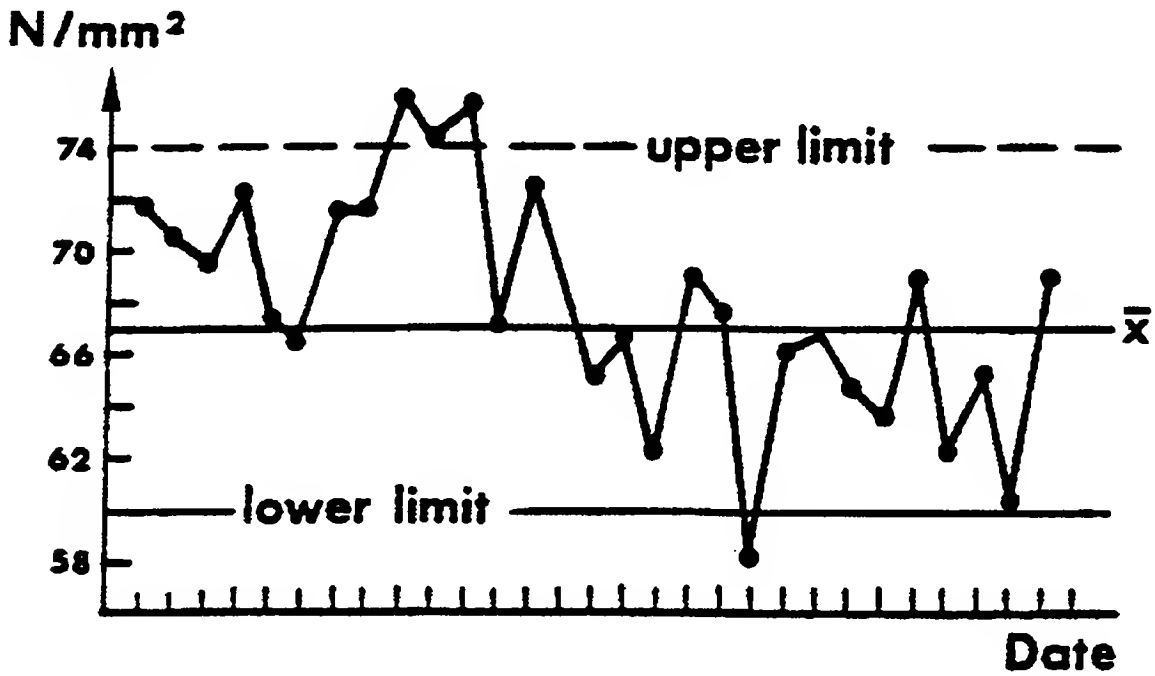


Chart of individual values

Instead of means, individual values ma y be plotted in the order of measurement.



CUSUM-chart

Cumulative-sum-chart. The deviations of individual values from a target value are cumulated.

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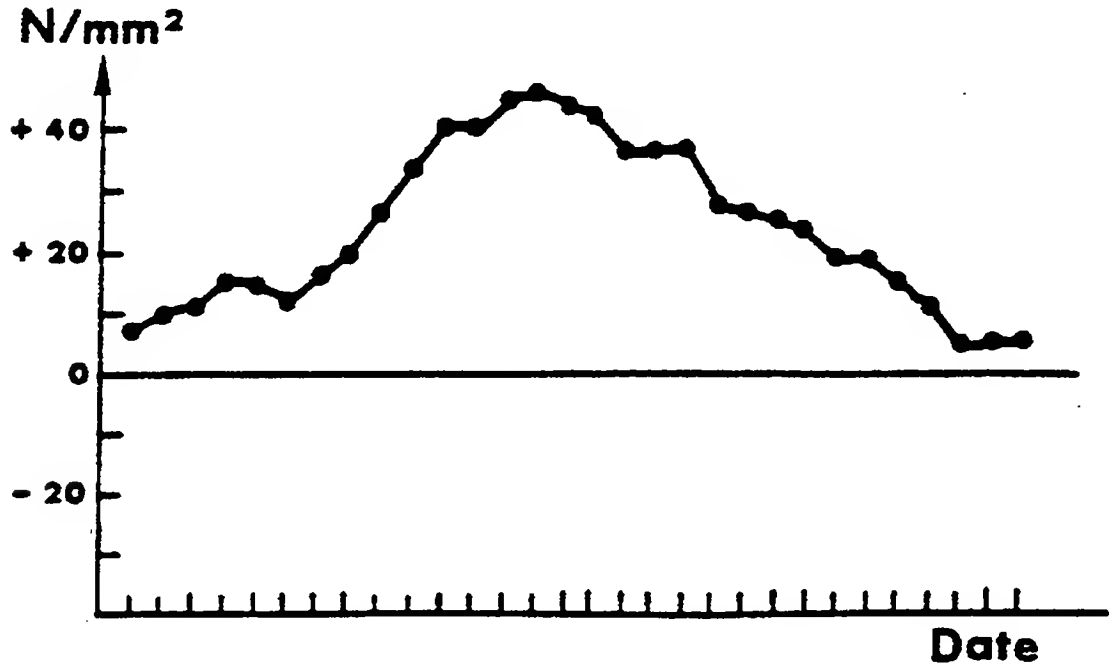
This chart is especially sensitive to slow fluctuations in the process.

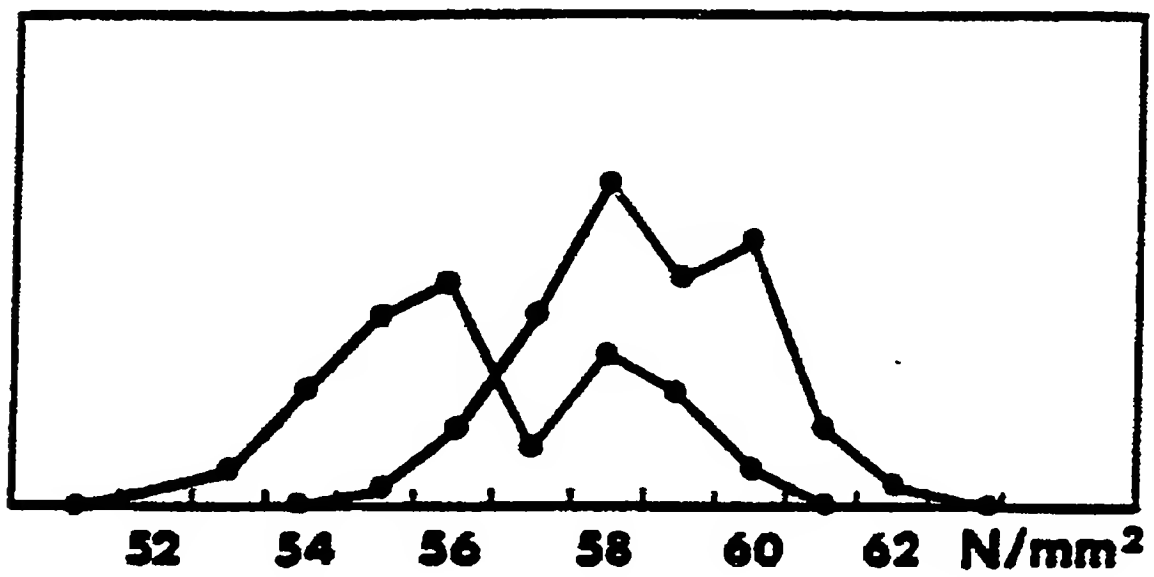


6.5 Comparative representation

Frequency polygon

Comparison of the distribution of two samples. Relative frequencies corresponding to the histogram are plotted on a line graph against the mid-points of the classes.

Example: Mortar strength in the first and the second half-year.



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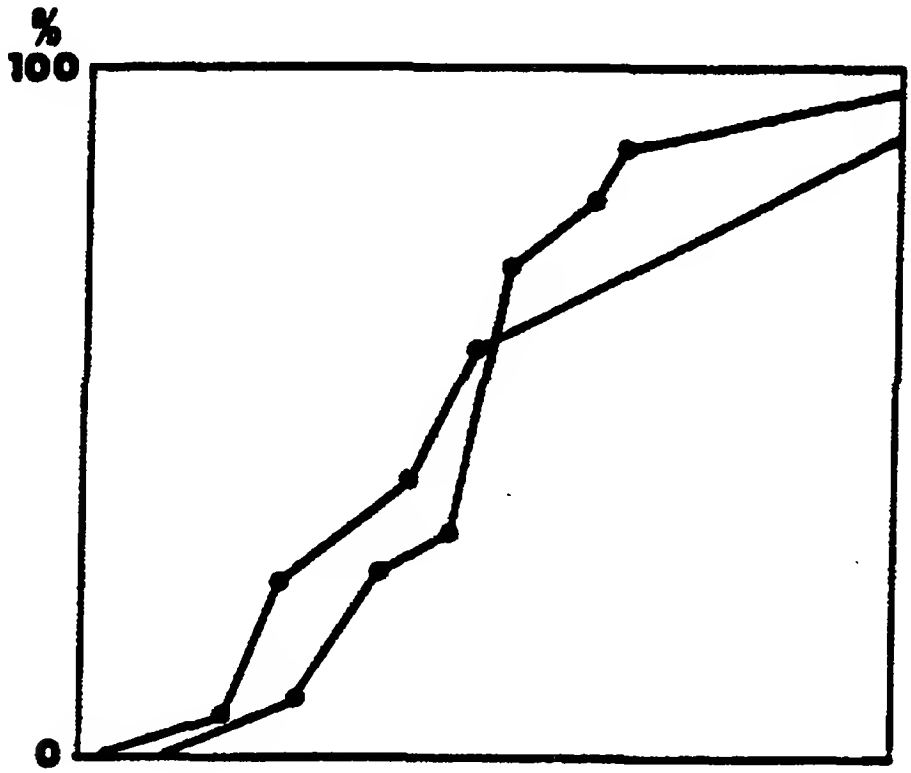
Cumulative Frequency plot

Comparison of the distribution of two samples. Relative cumulative frequencies are plotted against upper class limit.



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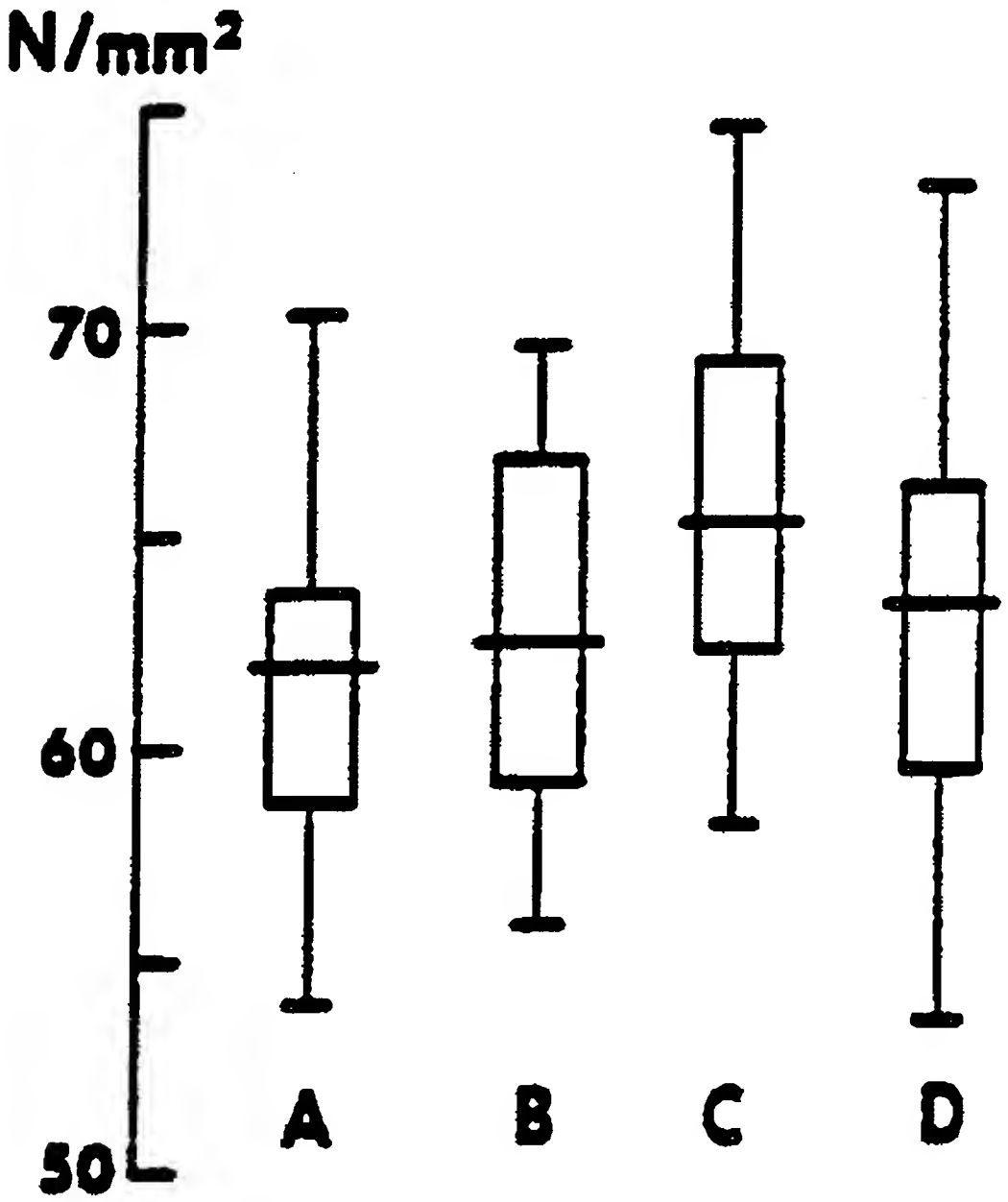
Grouped Box-Plots

Shows two or more box-plots in the same graph. May be used to show the distribution of different variables or the distribution of the same variable in different groups (samples). The notched box plot may be used for the latter comparison.



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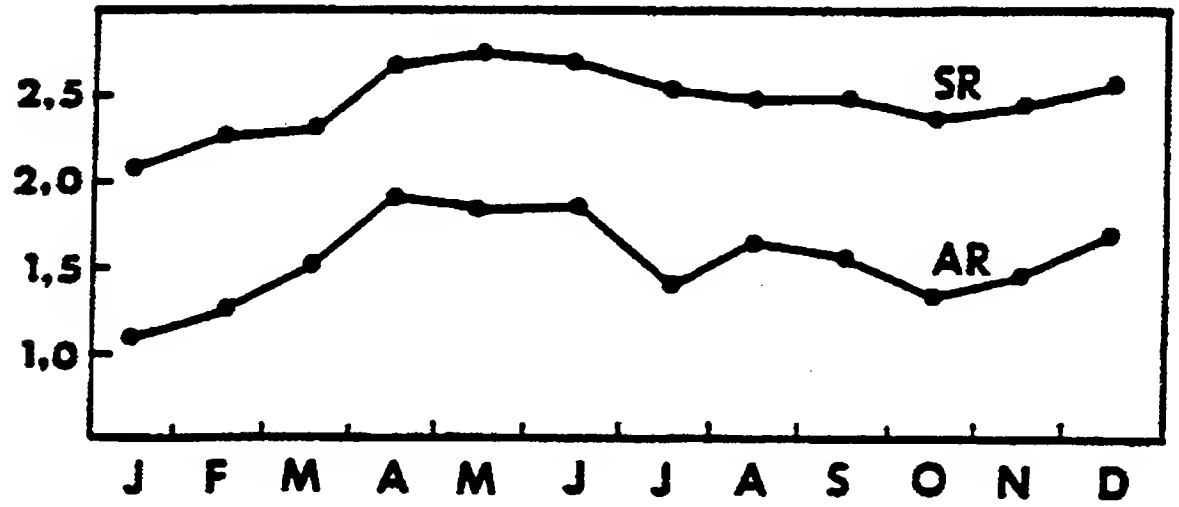
Time-plot

Comparison of simultaneous changes of several variables in an observed period.

Example: Silica- and alumina ratio of a cement type in a specific year.

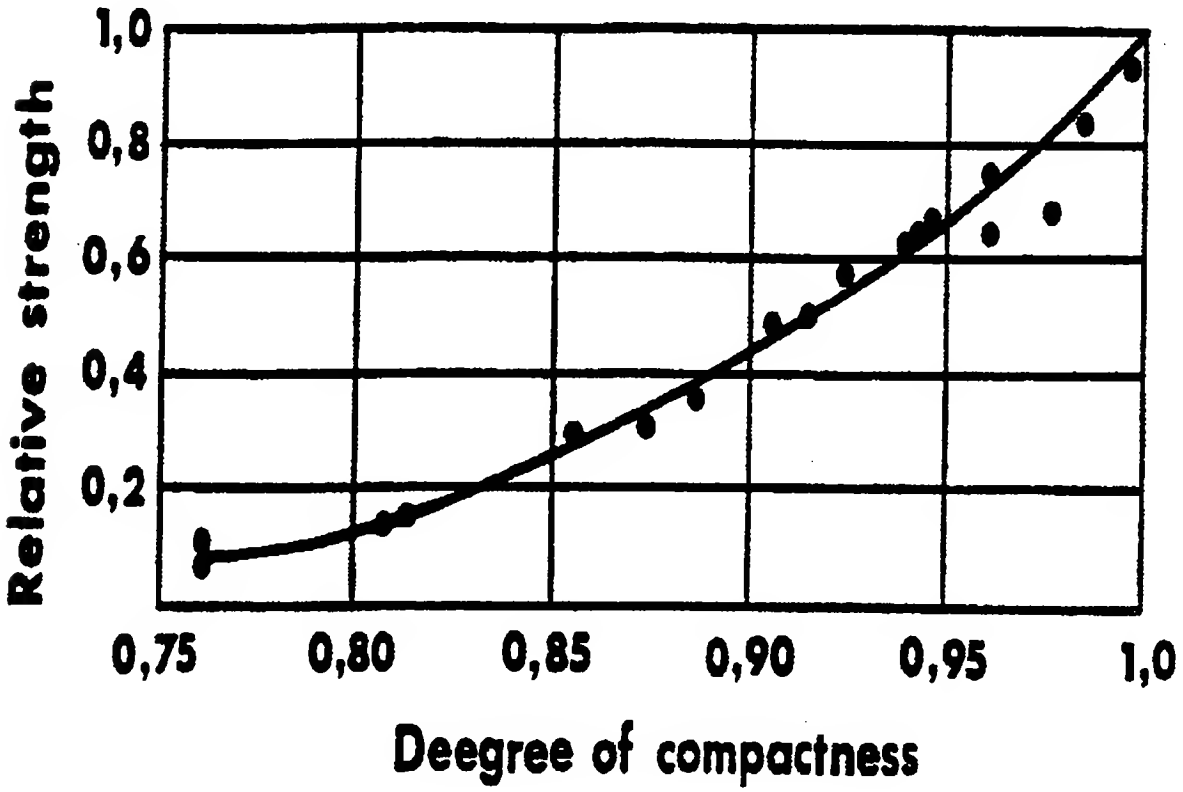


Scatter-plot

Illustration of the relationship between two variables.

Example: Relation between strength and density of concrete. The relationship is not linear.

Don't use the correlation coefficient in this case, because it is only a measure for linear dependence.



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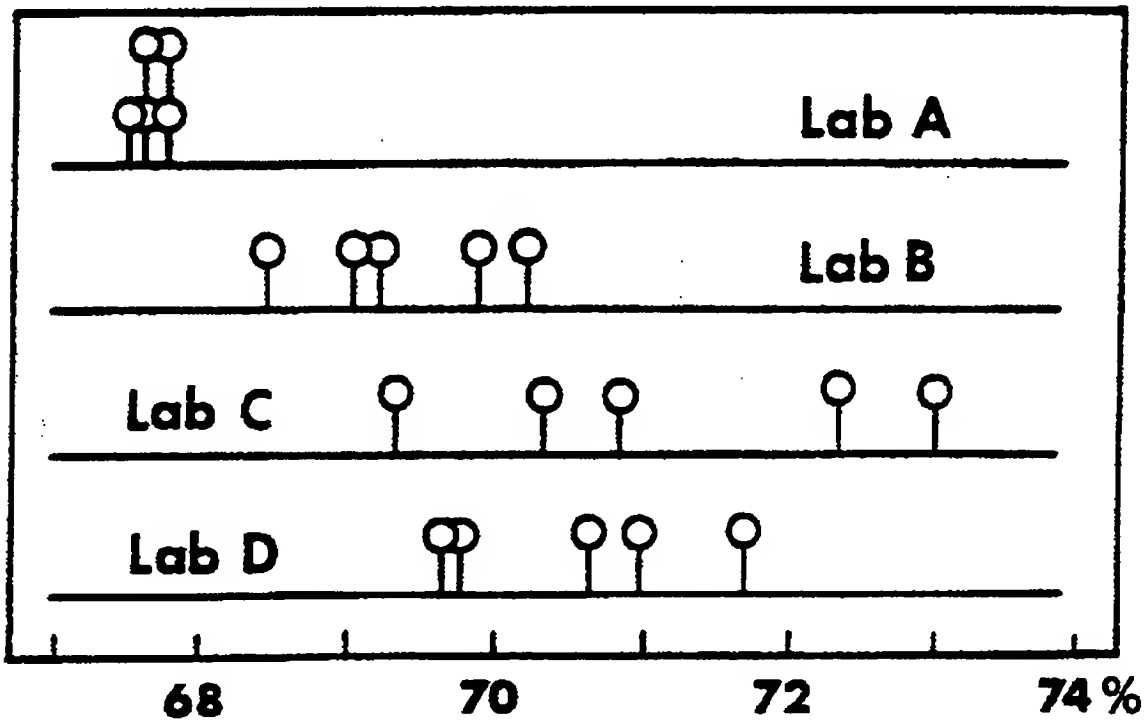
Comparison of small samples

Individual values are marked on the scale separately for each group of observations.

Example: Inter-laboratory-test concerned with elite-content.



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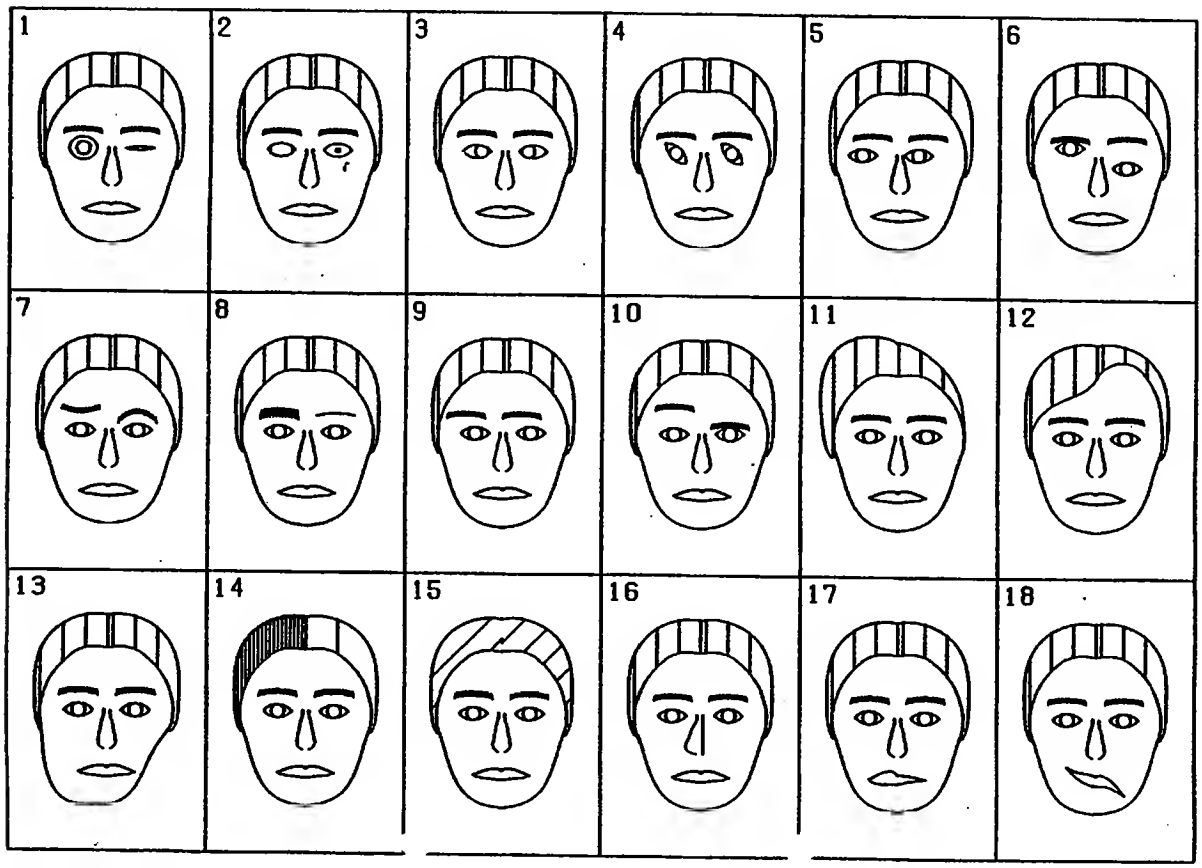
Asymmetrical Flury-Riedwyl Faces

This is a special version of the faces. E.g. the variables before and after treatment, respectively, are mapped to the same face parameter on the left and right face side, respectively. Or the left side shows the specification values of the variables and the right side the actual measurements.



7. CORRELATION AND REGRESSION

7.1 Correlation coefficient

The degree of linear dependence between two random variables X and Y can be expressed by the correlation coefficient r_{xy}.

$$r_{xy} = \frac{s_{xy}}{s_x s_y}$$

with $s_{xy} = \frac{1}{n-1}\sum_{i=1}^{n}(x_i - \bar{x})(y_i - \bar{y})$ the covariance of X and Y

and s_x, s_y standard deviations of X and Y.

For practical computations use

$$s_{xy} = \frac{1}{n-1}\sum x_i y_i \frac{1}{n}(\sum x_i)(\sum y_i)$$

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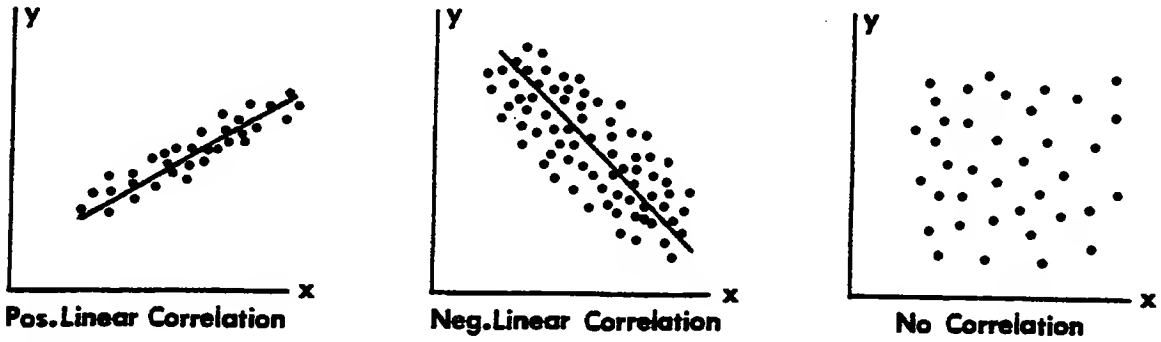
The following figure shows the scatter diagrams for several degrees of dependence.



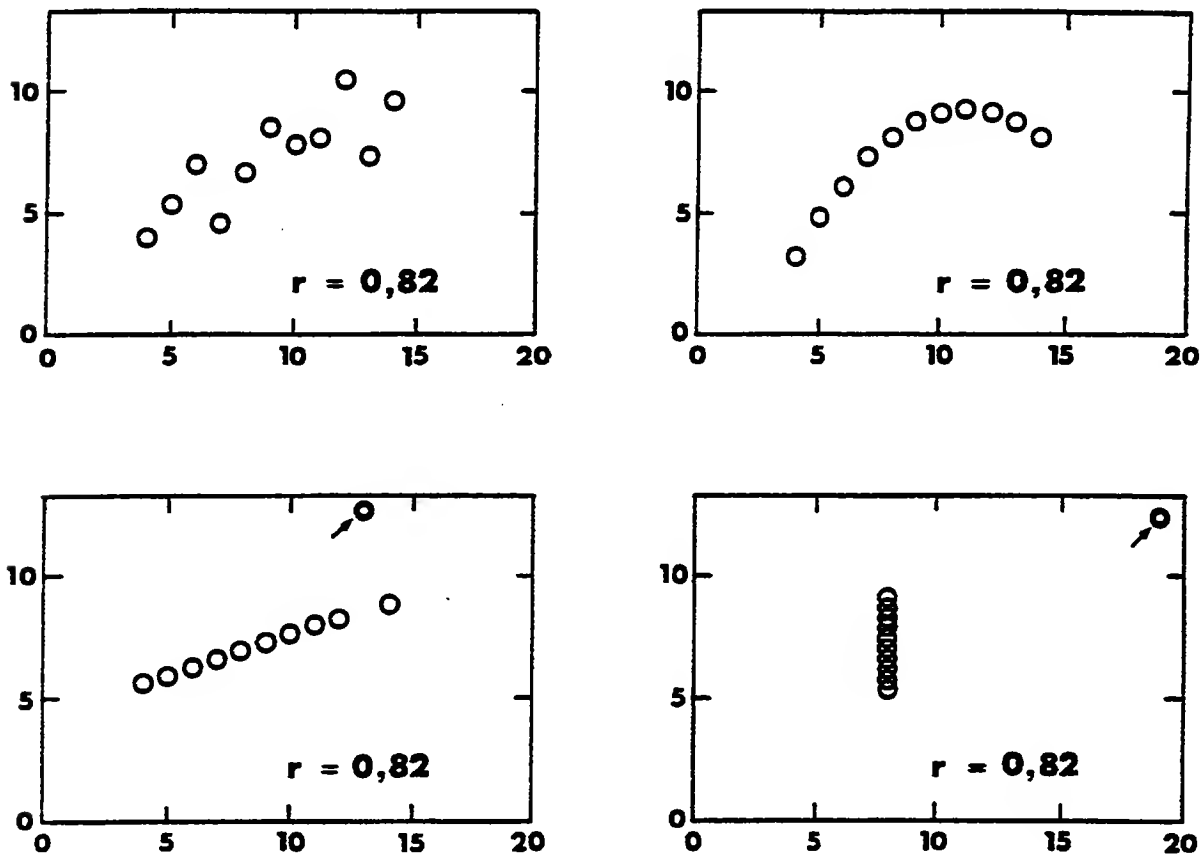
Properties of r_{xy}:

a) r_{xy} = ranges from -1 to +1
r_{xy} = +1 : All measured values lie on an increasing line
-1 : All measured values lie on a decreasing line
0 : No linear relationship between the measured values

b) r_{xy} is a measure of <u>linear</u> dependence. If the scatter diagram indicates a non-linear relationship, then the correlation coefficient will be misleading and should not be calculated.

c) r_{xy} is very sensitive (not robust) against outliers.

The necessity of drawing a scatter diagram is illustrated in the following figures. Completely different graphs may result with equal correlation coefficients (r = 0.82).



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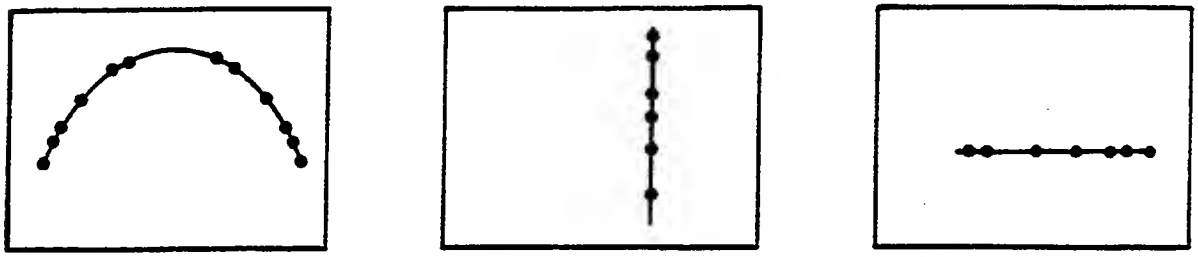
r_{xy} can be close to zero even though the variables are clearly non-linear dependent and r_{xy} is not defined if s_x or s_y is zero.



r = 0 **r is not defined**

Interpretation of r_{xy}

A high correlation coefficient between two variables does not necessarily indicate a causal dependence. There may be a third variable not under control which is causing the simultaneous change in the first two variables, and which produces a spuriously high correlation coefficient. In order to establish a causal relationship it is necessary to run a carefully controlled experiment (see chapter 8). Unfortunately it is often impossible to control all the variables which could possibly be relevant to a particular experiment, so that the experimenter should always be on the lookout for spurious correlation.

The following is an example for confusion of correlation with causation:

The following figure shows the population of Oldenburg at the end of each of 7 years plotted against the number of storks observed in the corresponding year. Although in this example few would be led to hypothesize that the increased number of storks caused the observed increase of population, investigators are sometimes guilty of this kind of mistake in other contexts. Correlation between two variables Y and X often occurs because they are both associated with a third factor W. In the stork example, since the human population Y and the number of storks X both increased with time W over this 7-year period, it is readily understandable that a correlation appears when they are plotted together as Y versus X.

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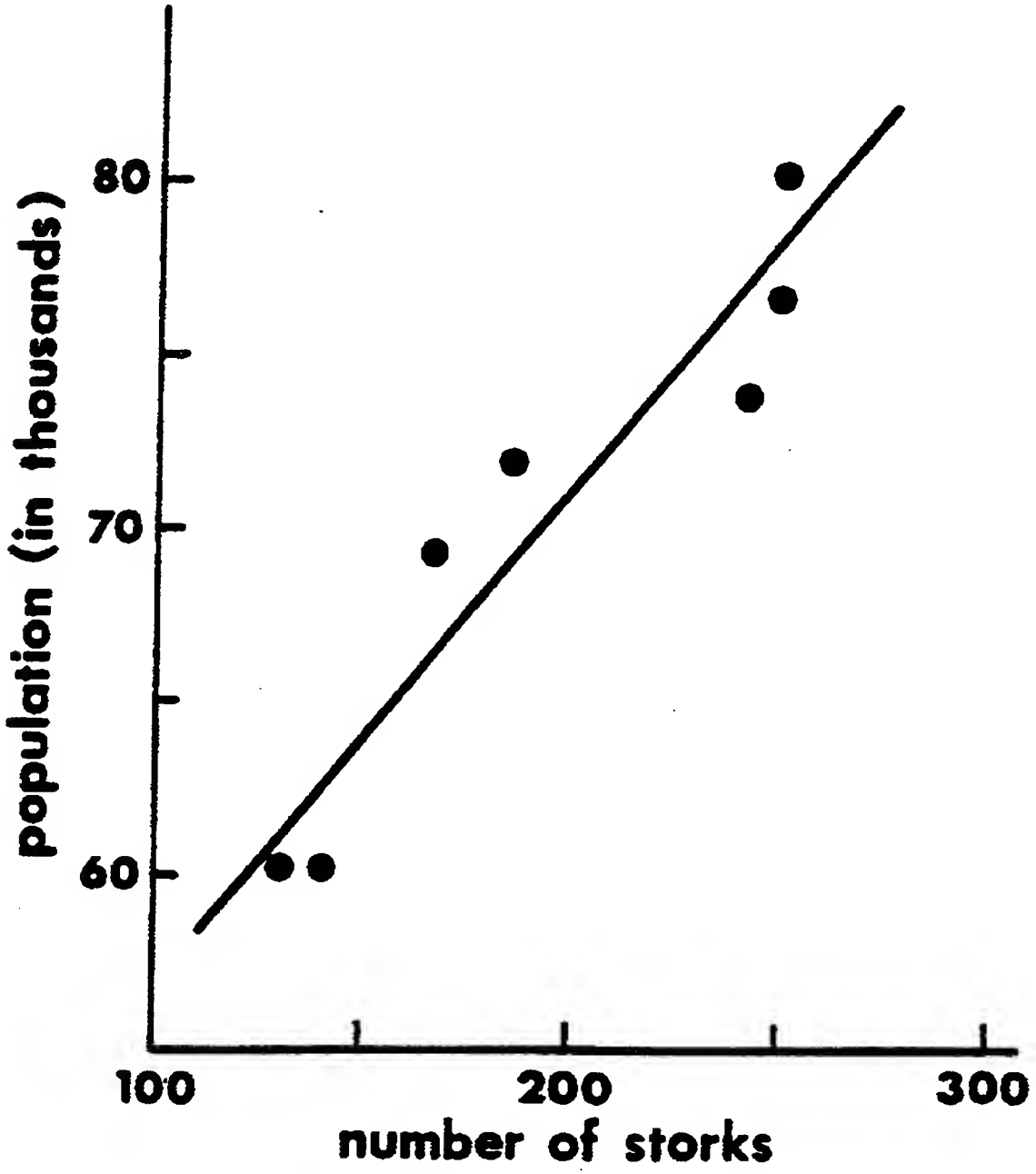
Fig.: **A plot of the population of Oldenburg at the end of each year against the number of storks observed in that year.**



By using sound principles of experimental design and, in particular, randomization, data can be generated that provide a more sound basis for deducing causality.

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7.2 Linear Regression

If the scatter diagram indicates that the two variables are linearly related, then we may want to predict the value of one of the variables from a given value of the other variable. For this purpose a regression line is fitted to the data (method of least squares).

Model assumptions

$$y_j = \alpha + \beta_j + E_j \qquad (1)$$

where the residuals E_j's are independent normal random variables with mean 0 and constant variance $\sigma 2$.

α = y -intercept

β = slope

x = independent variable - Y dependent variable

This assumption is essential if parameter reductions (see later), i.e. reduced models are tested and the assumptions have to be checked when a model is fitted.

The estimates of the parameters α and β in model (1) are obtained by the method of Least Squares (MLS), i.e. the estimates are obtained by minimizing the sum of squared residuals

$$\sum_{j=1}^{n}(y_j - \alpha - \beta x_j)^2 = \sum_{j=1}^{n}\hat{E}_j^{\,2}$$

with respect to α and β !

This method is very wide spread in statistics and the estimates have good statistical properties under the above model assumptions! Today there are also procedures which do not minimize the sum of squared residuals but use other criteria (Robust Methods!).

In the following let

$$s_{xx} = \sum(x_j - \bar{x})^2 = \sum x_j^{\,2} - \frac{1}{n}(\sum x_j)^2 = (n-1)s_x^{\,2}$$

$$s_{yy} = \sum(y_j - \bar{y})^2 = \sum y_j^{\,2} - \frac{1}{n}(\sum y_j)^2 = (n-1)s_y^{\,2}$$

$$s_{xy} = \sum(x_j - \bar{x})(y_j - \bar{y}) = \sum x_j y_j - \frac{1}{n}\sum x_j y_j = (n-1)s_{xy}$$

The resulting estimated regression line is

$$\hat{y} = \hat{\alpha} + \hat{\beta}\,, \text{ where } \hat{\beta} = \frac{s_{xy}}{s_{xx}}$$

$$\hat{\alpha} = \bar{y} + \hat{\beta}\bar{x}$$

with a resulting minimal sum of squares (MSSQ) $s_{\min} = s_{yy} - \frac{s^2{}_{xy}}{s_{xx}}$ with n-2 degrees of freedom (2 parameters α and β are estimated). $\hat{y}$ is the predicted value for the dependent variable y.

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Note:

1) In the mentioned problem there are two regression lines, one to predict y from x and one to predict x from y. The two lines are not identical and therefore it is not allowed to invert the regression equation.
2) The prediction equation is valid only in the observed range of observations. Extrapolations may give misleading results. We have no information that linearity holds outside of the present observations.

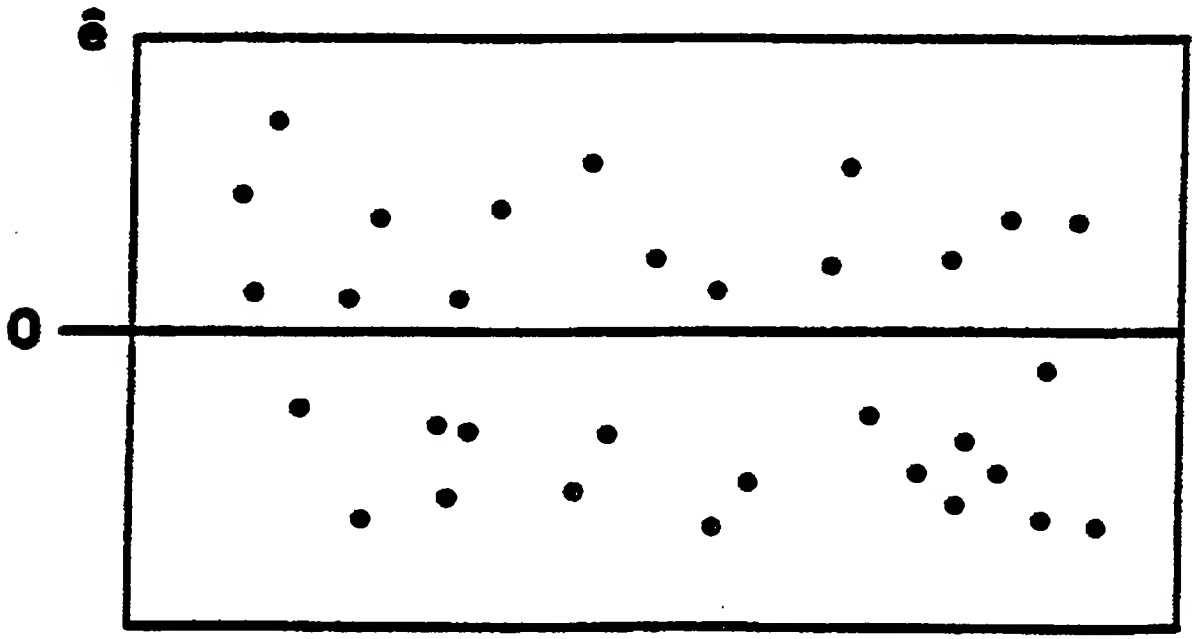
For each pair (x_j, y_j) we can therefore compute the predicted value $\hat{y}$ of y by means of the regression function $\hat{y} = \hat{\alpha} + \hat{\beta}x$ i.e.

$$\hat{y}_j = \hat{\alpha} + \hat{\beta}x_j$$

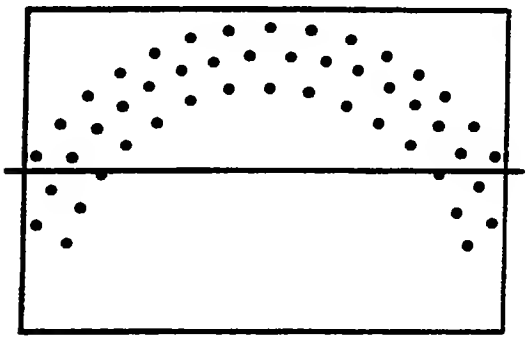
and the estimated residual

$$\hat{e}_j = y - \hat{y}_j$$

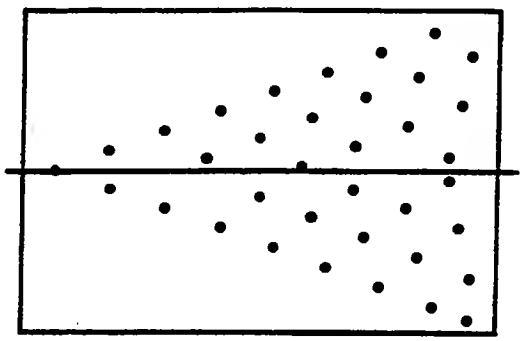
$\hat{\alpha}^2 = S_{min}/(n-2)$ is an estimate for the variance of the residuals. The analysis of the residuals $\hat{e}_j$ gives us the possibility to validate the model assumptions (Independence, Normality and constant Variability). The first check is done by plotting the residuals against the predicted y_j. The residuals $\hat{e}_j$ should be randomly scattered around the line e = 0 and show no pattern.



For example the following structures of the residuals would indicate, that the model is not correctly specified (e.g. that the variables x and y should be transformed before calculation of the regression line).



Relation not linear
(e.g. $y = h(\alpha + \beta x)$
or $y = \frac{1}{\alpha + \beta x}$



Variance not constant
(e.g. variance σ^2 proportional to x)

The second check would be a normal probability plot of the residuals.

If the residuals suggest that the assumptions do not hold, then further investigations are necessary (suitable transformations, other influencing variables, time-dependence => growth curves, time-series-analysis).

After confirmation of the assumption and calculation of the regression line (sometimes also before calculation) it is possible that we want to investigate whether perhaps the y-intercept equals zero, or the slope equals zero or whether there is no relation between y and x ($\alpha = \beta = 0$) or whether the α or β equal some specified values α_0 or β_0 (e.g. if a value of $\hat{\beta} = 1.1$ may be replaced by $\beta_0 = 1.0$). This means that we want to test if a reduced (simplified) model is sufficient to describe the relationship between y and x. For this it is essential that the model assumptions hold.

7.2.1 Regression line with slope 0:

Model: $y_j = \alpha + E_j$

Estimate $\hat{\alpha}$ for α : $\hat{\alpha} = \bar{y}$ (= mean of the y_j's)

Minimal Sum of Squares (MSQ): $S_{min} = S_{yy}$

Degrees of freedom (df): df = n-1

7.2.2 Regression line with y-intercept 0

Model: $y_j = \beta x_j + E_j$

Estimate $\hat{\beta}$ for β: $\hat{\beta} = \frac{\sum x_j y_{j1}}{\sum x_j^2}$

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MSQ: $S_{min} = \sum y_j^2 - \frac{(\sum x_j y_j)^2}{\sum x_j^2}$.

df = n-1

7.2.3 Intercept = 0. Slope = 0

Model: $y_j = E_j$

no parameters to estimate

$$MSQ = \sum y_j^2$$

df = n

Other models:

$$y = \alpha_0 + \beta x$$

$$y = \alpha + \beta_0 x$$

$$y = \alpha_0 + \beta_0 x$$

The estimates and MSQ can be obtained by differentiation of the corresponding Sum of Squared Residuals!

These special models are only useful if the data really speaks for them!

7.2.4 Comparison of models

This is done by performing an Analysis of Variance (ANOVA). ANOVA is a widely spread statistical technique. (Comparison of more than two means, Experimental Designs etc.). ANOVA compares the Minimal Sum of Squares of the models to each other. The basic results for the comparison are filled in a table, the ANOVA-Table.

ANOVA-TABLE

Model	MSQ	df
H_0	S^0_{min}	df^0
A	S_{min}	df
Reduction	$S^0_{min} - S_{min}$	df_0 - df

H_o denotes the null hypothesis, A the alternative model. By H_0 the test-statistic

$$F = \frac{(S^0_{min} - S_{min})/(df^0 - df)}{S_{min}/df}$$

is distributed according to a F-distribution with $m_1 = (df^0 - df)$ and m_2 = df degrees of freedom.

Significance limits: Look up $F_{1-\alpha;(df^0-df),df}$ in Table A-9 with $m_1 = (df^0-df)$ and m_2 = df degrees of freedom.

Decision: If $F > F_{1-\alpha;(df^0-df),df}$ then conclude that the simplification to the null-model H_0 is not permitted, therefore the alternative model has to be used; otherwise ($F \le F_{1-\alpha}$) there is no evidence that the null-model should be rejected.

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7.2.5 Standard deviation of the estimates

The ANOVA of the null hypotheses $H'_0 : y = \alpha$ and $H''_0 : y = \beta x$, resp. against the alternative $A : y = \alpha + \beta x$ allows us to compute the standard deviations (sd) of the estimated parameters $\hat{\alpha}$ and $\hat{\beta}$ in the alternative model $A : y = \alpha + \beta x$: let $F(\beta = 0)$ be the calculated F-statistic for the test H'_0 against A (e.g. ANOVA for $\beta = 0$) and correspondingly $F(\alpha = 0)$ for H''_0 against A.

Then

$$sd(\hat{\alpha}) = \frac{|\hat{\alpha}|}{\sqrt{F(\alpha = 0)}}$$

$$sd(\hat{\beta}) = \frac{|\hat{\beta}|}{\sqrt{F(\beta = 0)}}$$

so that the regression line of the alternative (full) model is

$$y = \underset{(sd(\hat{\alpha}))}{\hat{\alpha}} + \underset{(sd(\hat{\beta}))}{\hat{\beta}}\, x$$

In order to obtain the standard deviation in a model $A : y = \beta x$, this model has to be compared with the null-model $H_0 : y = 0$.

For $A : y = \alpha$, compare A to $H_0 : y = 0$

For $A : y = \alpha_0 + \beta_x$, compare A to $H_0 : y = \alpha_0$

For $A : y = \alpha + \beta_0 x$, compare A to $H_0 : y = \beta_0 x$

1-α confidence intervals for the estimated parameters can be obtained by calculating

$$\hat{\alpha} \pm t_{1-\alpha/2,m} sd(\hat{\alpha})$$

and

$$\hat{\beta} \pm t_{1-\alpha/2,m} sd(\hat{\beta})$$

where m equals the degrees of freedom in the alternative model and $t_{1_a/2,m}$ can be looked up in Table A-4.

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7.2.6 Coefficient of determination

The prediction of y with a regression line is more or less accurate dependent on the degree of linear dependence. How well does the regression line fit the data? A measure to express the relative accuracy of prediction compared with the total variation of y is the coefficient of determination r^2. In the case of a regression line the coefficient of determination is the square of the correlation coefficient

$$r^2 = r_{xy}^2 = \frac{s_{xy}^2}{s_x^2 s_y^2} = \frac{S_{xy}^2}{S_{xx} S_{yy}}$$

The total variation of y can be partitioned into two components. Total variation = explained variation + unexplained variation

where

total variation	$= \sum (y_i - y)^2$	
unexplained variation	$= \sum (y_i - \hat{y})^2$	(observed - predicted)
explained variation	$= \sum (\hat{y} - \bar{y})^2$	

The coefficient of determination is the ratio

$$r^2 = \frac{\text{explainedvariation}}{\text{totalvariation}}$$

In this form r^2 is defined also for non-linear curves and multiple regression.

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7.2.7 Transformations before Regression Analysis

Kind of transformation	Type of relation	Regression after transformation
$y' = \ln y, x (y > 0)$	$y = \alpha e^{\beta x}$ (exponential)	$y' = \ln + \beta x$
$y, x' = \ln x, (x > 0)$	$y = \alpha + \beta \ln x$ (logarithmic)	$y = \alpha + \beta x'$
$y' = \ln y, x' = \ln x, (x, y > 0)$	$y = \alpha x^{\beta}$ (exponentiation)	$y' = \ln \alpha + \beta x'$
$y' = 1/y, x (y \neq 0)$	$y = (\alpha + \beta x)^{-1}$	$y' = \alpha + \beta x$
$y, x' = 1/x (x \neq 0)$	$y = \alpha + \beta \frac{1}{x}$	$y = \alpha + \beta x'$
$y' = 1/y, x' = 1/x (x, y \neq 0)$	$y = \frac{x}{\alpha x + \beta}$ (hyperbolic)	$y' = \alpha + \beta x'$
$y' = \ln y, x' = 1/x (y > 0, x \neq 0)$	$y = \alpha e^{\beta / x}$	$y' = \ln \alpha + \beta x'$
$y' = 1/y, x' = e^{-x} (y \neq 0)$	$y = \frac{1}{\alpha + \beta e^{-x}}$	$y' = \alpha + \beta x'$

Model Overview for Linear Regression

Model Overview for Linear Repression

Model	Parameter Estimate(s)	MSQ (=Smin)	df
$\alpha + \beta x$	$\beta = S_{xy}/S_{xx}$ $\alpha = \bar{y} - \beta \bar{x}$	$S_{yy} - \left(\frac{S_x^2 y}{S_{xx}}\right)$	n - 2
$\alpha_o + \beta x$ (α_0 fest)	$\beta = \frac{\Sigma x_i y_i - \alpha_o \Sigma x_i}{\Sigma x^2}$	$\Sigma y_i^2 + n\alpha_o [\alpha_o - 2\bar{y}] - \frac{(\Sigma x_i y_i - \alpha_o \Sigma x_i)^2}{\Sigma x_i^2}$	n - 1
βx (α_o=0)	$\beta = (\Sigma x_i y_i)/\Sigma x^2_i$	$\Sigma y_i^2 - (\Sigma x_i y_i)^2/\Sigma x_i^2$	n - 1
$\alpha + \beta_o x$ ($\beta_o$fest)	$\alpha = \bar{y} - \beta_o \bar{x}$	$S_{yy} - 2\beta_o S_{xy} + \beta_o^2 S_{xx}$	n - 1
α (β_o=0)	$\alpha = \bar{y}$	S_{yy}	n - 1
$\alpha + x$ (β_o=1)	$\alpha = \bar{y} - \bar{x}$	$S_{yy} - 2 S_{xy} + S_{xx}$	n - 1
$\alpha_o + \beta_o x$	---	$S_{yy} + n[\alpha_o - (\bar{y} - \beta_o \bar{x})]^2 + \beta^2_o S_{xx} - 2\beta_o S_{xy}$	n

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7.2.8 Multiple and non-linear regression

Often the variable to be predicted is not only dependent on one but on several independent variables.

Model:

$$Y_j = \beta_0 + \beta_1 x_{1j} + \beta_2 x_{2j} + \cdots + \beta_p x_{pj} + E_j$$

where E_j are independent normal random variables with mean 0 and constant variance σ^2. Again these assumptions have to be checked after a model has been fitted.

In this case prediction can be improved in a multiple (or multivariate) regression expressed as

$$\hat{y} = \hat{\beta}_0 + \hat{\beta}_1 x_1 + \hat{\beta}_2 x_2 + \hat{\beta}_3 x_3 + \cdots + \hat{\beta}_p x_p$$

Or the scatter diagram shows a non-linear relationship which demands to fit a non-linear curve to the data. Often a logarithmic or some other transformation of one or both variables may lead to a linear relationship, and a regression line can be fitted to the transformed data. In more complex situations further methods are available.

For multiple, non-linear and mixed (multiple/non-linear) procedures consult literature (ref. Chatfield 1975)

Flury + Riedwyl (1988) give a very good praxis oriented introduction to multiple linear regression and multivariate analysis, including Discriminant Analysis, Principal Components, Identification and Specification analysis.

Example 6

The following data represent

y: water requirement (%) and

x: grain fraction 10-32 µ (%)

of n = 34 different cements.

x	y	x	y	x	y	x	y
32.9	25.0	38.6	27.2	29.8	23.3	37.2	26.8
35.4	26.3	32.5	25.4	30.3	25.6	42.2	25.6
33.1	24.9	31.3	25.1	36.0	25.2	35.8	24.6
38.7	27.9	35.1	26.9	30.2	24.5	39.1	25.2
33.8	24.8	32.2	24.4	35.6	24.0		
32.9	25.0	31.5	24.8	40.6	26.4	40.1	26.6
29.6	24.0	31.1	25.4	40.3	25.2	43.2	27.4
29.7	25.0	38.8	25.6	38.1	25.5	36.7	24.5
36.1	26.2	42.1	27.0	34.7	25.8		

The scatter diagram shows a positive linear relationship between water requirement and grain size. Regression lines to predict x or y are clearly distinct. The lines are identical if r_{xy} is 1 or -1.

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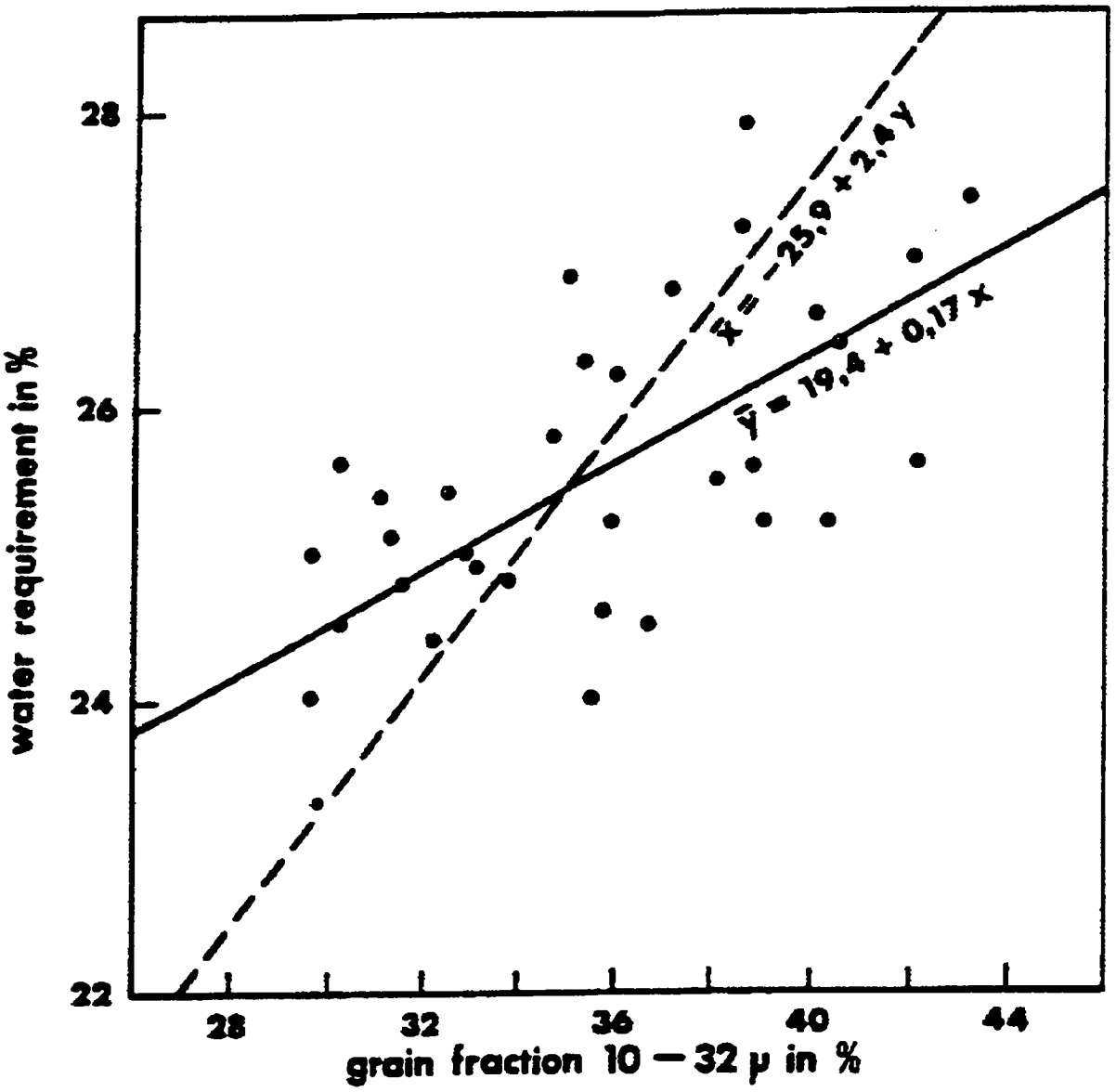
If r = 0 the lines are perpendicular, i.e. the best prediction of y is the arithmetical mean $\bar{y}$. In the same way $\bar{x}$ is the best prediction of x if no dependence exists.



Computations

In a first step five auxiliary sums are provided:

$A = \Sigma x_i$ = 1'205.3 n = 34

$B = \Sigma x^2_i$ = 43'253.19

$C = \Sigma y_j$ = 867.1

$D = \Sigma y^2_i$ = 22'150.93

$E = \Sigma x_j y_j$ = 30'828.44

$$s_{xy} = \frac{1}{n-1}\left[E - \frac{AC}{n}\right] = 2.719545$$

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$$s_x^2 = \frac{1}{n-1}\left[B - \frac{A^2}{n}\right] = 15.918333$$

$$s_y^2 = \frac{1}{n-1}\left[D - \frac{C^2}{n}\right] = 1.131203$$

$\bar{x} = A/n = 35.45$

$\bar{y} = C/n = 25.05$

$S_{xy} = 89.7550$

$S_{xx} = 525.3050$

$S_{yy} = 37.3297$

$S_{min} = 21.9973$

df = 32

$\hat{\sigma}^2 = 0.6874$

$\hat{\sigma} = 0.8291$

Correlation coefficient: $r_{xy} = \frac{S_{xy}}{\sqrt{S_{xx}S_{yy}}} = 0.64$

Regression line: $\hat{\beta} = \frac{S_{xy}}{S_{xx}} = 0.171$

$\hat{\alpha} = \bar{y} - b\bar{x} = 19.45$

Test: $\alpha = 0$

Model	**MSQ**	**df**
$H_0 : y = \beta x$	178 I527	33
$A : y = \alpha + \beta x$	21.9973	32
Reduction	156.1553	1

$$F = \frac{156.1553}{21.9973/32} = 227.16 = F(\alpha = 0)$$

$F_{.95;1,32} = 4.17$ (Table A-9)

→ Simplification not permitted

Test $\beta = 0$

Model	MSQ	df
Ho : $y = \alpha$	37.3297	33
A : $y = \alpha + \beta x$	21.9973	32
Reduction	15.3324	1

$$F = \frac{15.3324}{21.9973/32} = 22.304 = F(\beta = 0)$$

$F_{95,1,32} = 4.17$ (Table A-9)

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→ Simplified model not allowed !

We obtain the following standard deviations for the parameter estimates $\hat{\alpha}$ and $\hat{\beta}$

$$sd(\hat{\alpha}) = \frac{|\hat{a}|}{\sqrt{F(\alpha = 0)}} = \frac{19.45}{\sqrt{227.16}} = 1.2905$$

$$sd(\hat{\beta}) = \frac{|\hat{\beta}|}{\sqrt{F(\beta = 0)}} = \frac{0.171}{\sqrt{22.30}} = 0.0362$$

and the regression line is:

$$y = \underset{(1.29)}{19.45} + \underset{(0.036)}{0.171}x \quad \hat{\sigma} = 0.8291$$

95% confidence intervals for $\hat{\alpha}$ and $\hat{\beta}$

From table A-4: $t_{.975,32} = 2.042$

$$\hat{\alpha} : 19.45 \pm 2.042 * 1.29 = (16.82, 22.08)$$

$$\hat{\beta} : 0.171 \pm 2.042 * 0.036 = (0.097, 0.245)$$

Coefficient of determination: $r^2 = 0.41$

Comment:

Only 41% of the total variation of Y can be explained by the dependence of the grain fraction. The prediction can be improved when further variables are considered in a multiple regression. The coefficient of determination increases to 69% if C_3A- and alkali-content is included in the equation

$y = 17.44 + 0.15x_1 + 0.26\ x_2 + 0.52x_3$
x_1 = grain fraction 10-32µ%
x_2 = C_3A- content %
x_3 = $K_20 + Na_2O$%

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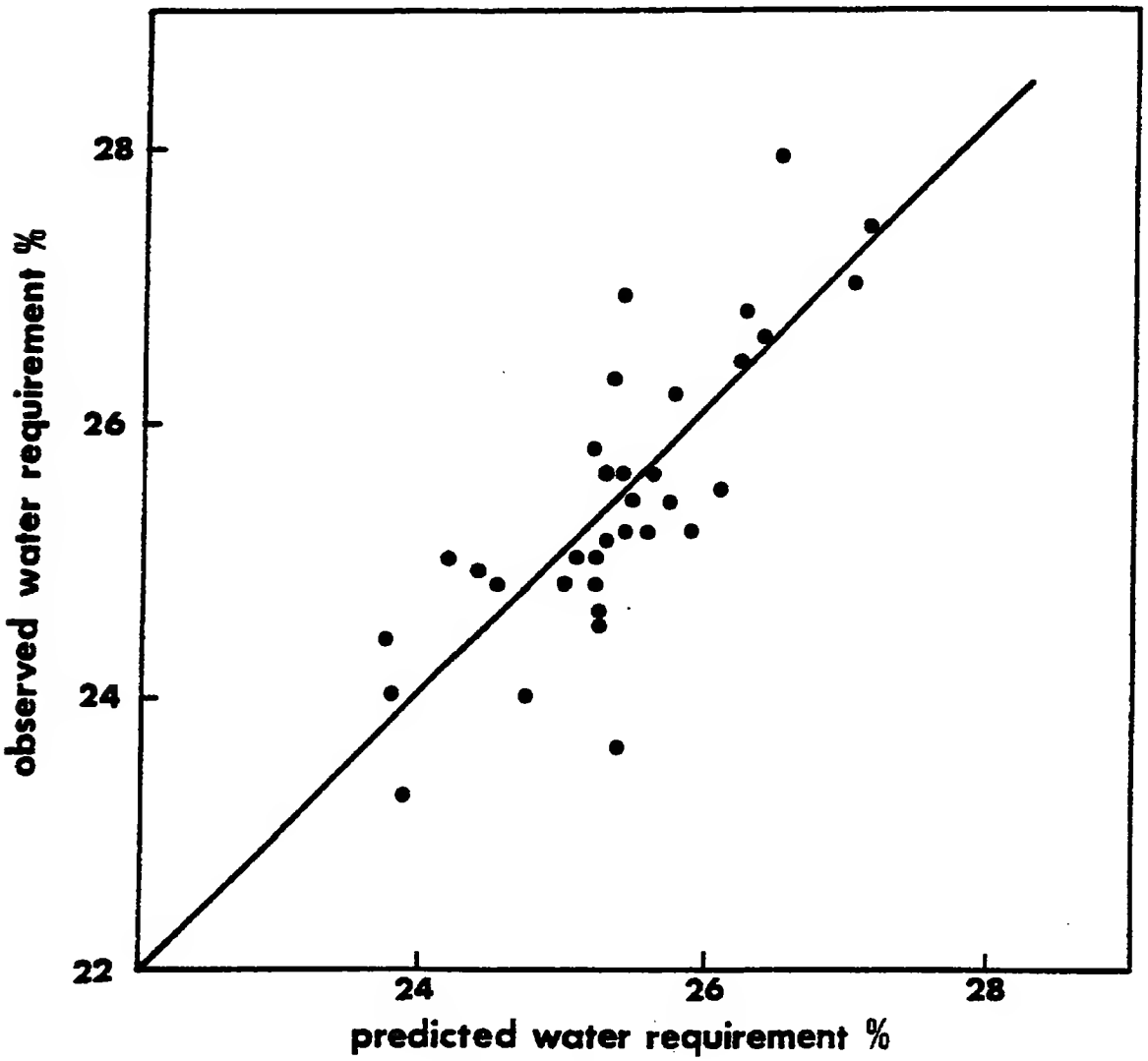
The improvement is visible in a scatter diagram of observed against predicted values.



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8. STATISTICAL INVESTIGATIONS AND THEIR DESIGN

In practice numerous problems and questions cannot be answered unless special quantitative investigations are performed. This applies to every practical field, starting with the raw material exploitation and ending with the sale of the finished product. In many cases, it is not possible to attain complete information as an effective basis for decision making. For this reason, it is becoming more and more common to resort to statistical investigations in form of sample surveys and experiments.

The proper performance and interpretation of statistical investigations is essential. The so-called "lies of statistics" generally have to be attributed to the wrong selection of data, a misinterpretation, or procedures which do not relate to the objective. Experience has shown that very rarely data are manipulated intentionally. However, people who employ "wrong statistics", are convinced, due to a lack of in-depth knowledge, that their argumentation is objective and correct.

8.1 The Five Phases of a Statistical Investigation

Despite the variety of potential applications, a statistical investigation can roughly be divided into five phases in the following order:

1) Formulation of problem:
   exact definition of purpose for which information is acquired
2) Planning:
   purpose-oriented planning of investigation according to statistical principles with the objective to acquire optimum information at given expenses. In case of doubt and problems contact an experienced statistician.
3) Performance:
   procurement of data strictly in accordance with the established planning
4) Evaluation:
   summary and presentation of results; inference from sample to population
5) Conclusions:
   realistic interpretation of results

Essential basic principle:

Statistical investigations will only provide truly efficient results if knowledge in the investigated field is optimally combined with thorough statistical knowledge. Usually this will lead to teamwork, since the investigator who lacks extensive knowledge in statistics is usually just as unsuccessful in his attempt to carry out the investigation on his own as the statistician who is entrusted with the entire problem complex.

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8.2 Sample Surveys and Experiments

Statistical investigations are subdivided into two groups sample surveys and experiments - which differ significantly with respect to their objective and interpretation.

Experiment

In an experiment response variables are investigated, which depend on various factors. The results are produced by a controlled variation of the factors of primary interest, whereas all other effects on the response have to be eliminated.

Factors that are being studied may be quantitative (e.g. temperature, concentration) or qualitative (e.g. type of ample preparation, origin of samples).

In an experiment, causal effects on the response variable can be determined quantitatively if the necessary provisions have been made in the planning and performance.

Sample survey

Samples are taken from one or several populations (the set of sample units is called sample of size n). Certain interesting characteristics are measured in order to evaluate (estimate)unknown characteristics of the populations or to compare populations. Contrary to the experiment, the results of a sample survey do not permit any conclusions regarding cause and effect. This is often ignored when correlations between two characteristics are assessed.

Reliable results of both experiments and sample surveys can only be obtained if the respective statistical principles have been adhered to in the planning and performance of the investigation.

8.3 Fundamental Principles in Statistical Investigations

8.3.1 Experiments

The five criteria of a good experiment are:

1) The experiment should serve a well defined purpose
   In a first step the problem has to be clearly identified for each experiment, and the hypotheses to be investigated have to be determined.

2) Factors. which are not of primary interest. should not influence the results
   Influencing factors, which are not included in the investigation, must be under control at fixed levels. Otherwise several effects become mixed up and cannot be separated by statistical methods. Systematic effects of factors, which cannot be controlled, are eliminated by random allocation of samples to treatments or factor levels and by random measuring sequence (randomization).
   Practical performance of randomization: In example A 1 we assign a number to each sample, No. 1 - 8 to tablets and No. 9 - 16 to beads. Each number is written on a leaflet and mixed in a box. The order of measurement is given by blind drawing of the leaflets from the box.

3) The experiment should be free of systematic errors
   This requirement is partially connected with 2). Systematic errors, due to any change of effects in time, are eliminated by randomization. An often underestimated source of systematic errors is the prejudice of the experimenter. In order to avoid this, mainly "blind" experiments should be performed. The samples are coded with random numbers, the decoding key only known to a confidential person, who herself is not in a position to perform analyses.

4) The experiment should provide a measure of its precision
   An estimate of the precision is obtained by a replication of the experiment. For each step

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of the experiment several measurements should be taken. In example A 1 it would be impossible to state whether a systematic difference exists, if only one tablet and one bead are measured. (Appendix I)

5) Precision and efficiency of an experiment should be high enough to reach the set goals
There are various measures to improve the precision and efficiency of an experiment:
   - Reduction of variability by using homogeneous materials and by carefully controlling all the factors, as well as by strictly observing the analytical regulations.
   - Increasing the number of replications; this will lead to the problem of determining the sample volume required to achieve the desired precision.
   - Blocking: Measurements are performed in homogeneous groups (blocks). Such blocks may be 'measurements made by the same operator' or 'measurements made on the same day'.
     A special blocking procedure is the paired comparison. In example A 3 both laboratories measure the compressive strength on samples drawn from the same cement bag.
     Block experiments should be balanced and randomized, i.e. the number of measurements and treatments is equal in each block and assignment of samples or treatments to blocks is random.
     Special cases of blocking: paired comparison.
   - Analysis of covariance: factors, which are not subject to the experimenter's control, are recorded in order to eliminate their effects in a later analysis.

The principles above are valid for any experimentation. Special designs for various problems are given in the literature. Problems may be: Effects of one or several factors on a response, additivity of factor effects, splitting of components of variability in a measurement procedure in order to achieve a prescribed precision at minimum cost, inter-laboratory tests, calibration, evaluation of systematic errors etc.

Note:

In contrast to a regression analysis with a given set of observations, a carefully designed experiment allows the evaluation of causal relationships. In regression the effects are usually mixed and can only be separated due to some mathematical model and not due to their real origin (ref. Interpretation of r_{xy} section 7.1).

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STRUCTURE OF AN EXPERIMENT

1) Recognize problem
2) Describe problem in detail and determine hypotheses
3) Define experimental area; contact statistician; determine factors, levels and number of replications; elaborate exact analytical regulations; determine a reference basis if required. It is often necessary to analyse what happens if no treatment is applied (e.g. Placebo in clinical experiments).
4) Establish experimental design, taking into account methods to improve precision (blocking).
   Randomization:
   - random selection of sample units
   - random allocation of treatments
   - coding of samples
5) Specify variables, which cannot be maintained constant (covariables)
6) Determine number of replications
7) Determine procedural organization
8) Determine procedures of statistical data analysis; exact regulation concerning the manner in which data are to be supplied (form)
9) Perform experiment
10) Evaluate results (statistical data analysis)
11) Draw conclusions
12) Take measures

Step 2) to 7) are designated as experimental planning. The results of this planning should absolutely be recorded in an experimental plan. Since experiments are usually quite expensive, careful planning will definitely be worthwhile.

The request to prepare a written experimental plan entails significant advantages:

- the necessity to formulate statements in precise terms
- clear conditions and thus less uncertainty in the performance of the experiment
- the circumstances of the experiment can be reconstructed, if later on the results are used for comparison with new results.

8.3.2 Sample Surveys

Sample surveys and experiments are performed in an analogous way. Differences are due to the different situation. While the purpose of an experiment is to directly produce results, sampling is carried out to analyse existing characteristics of a population.

Special attention should be paid to the following points:

1) Target population and sampled population
   The target population is the aggregate about which the investigator is trying to make inferences from his sample. It is usually helpful to focus the attention on differences between the population actually sampled and the population that is attempted to be studied.

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2) Sample unit
The sample units have to be described in clear terms. They should not overlap and the sum of all the sample units must be equal to the investigated population.
In the cement production for instance, if an individual sample is analysed, the results and interpretations are different from those of an analysis performed on a daily composite sample.

3) Representativity and sampling method
Samples only supply unbiased results if they represent the entire population. It is often mistakenly believed that the large sample size n ensures its representativity, or it is simply maintained that the sample is representative in order to prevent rejection of the results.
Representativity can only be ensured by an appropriate, random sampling method, giving each sample unit an equal chance to be included in the sample (exceptions in special cases: sampling with unequal probabilities, systematic sampling).

In literature a number of selection procedures are described. Depending upon the problem situation, various methods ma y be considered. They differ in their practicability, financial consequences (expenses) and efficiency.

In most cases an optimum selection procedure for a specific problem situation can be found.

STRUCTURE OF A SAMPLE SURVEY

1) Recognize problem. What information is required about what populations?
2) Detailed definition of problem
   - definition of target populations
   - what information with what accuracy?
3) Determine the sample population. Where, when and how are the sample units extracted from the population? Determine the variables to be recorded
4) Determine sampling method (equal probability sampling, stratified sampling, cluster sampling, multi-phase sampling, multi-stage sampling, systematic sampling, unequal probability sampling)\
5) Determine sample scheme (in accordance with the required accuracy)
6) Outline procedural organization
7) Determine procedures of statistical data analysis; exact regulation concerning the manner in which data are to be supplied
8) Perform sampling
9) If possible perform measurements in random sequence
10) Evaluation and presentation
11) Conclusions
12) Take measures

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9. OUTLOOK

This section will give a short outlook to further and perhaps more sophisticated statistical applications. It also contains references for further readings. All these applications require more than elementary knowledge of statistical methods.

9.1 Time series and growth curves analysis

This is the analysis of chronological observations, (e.g. daily temperatures at a certain location or observations over a certain time period of a patient obtaining a certain medicament). The characteristic of these data are that the observations are not independent from each other and the method of linear regression cannot be applied.

Literature:

Box, G.E.P. and Jenkins, G.M. (1976). Time Series Analysis, Forecasting and Control, second edition. San Francisco: Holden-Day, Inc.

Cox, D.R. and Lewis, P.A.W. (1966). The Statistical Analysis of Series of Events. London: Methuen.

Nelson, C.R. (1973). Applied Time Series Analysis for Managerial Forecasting. San Francisco: Holden-Day, Inc.

9.2 Categorical and Qualitative Data Analysis

This is the analysis of counts. Many experiments contain qualitative and nonmetric variables, e.g. when analyzing the number of smokers and non-smokers beyond male and female persons. Smoking behaviour and sex have no natural numeric values associated with them. Used statistical methods for such data are

- Crosstabulation
- Contingency tables
- Goodness of fit tests
- Log linear models
- Logistic regression
- Correspondence analysis

Literature:

Agresti, A. (1984). Analysis of ordinal categorical data. New York: Wiley-Interscience.

Bishop, Y.M.M., Fienberg, S.E. and Holland, P.W. (1975). Discrete Multivariate Analysis: Theory and Practice. Cambridge, MA: MIT Press.

Haberman, S.J. (1978). Analysis of Qualitative Data, Vol. 1: Introductory Topics. New York: Academic Press.

Haberman, S.J. (1979). Analysis of Qualitative Data, Vol. 2: New Developments. New York: Academic Press.

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9.3 Experimental Designs and ANOVA

This is a wide field containing among others:

- One-Way ANOVA
- Multifactor ANOVA
- Analysis of Nested Designs
- Full and Fractional Designs
- Response Surface Analysis
- Covariance Analysis
- and a lot more.

Literature:

Box, G.E.P., Hunter, W.G. and Hunter, J.S. (1978). Statistics for Experimenters. New York: Wiley.

Neter, J. and Wassermann, W. (1974). Applied Linear Statistical Models. Homewood, Illinois: Richard E. Irvin, Inc.

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9.4 Multivariate Methods

This title involves the analysis of multivariate data. It is not appropriate to analyze multivariate data univariate, because correlations among the different variables are not taken into account.

Statistical Methods:

- Correlation Analysis
- Covariance Analysis
- Multiple Regression
- Principal Components
- Factor Analysis
- Discriminant Analysis
- Cluster Analysis
- Canonical Correlations
- Multidimensional Scaling
- and a lot more.

Literature:

Sieber, G.A.F. (1984). Multivariate Observations. New York: Wiley.

Flury, B. and Riedwyl, H. (1988). Multivariate Statistics. A practical approach. London, New York: Chapman and Hall.

Johnson, R.A. and Wichern, D.W. (1982). Applied Multivariate Statistical Analysis. London: Prentice-Hall.

Morrison, D.F. (1976, 2nd edition). Multivariate Statistical Methods. New York: McGraw-Hill.

Everitt, B.S. (1980). Cluster Analysis, 2nd edition, London: Heinemann Education Books, Ltd.

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9.5 Nonparametric Methods

Parametric methods base on certain assumptions on the data (e.g. normality of residuals in linear regression, normality of the observations in testing etc.). If these assumptions do not hold, it is often more efficient to use methods that do not use a particular underlying distribution function, so called nonparametric methods (e.g. the Wilcoxon Test known for Section 5.2.4 is a nonparametric test). Some nonparametric procedures exist for:

- Tests of binary sequences
- Tests for randomness
- Tests for location
- Comparison of 2 samples
- Rank correlation analysis
- Goodness of fit tests

Literature:

Conover, W.J. (1980). Practical nonparametric statistics, 2nd edition. New York: John Wiley and Sons, Inc.

Lehmann, E.L. (1975). Nonparametrics. San Francisco: Holden-Day, Inc.

Gibbons, J.D. (1976). Nonparametric Methods for Quantitative Analysis. New York: Holt, Rinehart and Winston.

Hollander, M. and Wolfe, D.A. (1973). Nonparametric Statistical Methods. New York: Wiley.

9.6 Bootstrap and Jack-knife Methods

In dealing with complicated functions of data, it is mostly not possible to derive the underlying distribution function. Bootstrap and Jackknife sometimes provide the possibility to derive the distribution functions, statistical measures and a lot more at such complicated data.

Literature:

Efron, B. (1982). The Jackknife, the Bootstrap and Other Resampling Plans. Philadelphia: Soc. for Industrial and Applied Math.

9.7 Simulation and Monte Carlo Method

By use of simulation and Monte Carlo methods it is often possible to investigate and analyse complex functions of random variables or systems of events. Simulation incorporates also the generation of random variables.

Literature:

Fishman, G.S. (1978). Principles of Discrete Event Simulation. New York: John Wiley & Sons, Inc.

Rubinstein, R.Y. (1981). Simulation and the Monte Carlo Method. New York: John Wiley & Sons, Inc.

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9.8 General Literature

Seber, G.A.F. (1984). Multivariate Observations. New York: Wiley.

Belsley, D.A. Kuh, E. and Welsh, R.E. (1980). Regression Diagnostic: Identifying Influential Data and Sources of Collinearity. New York: Wiley.

Chambers, J.M., Cleveland, W.S., Kleiner, B. and Tukey, P.A. (1983). Graphical Methods for Data Analysis. Boston: Duxbury Press.

Flury, B. (1980). Construction of an asymmetrical face to represent multivariate data graphically. Tech. Rep. No. 3, University of Berne, Dept of Statistics.

Schupbach, M. (1984). ASYMFACE - Asymmetrical Faces on IBM-PC. Tech.Rep. No.16. University of Berne, Dept of Statistics.

Draper, N.R. and Smith, H. (1981), 2nd ed.). Applied Regression Analysis. New York: Wiley.

Grant, E.L. and Leavenworth, R.S. (1980). Statistical Quality Control, fifth ed. New York: McGraw Hill.

Montgomery, D.C. (1985). Introduction to Statistical Quality Control. New York: Wiley.

Recommended literature pp. 90 ff

Huff, D. (1974). How to lie with statistics.

Miller, R.G. (1981). Simultaneous Statistical Inference, 2nd ed. New York: Wiley.

Morrison, D.F. (1983). Applied Linear Models. Englewood Cliffs, NJ: PrenticeHall Inc.

10. STATISTICAL PROGRAM PACKAGES

Very complete and sophisticated packages are (all programs are available for PC's under DOS):

BMDP	Statistical Software Ltd., Cork Technology Park. Cork, Ireland (phone: 021-542722)
SAS:	SAS Inst. GmbH, Cary NC, USA (phone: (919)467-8000)
SPSS	SPSS Inc., Chicago Il, USA (phone: (312)329-3300)
SYSTAT	SYSTAT Inc., Evanston Il, USA (phone: (312)864-5670)
STATGRAPHICS	STSC Inc., Rockville MD, USA (phone: (301)984-5000)

A review of 49 statistical packages is given in the PC magazine, Vol. 8, No. 5, March 1989.

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Appendix I

Example A1:

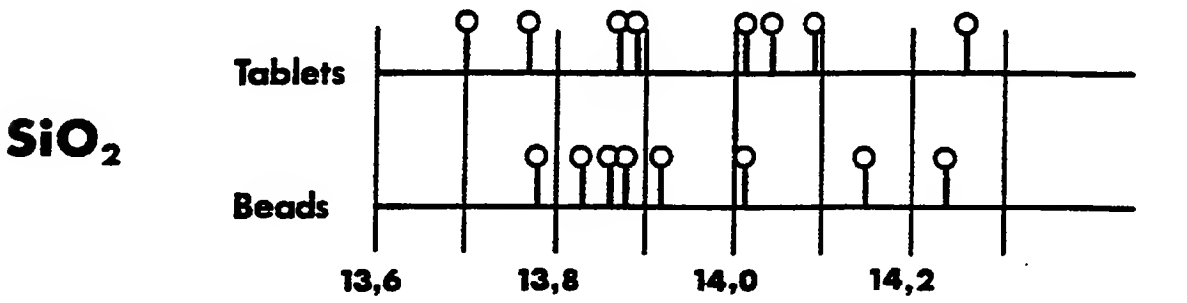
In the application of XRF-analysis of raw meal it was required to know to what extent the results are dependent on sample preparation, by pressed powder tablets and by fusion with Litetraborate, respectively.

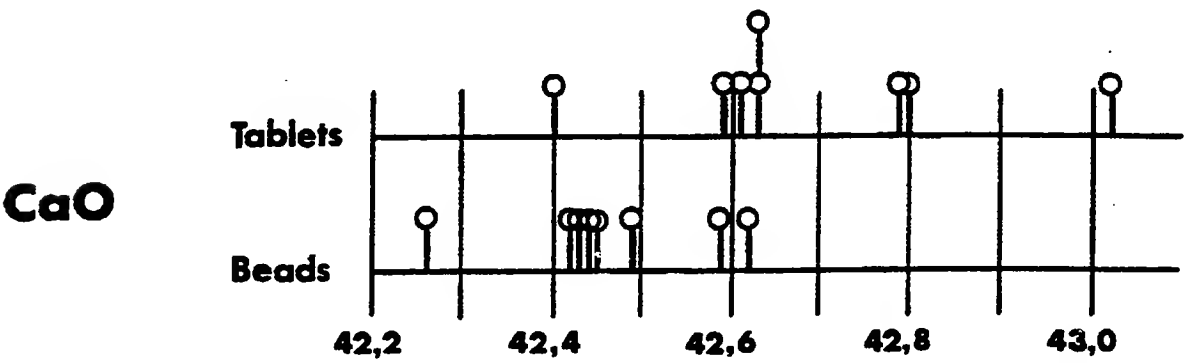
With 8 preparations of tablets and beads of the same raw meal the following results for CaO and $Si0_2$ were obtained.

CaO	
Tablets	**Beads**
42.63	42.42
42.59	42.44
42.63	42.45
42.80	42.42
43.02	42.59
42.61	42.49
42.40	42.62
42.79	42.26

SiO2	
tablets	**beads**
13.87	14.15
14.01	13.86
14.04	13.92
14.09	13.83
14.26	14.01
13.70	13.78
13.89	14.24
13.77	13.88

We are interested in differences between the preparation methods. To get a quick survey on the data we mark every observation on the measurement scale for tablets and beads respectively.





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Looking at the graph we suppose a significant difference of CaO-results between beads and tablets. For Si 02 results no difference is visible.

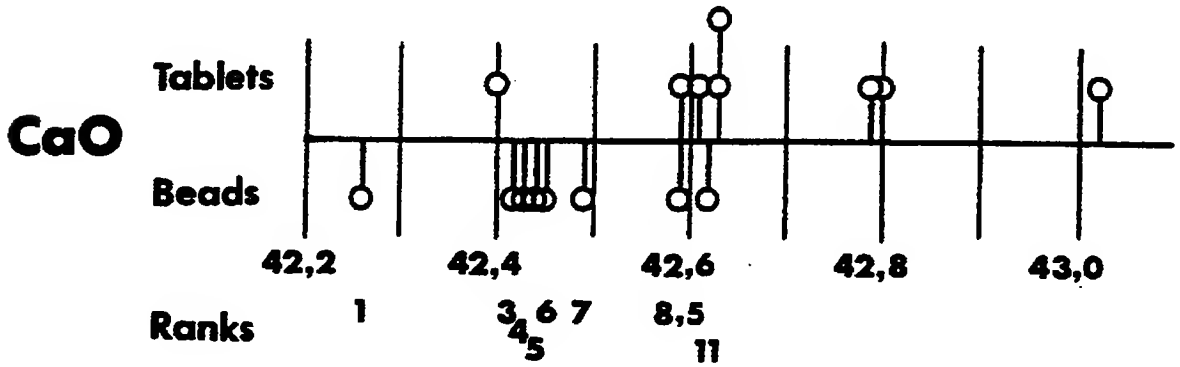
The final decision whether the differences are significant is made with an appropriate test.

Test situation:

- We test the hypothesis that the mean results of both preparation methods are the same H_0: $\mu_{tablets} = \mu_{beads}$
- Two independent samples with small sample sizes N = 8
- two-sided test: we have no idea in what direction a possible difference may occur before the observations are taken.

Test procedure: Wilcoxon-test at $\alpha = 5\%$ level

As an alternative representation we can mark the data on both side of one line. In this case ranks of the observations can be read directly from the graph.



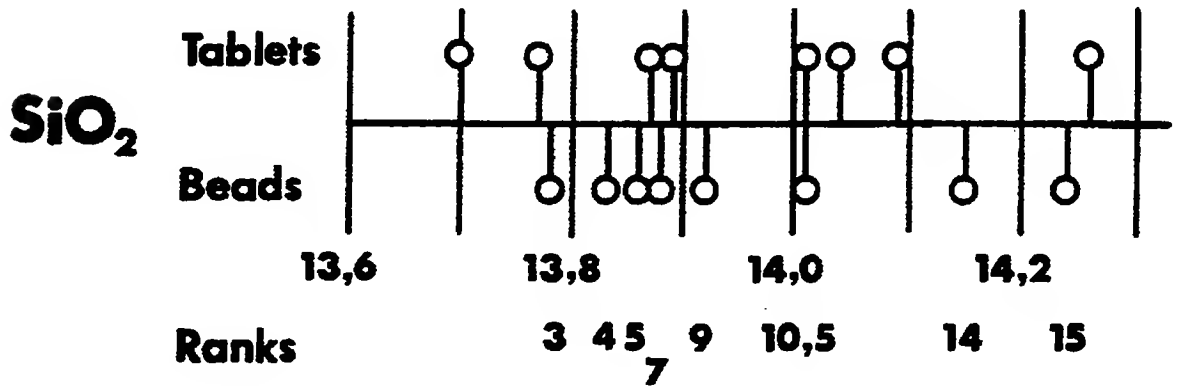
The ranksum of beads is R = 45.5

$$W = 2 \cdot 45.5 - 8 \cdot 17 = -45$$

$|W| = 45 \Rightarrow 38 = W_{0.975}$ (from Table A-8, Appendix III).

Decision:

The difference of CaO results between tablets and beads is significant.



$$|W| = 1 < 38 = W_{0.975}$$

There is no significant difference for SiO_2 results.

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Example A2:

Control of homogenization efficiency.

In intervals of 30 minutes, 20 samples are taken from the raw ma serial stream both before and after homogenization.

Measured values: $CaCO_3$ content

Total capacity of homogenization silo: 1000 tons

Sample No.	$CaCO_3$ before homogenization	$CaCO_3$ after homogenization
1	76.4	76.9
2	76.7	77.0
3	77.5	77.0
4	77.5	76.8
5	76.5	76.9
6	76.0	76.8
7	76.8	76.9
8	77.0	77.0
9	77.1	77.0
10	77.2	76.8
11	77.2	76.9
12	77.1	76.9
13	76.9	76.9
14	76.7	77.0
15	76.8	77.0
16	77.0	76.0
17	77.1	76.8
18	77.0	76.9
19	76.7	77.0
20	76.8	76.9

The graph is chronological order (time-plot) shows a systematical variation of $CaCO_3$ content before homogenization. Except for one outlying value the variation is small after homogenization.

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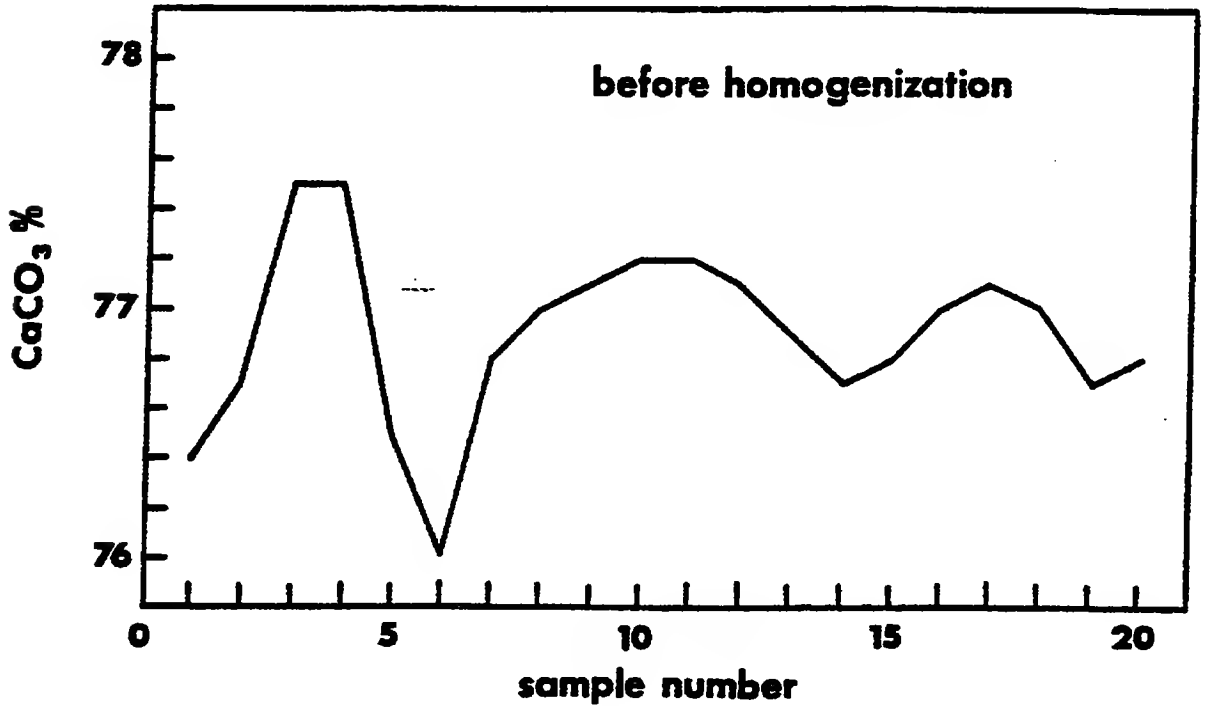
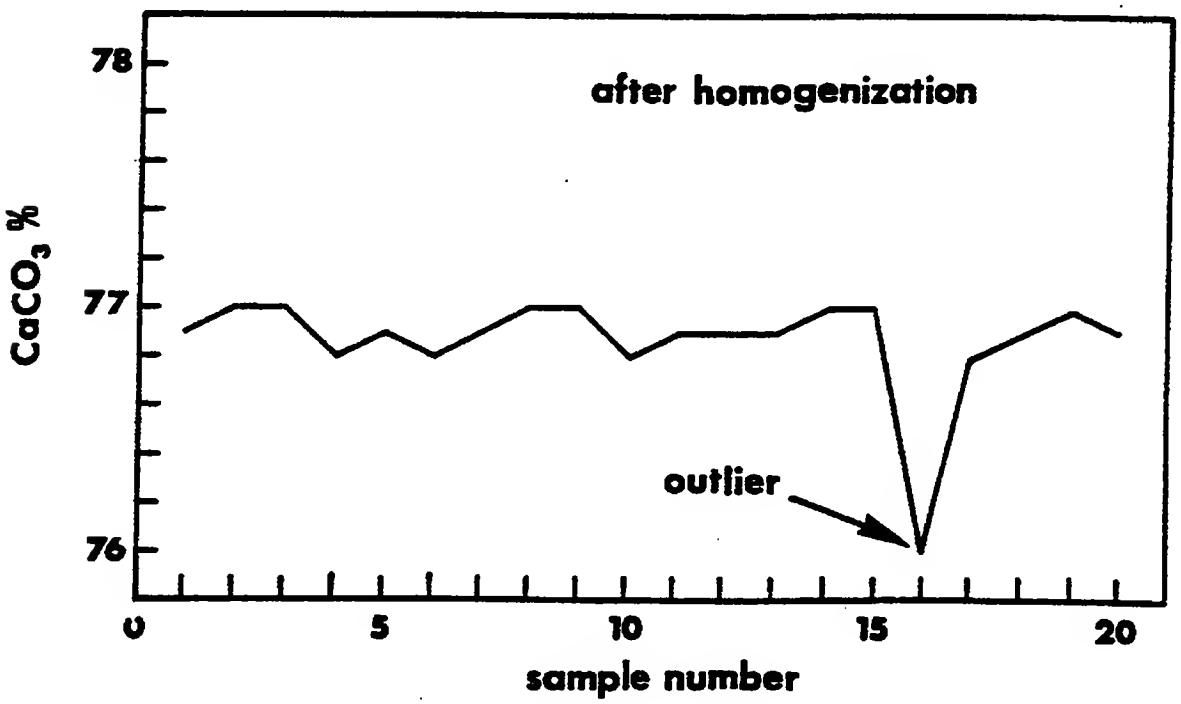
Before Homogenization



After Homogenization



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Using the Dixon criterion to check for outliers we compute

$$r_{22} = \frac{x_{(3)} - x_{(1)}}{x_{(n-2)} - x_{(1)}} = \frac{76.8 - 76.0}{77.0 - 76.0} = 0.8$$

with $x_{(1)}$ the smallest, $X_{(3)}$ the third smallest and $X_{(n-2)}$ the largest value.

Since $r_{22} = 0.8 > r_{0.995} = 0.562$ (Table A-2, Appendix III) the extreme value is considered to be a real outliner (with significance level $\alpha = 1\%$). It is deleted for further analysis.

Statistical data description

	$CaCO_3$ before homogenization	**$CaCO_3$ after homogenization**	
n	20	(20)	19
$\bar{x}$	76.9	(76.87)	76.92
$\tilde{x}$	76.95	(76.9)	76.9
s	0.36	(0.22)	0.076 one outlier deleted: 76.0
v	0.0046	(0.0028)	0.0010
x_{max}	77.5	(77.0)	77.0
x_{min}	76.0	(76.0)	76.8
R	1.5	(1.0)	0.2

Comment:

Mean and median are almost equal. The distribution of observations seems to be therefore symmetrical about the mean either before and after homogenization. Due to homogenization the standard deviation is reduced from 0.36 to 0.08 corresponding to a factor of four to five.

Note:

Elimination of the outliner reduces the standard deviation from 0.22. to 0.08 !

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Example A3:

In order to control mortar strength a national control laboratory takes every month a sample of cement in a plant. It was supposed that these control results tend to be lower than results of internal quality control.

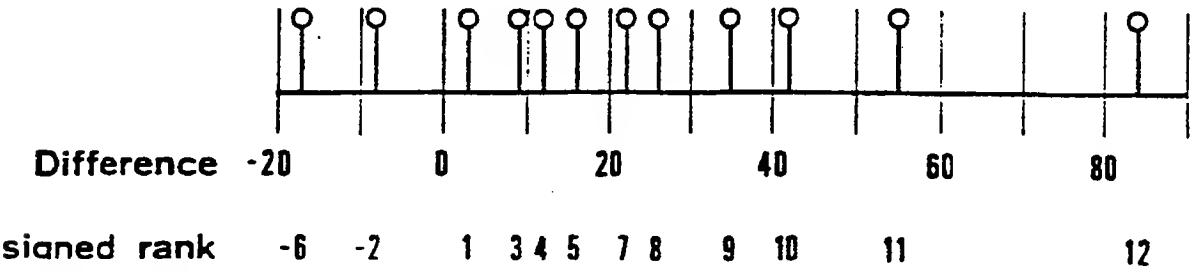
To find out a suspected syste ma tic error between results of control office and plant laboratory during a year the samples were measured in both laboratories:

	Plant x	**Control y**	**Difference d = x - y**
January	548	526	22
February	540	524	16
March	574	591	-17
Apri1	469	477	-8
May	540	531	9
June	515	431	84
July	520	485	35
August	531	476	55
September	530	490	42
October	464	452	12
November	524	498	26
December	519	516	3

The strength values are paired. Each pair of observation is concerned with the same sample. To test for a significant difference we use the singed-rank test at a significance level of $\alpha = 5\%$ (two-sided)

$$H_0: \mu = 0$$

Plot of differences $d_i = x_i - y_i$



Sum of signed ranks:

$$T = (-6) + (-2) + 1+ 3+ 4+ 5+ 7+ 8+ 9+ 10+ 11+ 12 = 62$$

Since $T = 62 > 52 = T_{0\,975}$ we decide that the difference between results is systematic.

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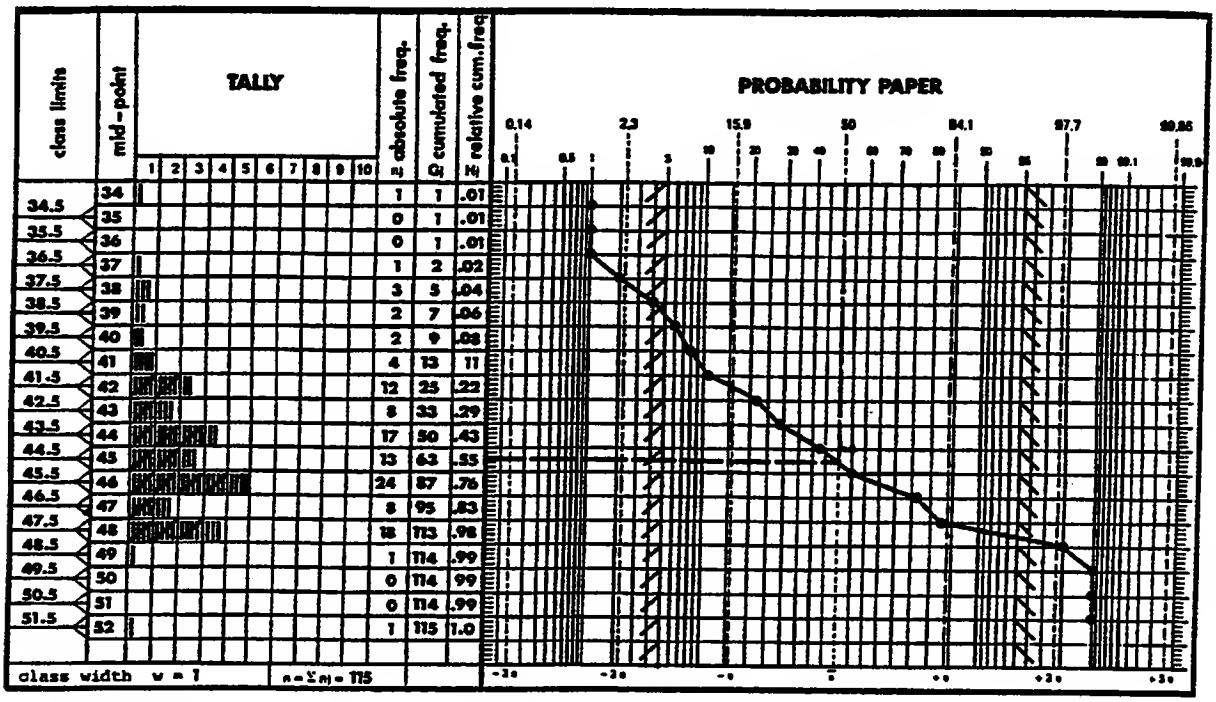
Example A4:

The following data represent 115 measurements of Schmidt-haamer-strength (abutment of a bridge).

44	37	46	45	46	45	46	46	46	48	46	46	48	44
42	45	45	46	48	47	45	46	41	46	43	42	44	45
40	45	42	44	38	39	39	45	44	44	46	48	47	48
48	46	48	42	48	45	45	48	52	43	38	43	42	44
46	44	44	42	34	42	42	42	41	46	45	48	46	46
48	47	46	44	45	40	47	41	42	47	43	45	43	43
44	44	41	44	48	46	47	48	48	46	44	48	44	49
46	48	44	43	43	46	47	46	48	46	46	47	38	42
48	44	42											

A detailed interpretation of these data requires their representation in a frequency table and histogram. With the probability paper we check the data for normality.

The cumulative frequency curve plotted on the probability paper is not linear and the distribution is therefore not normal.



Histogram and tally show that even values of Schmidt-hammer strength are more frequent than odd values. This seems to be caused by a reading error of scale. Probably only even values are marked on the measurement scale.

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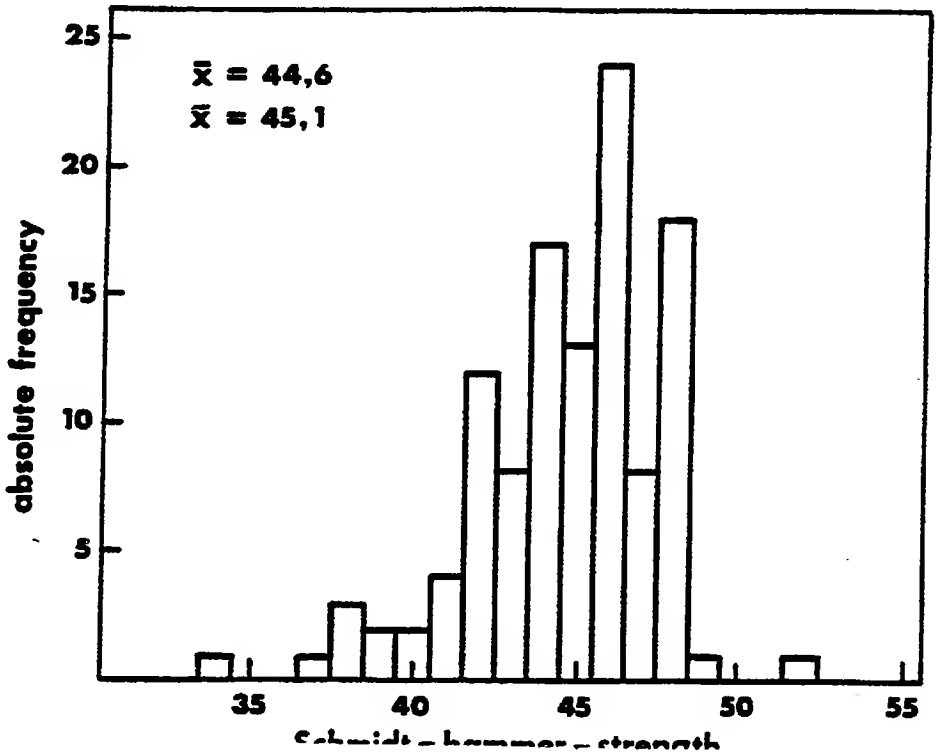


EXTENDED PROBABILITY PAPER

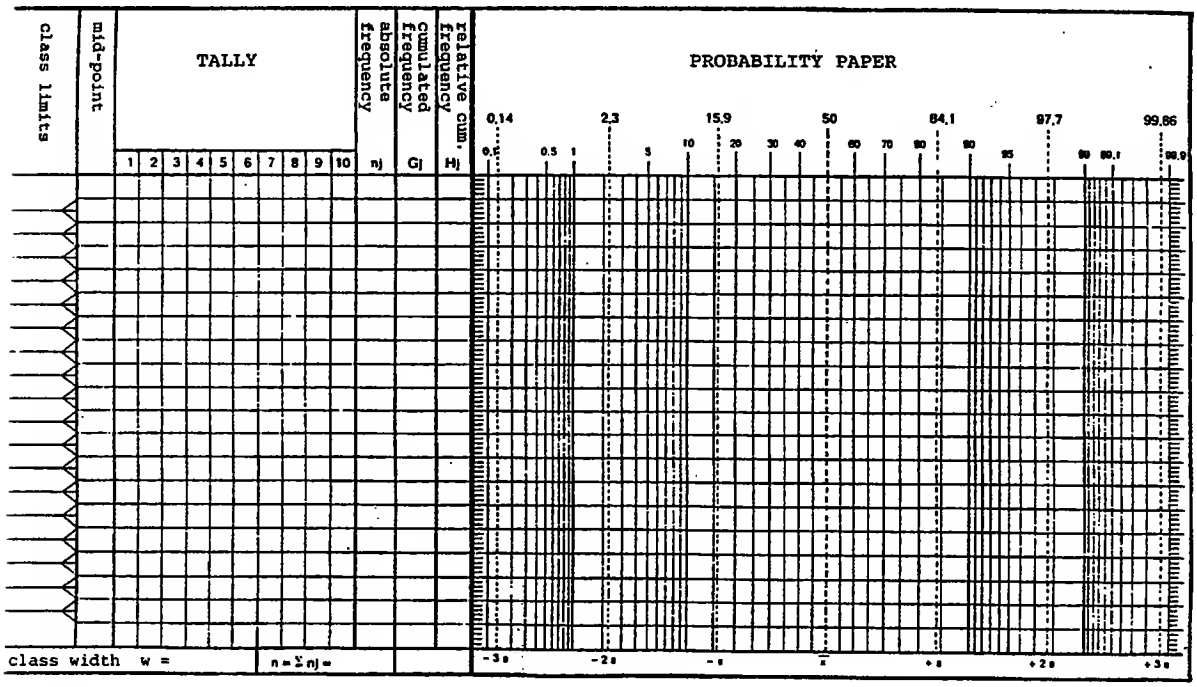
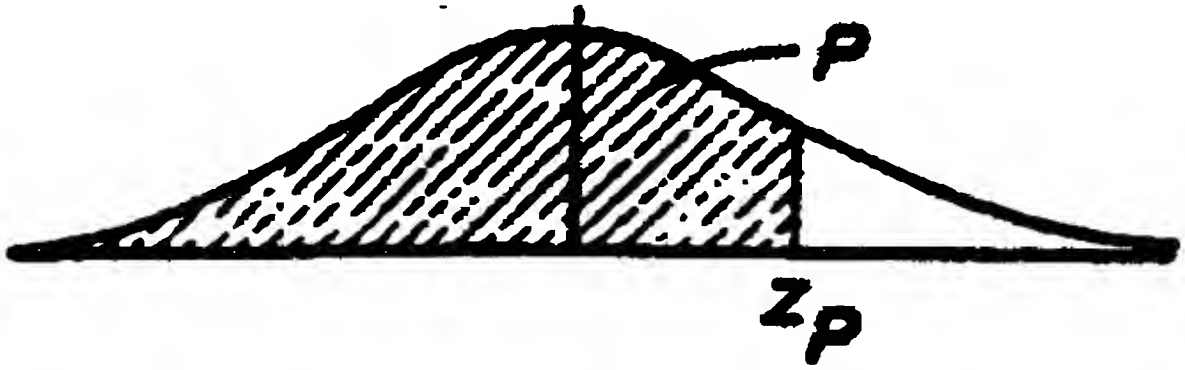


Table A-1 Standard Normal Distribution - Values of P



Values of P corresponding to z_p for the normal curve.

z is the standard normal variable. The value of P $-z_p$ equals one minus the value of P for $+z_p$, e.g. the P for -1.62 equals 1-0.9474 = .0526.

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z_p	.00	.01	.02	.03	.04	.05	.06	.07	.08	.09
.0	.5000	.5040	.5080	.5120	.5160	.5199	.5239	.5279	.5319	.5359
.1	.5398	.5438	.5478	.5517	.5557	.5596	.5636	.5675	.5714	.5753
.2	.5793	.5832	.5871	.5910	.5948	.5987	.6026	.6064	.6103	.6141
.3	.6179	.6217	.6255	.6293	.6331	.6368	.6406	.6443	.6480	.6517
.4	.6554	.6591	.6628	.6664	.6700	.6736	.6772	.6808	.6844	.6879
.5	.6915	.6950	.6985	.7019	.7054	.7088	.7123	.7157	.7190	.7224
.6	.7257	.7291	.7324	.7357	.7389	.7422	.7454	.7486	.7517	.7549
.7	.7580	.7611	.7642	.7673	.7704	.7734	.7764	.7794	.7823	.7852
.8	.7881	.7910	.7939	.7967	.7995	.8023	.8051	.8078	.8106	.8133
.9	.8159	.8186	.8212	.8238	.8264	.8289	.8315	.8340	.8365	.8389
1.0	.8413	.8438	.8461	.8485	.8508	.8531	.8554	.8577	.8599	.8621
1.1	.8643	.8665	.8686	.8708	.8729	.8749	.8770	.8790	.8810	.8830
1.2	.8849	.8869	.8888	.8907	.8925	.8944	.8962	.8980	.8997	.9015
1.3	.9032	.9049	.9066	.9082	.9099	.9115	.9131	.9147	.9162	.9177
1.4	.9192	.9207	.9222	.9236	.9251	.9265	.9279	.9292	.9306	.9319
1.5	.9332	.9345	.9357	.9370	.9382	.9394	.9406	.9418	.9429	.9441
1.6	.9452	.9463	.9474	.9484	.9495	.9505	.9515	.9525	.9535	.9545
1.7	.9554	.9564	.9573	.9582	.9591	.9599	.9608	.9616	.9625	.9633
1.8	.9641	.9649	.9656	.9664	.9671	.9678	.9686	.9693	.9699	.9706
1.9	.9713	.9719	.9726	.9732	.9738	.9744	.9750	.9756	.9761	.9767
2.0	.9772	.9778	.9783	.9788	.9793	.9798	.9803	.9808	.9812	.9817
2.1	.9821	.9826	.9830	.9834	.9838	.9842	.9846	.9850	.9854	.9857
2.2	.9861	.9864	.9868	.9871	.9875	.9878	.9881	.9884	.9887	.9890
2.3	.9893	.9896	.9898	.9901	.9904	.9906	.9909	.9911	.9913	.9916
2.4	.9918	.9920	.9922	.9925	.9927	.9929	.9931	.9932	.9934	.9936
2.5	.9938	.9940	.9941	.9943	.9945	.9946	.9948	.9949	.9951	.9952
2.6	.9953	.9955	.9956	.9957	.9959	.9960	.9961	.9962	.9963	.9964
2.7	.9965	.9966	.9967	.9968	.9969	.9970	.9971	.9972	.9973	.9974
2.8	.9974	.9975	.9976	.9977	.9977	.9978	.9979	.9979	.9980	.9981
2.9	.9981	.9982	.9982	.9983	.9984	.9984	.9985	.9985	.9986	.9986
3.0	.9987	.9987	.9987	.9988	.9988	.9989	.9989	.9989	.9990	.9990
3.1	.9990	.9991	.9991	.9991	.9992	.9992	.9992	.9992	.9993	.9993
3.2	.9993	.9993	.9994	.9994	.9994	.9994	.9994	.9995	.9995	.9995
3.3	.9995	.9995	.9995	.9996	.9996	.9996	.9996	.9996	.9996	.9997
3.4	.9997	.9997	.9997	.9997	.9997	.9997	.9997	.9997	.9997	.9998

Special Values

P	.001	.005	.010	.025	.050	.100
z_p	−3.090	−2.576	−2.326	−1.960	−1.645	−1.282

P	.999	.995	.990	.975	.950	.900
z_p	3.090	2.576	2.326	1.960	1.645	1.282

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Table A-2 Critical values for the Dixon Criterion

Statistic	Number of Observations, n	Upper Percentiles						
		.70	.80	.90	.95	.98	.99	.995
r_{10}	3	.684	.781	.886	.941	.976	.988	.994
	4	.471	.560	.679	.765	.846	.889	.926
	5	.373	.451	.557	.642	.729	.780	.821
	6	.318	.386	.482	.560	.644	.698	.740
	7	.281	.344	.434	.507	.586	.637	.680
r_{11}	8	.318	.385	.479	.554	.631	.683	.725
	9	.288	.352	.441	.512	.587	.635	.677
	10	.265	.325	.409	.477	.551	.597	.639
r_{21}	11	.391	.442	.517	.576	.638	.679	.713
	12	.370	.419	.490	.546	.605	.642	.675
	13	.351	.399	.467	.521	.578	.615	.649
r_{22}	14	.370	.421	.492	.546	.602	.641	.674
	15	.353	.402	.472	.525	.579	.616	.647
	16	.338	.386	.454	.507	.559	.595	.624
	17	.325	.373	.438	.490	.542	.577	.605
	18	.314	.361	.424	.475	.527	.561	.589
	19	.304	.350	.412	.462	.514	.547	.575
	20	.295	.340	.401	.450	.502	.535	.562
	21	.287	.331	.391	.440	.491	.524	.551
	22	.280	.323	.382	.430	.481	.514	.541
	23	.274	.316	.374	.421	.472	.505	.532
	24	.268	.310	.367	.413	.464	.497	.524
	25	.262	.304	.360	.406	.457	.489	.516

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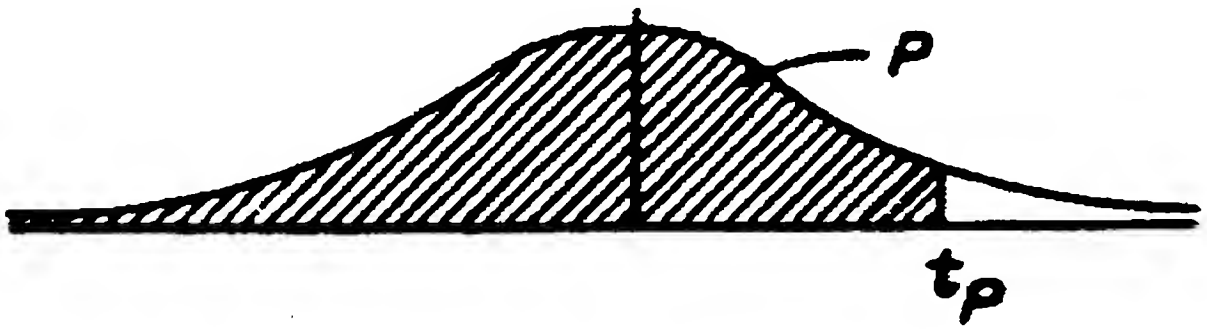
Table A-3 Critical values for Outlier-tests, (large sample size n)

n	probability 1-α										n
	0.5%	1.0%	2.5%	5.0%	10.0%	90%	95%	97.5%	99%	99.5%	
20	2,95	3,01	3,10	3,18	3,29	4,32	4,49	4,63	4,79	4,91	20
30	3,22	3,27	3,37	3,46	3,58	4,70	4,89	5,06	5,25	5,39	30
40	3,41	3,46	3,57	3,66	3,79	4,96	5,15	5,34	5,54	5,69	40
50	3,57	3,61	3,72	3,82	3,94	5,15	5,35	5,54	5,77	5,91	50
60	3,69	3,74	3,85	3,95	4,07	5,29	5,50	5,70	5,93	6,09	60
80	3,88	3,93	4,05	4,15	4,27	5,51	5,73	5,93	6,18	6,35	80
100	4,02	4,09	4,20	4,31	4,44	5,68	5,90	6,11	6,36	6,54	100
150	4,30	4,36	4,47	4,59	4,72	5,96	6,18	6,39	6,64	6,84	150
200	4,50	4,56	4,67	4,78	4,90	6,15	6,38	6,59	6,85	7,03	200
500	5,06	5,13	5,25	5,37	5,49	6,72	6,94	7,15	7,42	7,60	500
1000	5,50	5,57	5,68	5,79	5,92	7,11	7,33	7,54	7,80	7,99	1000

Table A-4 Critical values of the t-distribution



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df	$t_{.90}$	$t_{.95}$	$t_{.975}$	$t_{.99}$	$t_{.995}$
1	3.078	6.314	12.706	31.821	63.657
2	1.886	2.920	4.303	6.965	9.925
3	1.638	2.353	3.182	4.541	5.841
4	1.533	2.132	2.776	3.747	4.604
5	1.476	2.015	2.571	3.365	4.032
6	1.440	1.943	2.447	3.143	3.707
7	1.415	1.895	2.365	2.998	3.499
8	1.397	1.860	2.306	2.896	3.355
9	1.383	1.833	2.262	2.821	3.250
10	1.372	1.812	2.228	2.764	3.169
11	1.363	1.796	2.201	2.718	3.106
12	1.356	1.782	2.179	2.681	3.055
13	1.350	1.771	2.160	2.650	3.012
14	1.345	1.761	2.145	2.624	2.977
15	1.341	1.753	2.131	2.602	2.947
16	1.337	1.746	2.120	2.583	2.921
17	1.333	1.740	2.110	2.567	2.898
18	1.330	1.734	2.101	2.552	2.878
19	1.328	1.729	2.093	2.539	2.861
20	1.325	1.725	2.086	2.528	2.845
21	1.323	1.721	2.080	2.518	2.831
22	1.321	1.717	2.074	2.508	2.819
23	1.319	1.714	2.069	2.500	2.807
24	1.318	1.711	2.064	2.492	2.797
25	1.316	1.708	2.060	2.485	2.787
26	1.315	1.706	2.056	2.479	2.779
27	1.314	1.703	2.052	2.473	2.771
28	1.313	1.701	2.048	2.467	2.763
29	1.311	1.699	2.045	2.462	2.756
30	1.310	1.697	2.042	2.457	2.750
40	1.303	1.684	2.021	2.423	2.704
60	1.296	1.671	2.000	2.390	2.660
120	1.289	1.658	1.980	2.358	2.617
∞	1.282	1.645	1.960	2.326	2.576

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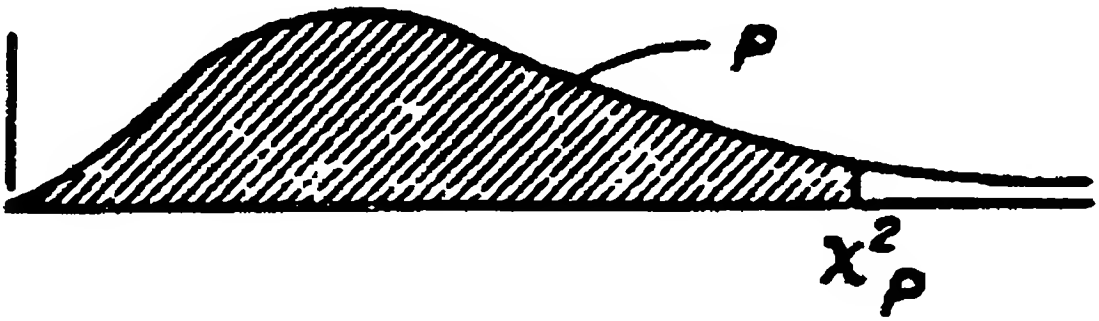
Table A-5 Critical values of the x^2-distribution



Values of χ^2_p corresponding to P

df	$x^2_{.005}$	$x^2_{.01}$	$x^2_{.025}$	$x^2_{.05}$	$x^2_{.10}$	$x^2_{.90}$	$x^2_{.95}$	$x^2_{.975}$	$x^2_{.99}$	$x^2_{.995}$
1	.000039	.00016	.00098	.0039	.0158	2.71	3.84	5.02	6.63	7.88
2	.0100	.0201	.0506	.1026	.2107	4.61	5.99	7.38	9.21	10.60
3	.0717	.115	.216	.352	.584	6.25	7.81	9.35	11.34	12.84
4	.207	.297	.484	.711	1.064	7.78	9.49	11.14	13.28	14.86
5	.412	.554	.831	1.15	1.61	9.24	11.07	12.83	15.09	16.75
6	.676	.872	1.24	1.64	2.20	10.64	12.59	14.45	16.81	18.55
7	.989	1.24	1.69	2.17	2.83	12.02	14.07	16.01	18.48	20.28
8	1.34	1.65	2.18	2.73	3.49	13.36	15.51	17.53	20.09	21.96
9	1.73	2.09	2.70	3.33	4.17	14.68	16.92	19.02	21.67	23.59
10	2.16	2.56	3.25	3.94	4.87	15.99	18.31	20.48	23.21	25.19
11	2.60	3.05	3.82	4.57	5.58	17.28	19.68	21.92	24.73	26.76
12	3.07	3.57	4.40	5.23	6.30	18.55	21.03	23.34	26.22	28.30
13	3.57	4.11	5.01	5.89	7.04	19.81	22.36	24.74	27.69	29.82
14	4.07	4.66	5.63	6.57	7.79	21.06	23.68	26.12	29.14	31.32
15	4.60	5.23	6.26	7.26	8.55	22.31	25.00	27.49	30.58	32.80
16	5.14	5.81	6.91	7.96	9.31	23.54	26.30	28.85	32.00	34.27
18	6.26	7.01	8.23	9.39	10.86	25.99	28.87	31.53	34.81	37.16
20	7.43	8.26	9.59	10.85	12.44	28.41	31.41	34.17	37.57	40.00
24	9.89	10.86	12.40	13.85	15.66	33.20	36.42	39.36	42.98	45.56
30	13.79	14.95	16.79	18.49	20.60	40.26	43.77	46.98	50.89	53.67
40	20.71	22.16	24.43	26.51	29.05	51.81	55.76	59.34	63.69	66.77
60	35.53	37.48	40.48	43.19	46.46	74.40	79.08	83.30	88.38	91.95
120	83.85	86.92	91.58	95.70	100.62	140.23	146.57	152.21	158.95	163.64

For large degrees of freedom,

$$x^2_P = \tfrac{1}{2}(z_P + \sqrt{2m - 1})^2 \text{ approximately,}$$

where m = degrees of freedom and z_P is given in Table A-1

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Table A-6 Values of $\lambda_{1-\alpha}$ for confidence limits (Range method)

n \ α	5%	2,5%	1%	0,5%	0,1%	0,05%
2	3.157	6,353	15,910	31,828	159,16	318,31
3	0,885	1,304	2,111	3,008	6.77	9,58
4	.529	0,717	1,023	1,316	2,29	2,85
5	,388	,507	0,685	0,843	1,32	1,58
6	0,312	0,399	0,523	0,628	0,92	1,07
7	,263	,333	.429	,507	.71	0,82
8	,230	,288	.366	.429	,59	,67
9	,205	,255	,322	,374	,50	,57
10	,186	,230	,288	,333	,44	,50
11	0,170	0,210	0,262	0,302	0.40	0.44
12	,158	,194	.241	,277	.36	,40
13	,147	,181	,224	,256	,33	,37
14	,138	,170	,209	,239	,31	,34
15	,131	,160	,197	,224	,29	,32
16	0,124	0,151	0,186	0,212	0,27	0,30
17	,118	,144	,177	,201	,26	,28
18	,113	,137	.168	,191	,24	,26
19	.108	,131	.161	.182	,23	,25
20	,104	,126	.154	,175	,22	,24

[1] Nach E. Lord. The use of range in place of standard deviation in the *t*-test. Biometrika 34, 1947, S. 41.

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Table A-7 Critical values of the signed-rank test

number of differences not equal zero: m	$T_{0.95}$	$T_{0.975}$
5	7	-
6	8	10
7	11	12
8	13	15
9	14	17
10	17	19
11	20	23
12	22	26
13	24	28
14	27	31
15	30	35
16	33	39
17	35	42
18	38	45
19	42	49
20	45	53
21	48	57
22	51	60
23	55	65
24	59	69
25	62	73

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Table A-8 Critical values of the Wilcoxon-test

n_2 \ size of smaller sample n_1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
20	32	38	44	50	56	62	66	72	76	82	86	92	96	100	106	110	114	120	124
19	30	37	42	49	54	59	64	69	74	79	84	87	92	97	102	105	110	115	
18	28	36	40	46	52	56	62	66	70	76	80	84	88	94	98	102	106		
17	28	33	38	45	50	53	58	63	68	73	76	81	84	89	94	97			
16	26	32	36	42	46	52	56	60	64	68	72	78	82	86	90				
15	24	31	36	39	44	49	54	57	62	65	70	73	78	81					
14	22	28	34	38	42	46	50	54	58	62	66	70	74						
13	22	27	32	35	40	43	48	51	56	59	62	67							
12	20	26	30	34	38	42	44	48	52	56	60								
11	20	23	28	31	34	39	42	45	48	53									
10	18	22	26	28	32	36	40	42	46										
9	16	21	24	27	30	33	36	39											
8	14	18	22	24	28	30	34												
7	14	17	20	23	26	27													
6	12	14	18	20	22														
5	10	13	16	17															
4	—	12	14																
3	—	9																	

Values of $W_{0.95}$

n_2 \ size of smaller sample n_1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
20	36	44	52	60	66	72	78	84	90	96	102	108	114	120	124	130	136	142	146
19	34	43	50	57	64	69	76	81	86	93	98	103	110	115	120	125	130	135	
18	32	40	48	54	60	66	72	78	84	88	94	100	104	110	116	120	126		
17	30	39	46	51	58	63	68	75	80	85	90	95	100	105	110	115			
16	30	36	42	50	54	60	66	70	76	82	86	90	96	100	106				
15	28	35	40	47	52	57	62	67	72	77	82	87	92	97					
14	26	32	38	44	50	54	60	64	68	74	78	82	86						
13	24	31	36	41	46	51	56	61	64	69	74	79							
12	22	28	34	38	44	48	52	56	62	66	70								
11	22	27	32	37	40	45	50	53	58	61									
10	20	24	30	34	38	42	46	50	54										
9	18	23	28	31	34	39	42	47											
8	16	20	24	28	32	36	38												
7	—	19	22	25	30	33													
6	—	16	20	24	26														
5	—	15	18	21															
4	—	—	16																
3	—	—																	

Values of $W_{0.975}$

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Table A-9 (continued) Critical values of the F-distribution : $F_{0.975}$

$F_{.975}\ (m_1, m_2)$

m_1 = degrees of freedom for numerator

m_2 = degrees of freedom for denominator

m_2 \ m_1	1	2	3	4	5	6	7	8	9	10	12	15	20	24	30	40	60	120	∞
1	647.8	799.5	864.2	899.6	921.8	937.1	948.2	956.7	963.3	968.6	976.7	984.9	993.1	997.2	1001	1006	1010	1014	1018
2	38.51	39.00	39.17	39.25	39.30	39.33	39.36	39.37	39.39	39.40	39.41	39.43	39.45	39.46	39.46	39.47	39.48	39.49	39.50
3	17.44	16.04	15.44	15.10	14.88	14.73	14.62	14.54	14.47	14.42	14.34	14.25	14.17	14.12	14.08	14.04	13.99	13.95	13.90
4	12.22	10.65	9.98	9.60	9.36	9.20	9.07	8.98	8.90	8.84	8.75	8.66	8.56	8.51	8.46	8.41	8.36	8.31	8.26
5	10.01	8.43	7.76	7.39	7.15	6.98	6.85	6.76	6.68	6.62	6.52	6.43	6.33	6.28	6.23	6.18	6.12	6.07	6.02
6	8.81	7.26	6.60	6.23	5.99	5.82	5.70	5.60	5.52	5.46	5.37	5.27	5.17	5.12	5.07	5.01	4.96	4.90	4.85
7	8.07	6.54	5.89	5.52	5.29	5.12	4.99	4.90	4.82	4.76	4.67	4.57	4.47	4.42	4.36	4.31	4.25	4.20	4.14
8	7.57	6.06	5.42	5.05	4.82	4.65	4.53	4.43	4.36	4.30	4.20	4.10	4.00	3.95	3.89	3.84	3.78	3.73	3.67
9	7.21	5.71	5.08	4.72	4.48	4.32	4.20	4.10	4.03	3.96	3.87	3.77	3.67	3.61	3.56	3.51	3.45	3.39	3.33
10	6.94	5.46	4.83	4.47	4.24	4.07	3.95	3.85	3.78	3.72	3.62	3.52	3.42	3.37	3.31	3.26	3.20	3.14	3.08
11	6.72	5.26	4.63	4.28	4.04	3.88	3.76	3.66	3.59	3.53	3.43	3.33	3.23	3.17	3.12	3.06	3.00	2.94	2.88
12	6.55	5.10	4.47	4.12	3.89	3.73	3.61	3.51	3.44	3.37	3.28	3.18	3.07	3.02	2.96	2.91	2.85	2.79	2.72
13	6.41	4.97	4.35	4.00	3.77	3.60	3.48	3.39	3.31	3.25	3.15	3.05	2.95	2.89	2.84	2.78	2.72	2.66	2.60
14	6.30	4.86	4.24	3.89	3.66	3.50	3.38	3.29	3.21	3.15	3.05	2.95	2.84	2.79	2.73	2.67	2.61	2.55	2.49
15	6.20	4.77	4.15	3.80	3.58	3.41	3.29	3.20	3.12	3.06	2.96	2.86	2.76	2.70	2.64	2.59	2.52	2.46	2.40
16	6.12	4.69	4.08	3.73	3.50	3.34	3.22	3.12	3.05	2.99	2.89	2.79	2.68	2.63	2.57	2.51	2.45	2.38	2.32
17	6.04	4.62	4.01	3.66	3.44	3.28	3.16	3.06	2.98	2.92	2.82	2.72	2.62	2.56	2.50	2.44	2.38	2.32	2.25
18	5.98	4.56	3.95	3.61	3.38	3.22	3.10	3.01	2.93	2.87	2.77	2.67	2.56	2.50	2.44	2.38	2.32	2.26	2.19
19	5.92	4.51	3.90	3.56	3.33	3.17	3.05	2.96	2.88	2.82	2.72	2.62	2.51	2.45	2.39	2.33	2.27	2.20	2.13
20	5.87	4.46	3.86	3.51	3.29	3.13	3.01	2.91	2.84	2.77	2.68	2.57	2.46	2.41	2.35	2.29	2.22	2.16	2.09
21	5.83	4.42	3.82	3.48	3.25	3.09	2.97	2.87	2.80	2.73	2.64	2.53	2.42	2.37	2.31	2.25	2.18	2.11	2.04
22	5.79	4.38	3.78	3.44	3.22	3.05	2.93	2.84	2.76	2.70	2.60	2.50	2.39	2.33	2.27	2.21	2.14	2.08	2.00
23	5.75	4.35	3.75	3.41	3.18	3.02	2.90	2.81	2.73	2.67	2.57	2.47	2.36	2.30	2.24	2.18	2.11	2.04	1.97
24	5.72	4.32	3.72	3.38	3.15	2.99	2.87	2.78	2.70	2.64	2.54	2.44	2.33	2.27	2.21	2.15	2.08	2.01	1.94
25	5.69	4.29	3.69	3.35	3.13	2.97	2.85	2.75	2.68	2.61	2.51	2.41	2.30	2.24	2.18	2.12	2.05	1.98	1.91
26	5.66	4.27	3.67	3.33	3.10	2.94	2.82	2.73	2.65	2.59	2.49	2.39	2.28	2.22	2.16	2.09	2.03	1.95	1.88
27	5.63	4.24	3.65	3.31	3.08	2.92	2.80	2.71	2.63	2.57	2.47	2.36	2.25	2.19	2.13	2.07	2.00	1.93	1.85
28	5.61	4.22	3.63	3.29	3.06	2.90	2.78	2.69	2.61	2.55	2.45	2.34	2.23	2.17	2.11	2.05	1.98	1.91	1.83
29	5.59	4.20	3.61	3.27	3.04	2.88	2.76	2.67	2.59	2.53	2.43	2.32	2.21	2.15	2.09	2.03	1.96	1.89	1.81
30	5.57	4.18	3.59	3.25	3.03	2.87	2.75	2.65	2.57	2.51	2.41	2.31	2.20	2.14	2.07	2.01	1.94	1.87	1.79
40	5.42	4.05	3.46	3.13	2.90	2.74	2.62	2.53	2.45	2.39	2.29	2.18	2.07	2.01	1.94	1.88	1.80	1.72	1.64
60	5.29	3.93	3.34	3.01	2.79	2.63	2.51	2.41	2.33	2.27	2.17	2.06	1.94	1.88	1.82	1.74	1.67	1.58	1.48
120	5.15	3.80	3.23	2.89	2.67	2.52	2.39	2.30	2.22	2.16	2.05	1.94	1.82	1.76	1.69	1.61	1.53	1.43	1.31
∞	5.02	3.69	3.12	2.79	2.57	2.41	2.29	2.19	2.11	2.05	1.94	1.83	1.71	1.64	1.57	1.48	1.39	1.27	1.00

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Recommended Literature

The following list gives a selection of books on applied statistics for further reading. The list is restricted to books which are easily comprehensible for users without profound knowledge in mathematics.

C. Chatfield	"Statistics for Technology" Chapman and Hall, London (1975), 350 pages General survey on statistical methods with good comments on the interpretation of statistical results. Special chapters on regression, design of experiments and quality control
Noether	"Introduction to Statistics: A Fresh Approach" Houghton Mifflin Company, Boston (1971) 230 pages Modern statistical methods, especially for the analysis of experiments. Good explanation of basic statistical ideas and problems based on intuition.
M.G. Natrella	"Experimental Statistics" US Department of Commerce, NBS Handbook 91 (1963) "Cookbook" with many test procedures and designs of experiments. Often it is somewhat difficult to know what procedure has to be chosen.
M.R. Spiegel	"Theory and Problems of Statistics" Schaum's Outline Series,(1961) New York 350 pages Definitions and procedures of classical statistics accompanied by many examples (875 solved problems)

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Chapter 1

Cement Hydration

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CEMENT HYDRATION

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1. INTRODUCTION

The term cement hydration applies to all the reactions of cement with water. These reactions determine to a great extent the concrete properties, as it is finally the mixture of cement with water which is the binding agent in concrete. With regard to the cement application, it is thus essential to have a basic understanding of the processes occurring during cement hydration.

To a minor extent, hydration reactions can also take place before the cement is applied in concrete during the storage of clinker and the grinding and storage of cement. Even this minor surface hydration may cause serious changes of the physical properties of cement. Table 1 gives an idea of the possible hydration of cement from clinker storage up to cement application.

Table 1: Occurrence of cement hydration

	Percentage hydration
Storage of clinker	0 to 10%
Grinding of cement	0 to 1%
Storage of cement	0 to 4%
Concrete mix	0 to 100%

The study of the mechanisms and phenomena of cement hydration has a long tradition in cement research. First basic theories to explain setting and hardening of cement have been established by Le Châtelier and Michaelis at the end of the last century. The actual more refined theories still base on the work of these two researchers. The open questions remaining are principally related to the reaction of C_3S and C_3A at early stages.

2. MAIN FEATURES OF CEMENT HYDRATION

As for any chemical reaction, main features of interest with regard to the cement hydration are the hydration reactions, the mechanisms and kinetics and the heat of hydration. In the following, a brief overview on these three aspects shall be given:

- Hydration reactions
  The nature of the hydration products is decisive for the mechanical properties of the hardened concrete. In Portland cement, the hydration is predominantly a reaction of the calcium silicates with water, producing a gel-like calcium silicate hydrate and calcium hydroxide:
  Calcium silicates + $H_2O \rightarrow$ CSH gel + $Ca(OH)_2$
  In blended cements, the reaction schemes get more complex. Nevertheless, the hydration products are in general quite similar.
- Mechanisms and kinetics
  The understanding of the mechanisms of cement hydration gives valuable indications on the reaction behaviour to be expected, including the kinetics. The knowledge of the kinetics of cement hydration and the influencing factors is very important for the concrete practice. The reaction speed of the cement hydration must be slow enough to allow the placement of concrete; after that, a rapid reaction is desired.

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- Heat of hydration
  The heat liberated during cement hydration may improve or impair the quality of the concrete. For ordinary Portland cement, the heat of hydration is typically 380 J/g at 28 days. This value is lower for the other special Portland cements with the exception of rapid hardening cement. In case of the blended cements, generally less heat is developed.

Besides the above mentioned chemical aspects, considerable changes in the physical properties of the cement are associated with the cement hydration. The principal changes are:

- Development of strength
  From the practical point of view of course the most important change. As long as the hydration proceeds, the cement is continuously gaining strength.
- Increase in specific surface
  The transformation of the low surface cement to a gel-like hydrated product of extremely high surface is one of the most striking changes of cement hydration. After complete hydration, typically a 1000 fold increase in specific surface is obtained.
- Increase in solid volume
  The total volume occupied by the hydration products is roughly twice the volume occupied originally by the unhydrated cement. That means that 1 cm^3 of cement will give more or less 2 cm^3 of hydrated cement.

More detailed and specific indications on the different aspects of cement hydration will be given in the following chapters.

3. HYDRATION REACTIONS

3.1 Hydration of the individual clinker components

3.1.1 Calcium silicates

The hydration reactions of the two calcium silicates in clinker - C_3S and C_2S - can be represented by the following chemical equations:

$2\ (3CaO \cdot SiO_2) + 6H_2O \rightarrow 3CaO \cdot 2SiO_2 \cdot 3H_2O + 3Ca(OH)_2$

Weight fraction 100 + 24 → 75 + 49

(in symbols of cement chemistry: $2C_3S + 6H \rightarrow C_3S_2H_3 + 3CH$)

$2\ (2CaO \cdot SiO_2) + 4H_2O \rightarrow 3CaO \cdot 2SiO_2 \cdot 3H_2O + Ca(OH)_2$

Weight fraction 100 + 21 → 99 + 22

(in symbols of cement chemistry: $2\ C_2S + 4H \rightarrow C_3S_2H_3 + CH$)

The reactions of both silicates are thus stochiometrically very similar and require more or less the same amount of water. The main difference between the two reactions is that C_3S produces more than twice as much calcium hydroxide as C_2S.

The principal hydration product of the calcium silicates is a gel-like or microcrystalline calcium silicate hydrate. The formula $C_3S_2H_3$ is only approximate, because the composition of this hydrate is actually variable over quite a wide range. In contrast, the calcium hydroxide formed is a crystalline material with a fixed composition.

It is believed that calcium hydroxide is not significantly contributing to strength and that the calcium silicate hydrates are in the first place responsible for the strength development. The

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importance of the calcium hydroxide being a strong base is mainly related to its effects on the passivation of the steel reinforcement, the leaching of concrete and the reaction with reactive silica and alumina from pozzolanic additives.

3.1.2 Tricalciumaluminate

The hydration of aluminates is heavily influenced by the presence of gypsum. In the absence of gypsum, the reaction of C_3A with water is very violent and leads to immediate stiffening of the paste (known as flash set) due to the rapid formation of hexagonal calcium aluminate hydrates (C_2AH_8 and C_4AH_{13}). These hydrates are not stable and later convert into the cubic form C_3AH_6, so that the final form of reaction can be written as follows:

$3CaO \cdot Al_2O_3 + 6H_2O \rightarrow 3CaO \cdot Al_2O_3 \cdot 6H_2O$

Weight fraction $100 + 40 \rightarrow 40$

(in symbols of cement chemistry: $C_3A + 6H \rightarrow C_3AH_6$)

The theoretical capacity of C_3A to combine with water is such that 100 parts by weight of C_3A combine with 40 parts by weight of H_2O; it is thus nearly double that of the silicates.

When gypsum is present in the cement, flash set can be avoided and the C_3A reacts first with the gypsum to form calcium trisulfate-aluminate hydrate (ettringite):

$3\ CaO \cdot Al_2O_3 + 3CaSO_4 + 32H_2O \rightarrow 3\ CaO \cdot Al_2O_3 \cdot 3CaSO_4 \cdot 32H_2O$

Weight fraction $66 + 100 + 137 \rightarrow 203$

(in symbols of cement chemistry: $C_3A + 3CS + 32H \rightarrow C_3A \cdot 3\ CS \cdot H_{32}$)

This reaction continues until all the available gypsum is used up. After the depletion of gypsum, ettringite is converted into the monosulfate:

$3CaO \cdot Al_2O_3 \cdot 3CaSO_4 \cdot 32H_2O + 2(3CaO \cdot Al_2O_3) + 4H_2O$

$\rightarrow (3CaO \cdot Al_2O_3 \cdot CaSO_4 \cdot 12H_2O)$

$(C_3A \cdot 3CS \cdot H_{32} + 2C_3A + 4H \rightarrow 3C_3A \cdot CS \cdot H_{12})$

Calcium aluminate hydrate continues to be formed from any remaining unhydrated C_3A and the final product of C_3A hydration is generally a monosulphate-calciumaluminate hydrate solid solution.

3.1.3 Ferrite phase

The C_4AF forms the same sequence of hydration products as does C_3A, with or without gypsum. The reactions are, however, slower; C_4AF never hydrates rapidly enough to cause flash set and gypsum retards C_4AF hydration more drastically than it does C_3A.

3.1.4 Calcium oxide and magnesium oxide

The uncombined lime (calcium oxide) and periclase (magnesium oxide), if present in large quantities in cement, may cause expansion due to a slow hydration reaction after the setting:

$CaO + H_2O \rightarrow Ca(OH)_2$; $MgO + H_2O \rightarrow Mg(OH)_2$

The actual degree of expansion depends on the state and distribution of these oxides in cement, in particular the large crystals of periclase or hard burnt free lime produce unsoundness.

3.1.5 Alkalis

Alkalis are present as alkalis sulphates or are incorporated in the main clinker phases. The first type of alkalis go readily into solution upon mixing with water and accelerate the early hydration reactions. The effect of alkalis contained in the clinker phases is not well known;

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there is at least some dissolution with time which has some effect on later hydration reactions.

The presence of alkalis in the pore solution generally does not give any special problems except when certain aggregates are used that can participate in the alkali-aggregate reaction leading to expansion and disruption of the concrete. The aggregates giving raise to such a reaction contain either reactive silica or dolomite minerals.

3.2 Mineral additives

3.2.1 Blast furnace slag

Granulated blast furnace slag does not hydrate per se in water, but in the presence of activators like lime, alkalis, sulphate and Portland cement, it shows hydraulic properties. The activators appear to function by the removal of passive surface films that act as a barrier to significant hydration. The glass in the granulated slag is the hydraulic component in the presence of Portland cement.

The hydration reactions of blast furnace slag are not easy to follow and it is thus difficult to establish clear reaction equations. Studies on slag cements have revealed that the hydration products of the slag hydration are similar to those of the calcium silicate reaction. It is believed that the main difference to Portland cement hydration is the lower C/S ratio of the calcium silicate hydrates and the reduced amount of calcium hydroxide present.

3.2.2 Pozzolanic additives

The natural and artificial pozzolans contain reactive siliceous and aluminous materials which on their own do not possess cementitious properties. They react, however, in the presence of water with dissolved calcium hydroxide released from the hydration of the Portland cement clinker. The reaction products are calcium silicate and calcium aluminate hydrates similar to those found in the hydration of Portland cement clinker. Favourable for this pozzolanic reaction seems to be the presence of alkalis.

3.2.3 Limestone filler

Finely ground limestone can react during hydration with the C_3A from the clinker to form monocarboaluminate ($C_3A \cdot CaCO_3 \cdot 11H_2O$). The presence of limestone in the cement can thus to a certain extent retard the hydration of C_3A, but it is by far less effective than gypsum and can not avoid flash set. Moreover, fine limestone filler can have an indirect influence on the hydration reaction of C_3S.

4. MECHANISM AND KINETICS

4.1 Basic theories

As mentioned in the introduction, the first theories to explain setting and hardening of cement were advanced by Le Châtelier and Michaelis. Le Châtelier attributed the development of cementing action to the passage of the anhydrous cement compounds into solution and the precipitation of the hydration products as interlocking crystals. In the theory put forward by Michaelis, cohesion is considered to be the result of the formation of a colloidal gelatinous mass. The formation of the gel may take place without the cement compounds going into solution by a direct „topochemical" or „solid state" reaction.

Both hydration mechanisms take place within in the cement paste. Cementitious reaction initially takes place by some dissolution and precipitation, whilst the hydrated material formed has only a small degree of crystalline order being gel-like and of colloidal dimensions. The colloidal theory can also explain the phenomena of swelling, shrinkage, creep and selfhealing of cracks.

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4.2 Reaction sequence during cement hydration

Hydration of cement is a sequence of overlapping chemical reactions between clinker components, mineral additives, calcium sulphate and water, leading to continuous cement paste stiffening and hardening. The hydration reactions described before proceed simultaneously at differing rates and influence each other.

A simplified schematic presentation of the hydration process of Portland cement is given in Figure 1. The three main stages which can be distinguished in this presentation are:

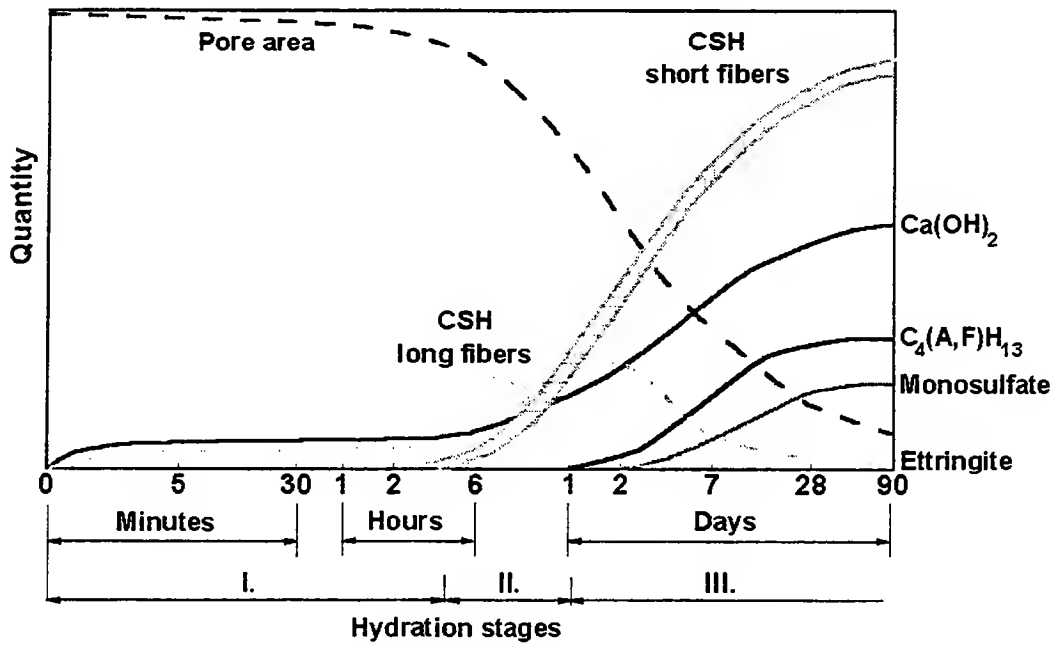
First stage

In the first stage, ettringite and calcium hydroxide are formed. The ettringite covers the aluminate particles and avoids flash set of the cement paste. After these first reactions, the so-called „dormant" period of typically four to six hours starts where no considerable further hydration takes place (certain increase in Ca^{2+} concentration).

The dormant period is usually explained by the behaviour of the C_3S, the main constituent of Portland cement. The two principal theories are based on the protective layer and the delayed nucleation concept respectively. The first ascribes the „dormant" period to the formation of a protective layer on the C_3S particles which is destroyed with time and the latter regards the retardation in C_3S reaction as the cause of a delayed nucleation of the corresponding hydration phases.

Still in the first stage, the cement paste is setting. This stiffening is attributed by some researchers to the recrystallisation of the initially finely divided ettringite. Others believe that the setting is caused by the loss in plasticity due to some first formation of calcium silicate hydrate from C_3S.

Figure 1: Schematic representation of the formation of hydrate phases in Portland cement paste



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Second stage

The second stage after the dormant period is characterised by the restarting of the hydration. Precipitation of CSH in form of long fibres and calcium hydroxide takes place on the surface of the silicate phases and the reaction proceeds rapidly for all clinker phases. This stage ends with the termination of the ettringite formation and the development of a basic matrix after about 24 hours.

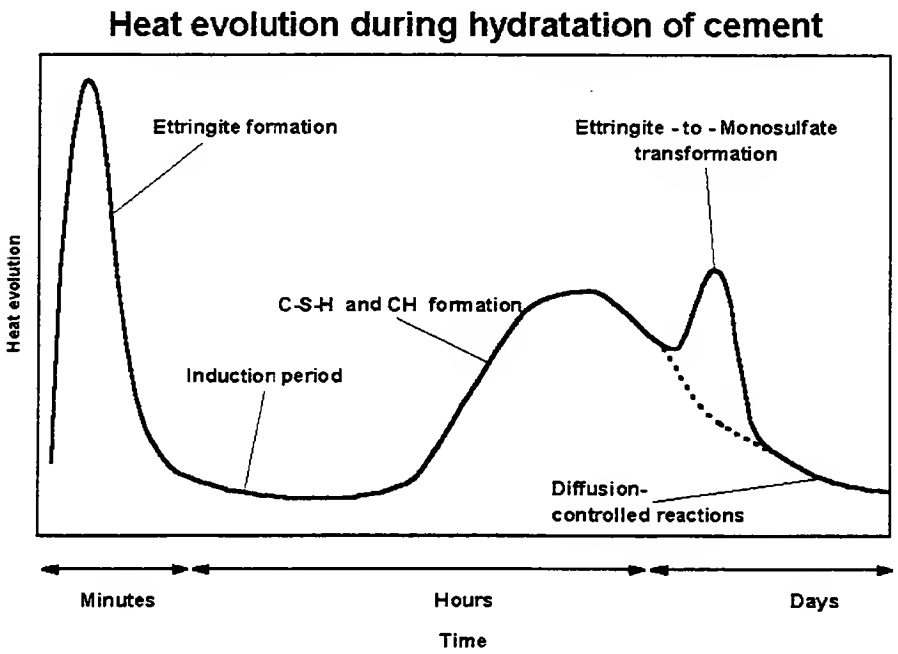
Third stage

In the third stage, the spaces between the solid particles are overbridged and filled with short fibres of the CSH phase. The initial matrix, formed by first hydrates of the aluminate, ferrite and silicates, is thus densified and the strength of the cement paste increases. The calcium hydroxide formed in large crystals is built into the CSH gel matrix. At this stage, ettringite is converted into monosulfate and corresponding hydration products are formed from the still not hydrated aluminate and ferrite phases.

As the hydration proceeds, the reactions get more and more diffusion controlled and the overall rate of reaction is decreasing. Hydration reactions, primarily those of C_2S, will continue as long as the reactants and space permit it; this can go on for years. In practice, there is usually never a complete hydration of the cement and unhydrated cement is nearly always remaining in the hardened cement paste.

A good overview on the hydration reactions in Portland cements can also be obtained by taking a look at the heat evolution curves. Figure 2 shows such a curve with the indications of the main reactions going on in the process of cement hydration.

Figure 2: Heat evolution curve of Portland cement



For blended cements, the reactions sequences get more complex. In case of the active mineral additives, the main differences to Portland cement hydration can be seen in more formation of CSH phase at later stage and a decrease in the amount of calcium hydroxide and the pore space. For the inert mineral additives, the effect will primarily be a „dilution" of the Portland cement matrix.

A general effect observed in blended cement is a certain delay of setting and hardening at early stages. Normally, the active mineral additives start to contribute to cement hydration only after more than 3 days, when suitable activation has taken place (blast furnace slag) or sufficient calcium hydroxide has been formed (pozzolans) for their reaction.

4.3 Acceleration or retardation of cement hydration

The process of hydration can be accelerated or retarded by different factors:

- Use of accelerators and retarders
  Until now, the most efficient accelerator is calcium chloride ($CaCl_2$), but the problem with this product is the corrosion of the steel reinforcement. The exact mechanism of acceleration by $CaCl_2$ is not known; it is believed that $CaCl_2$ is acting like a catalysator. Retarders for cement hydration are for instance phosphates, zincates and carbohydrates (including sugar). The retarding action is simply explained by the formation of a monomolecular protective film around the cement particles which slows down their reaction with water.
- Hydration temperature
  The rate of hydration is strongly influenced by the temperature. Rate of hydration at early age roughly doubles when the temperature increases by 10°C. The effect of temperature on cement hydration is of particular importance in extreme climates and for steam curing.
- Cement fineness
  The higher the fineness of the cement, the more extended is the zone of reaction. This leads of course to a higher rate of hydration.
- Cement composition
  The activity of the clinker used in the cement affects considerably the rate of hydration. A high rate of hydration can be expected of clinkers rich in C_3A and C_3S (see Figure 3) and with a high alkali content.
  Important for the rate of hydration is also the content of mineral additives in the cement. The rate of hydration generally is the lower, the higher the dosage of mineral additives in the cement.

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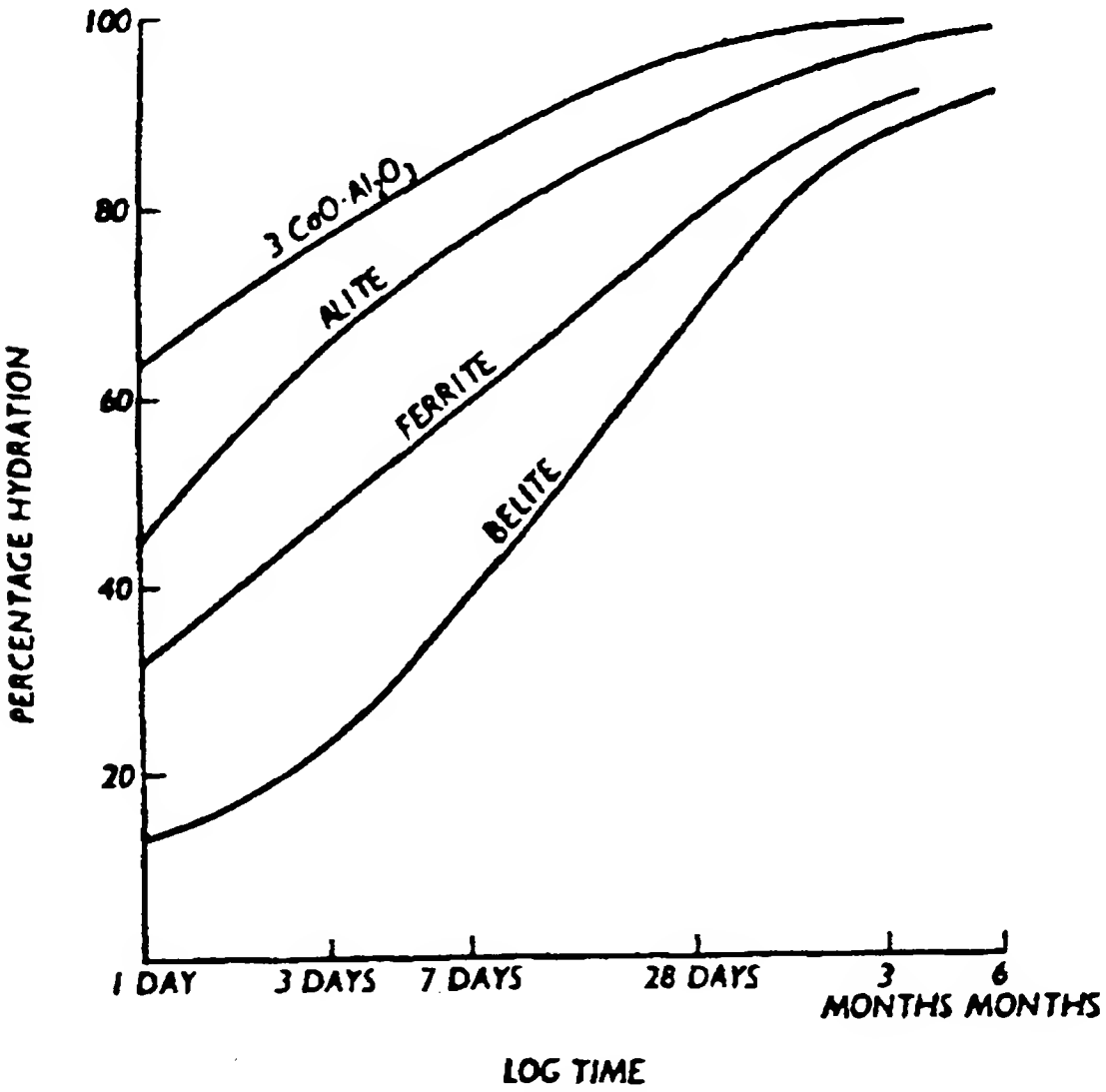
Figure 3: Rates of hydration of compounds in Portland cement



Increasing the rate of hydration, and hence the early strength development, is always done at the expense of the ultimate strength. An explanation is that the microstructure formed during the accelerated hydration is usually more coarse and less favourable for strength development. Contrary to this, retarded, slow hydration leads to the formation of a more refined microstructure which results in better final strength.

5. HEAT OF HYDRATION

The hydration reaction is an exothermic process, that means during the reaction of cement with water heat is liberated. The quantity of liberated heat is quite appreciable (typically 380 J/g for OPC at 28 days) and can lead to a significant temperature rise in mass concrete construction, where the heat is not allowed to escape (see Figure 4). During cooling of the hardened concrete, thermal gradients may develop and give problems with crack formation. That is the reason why special low heat cements have to be used for mass concrete applications.

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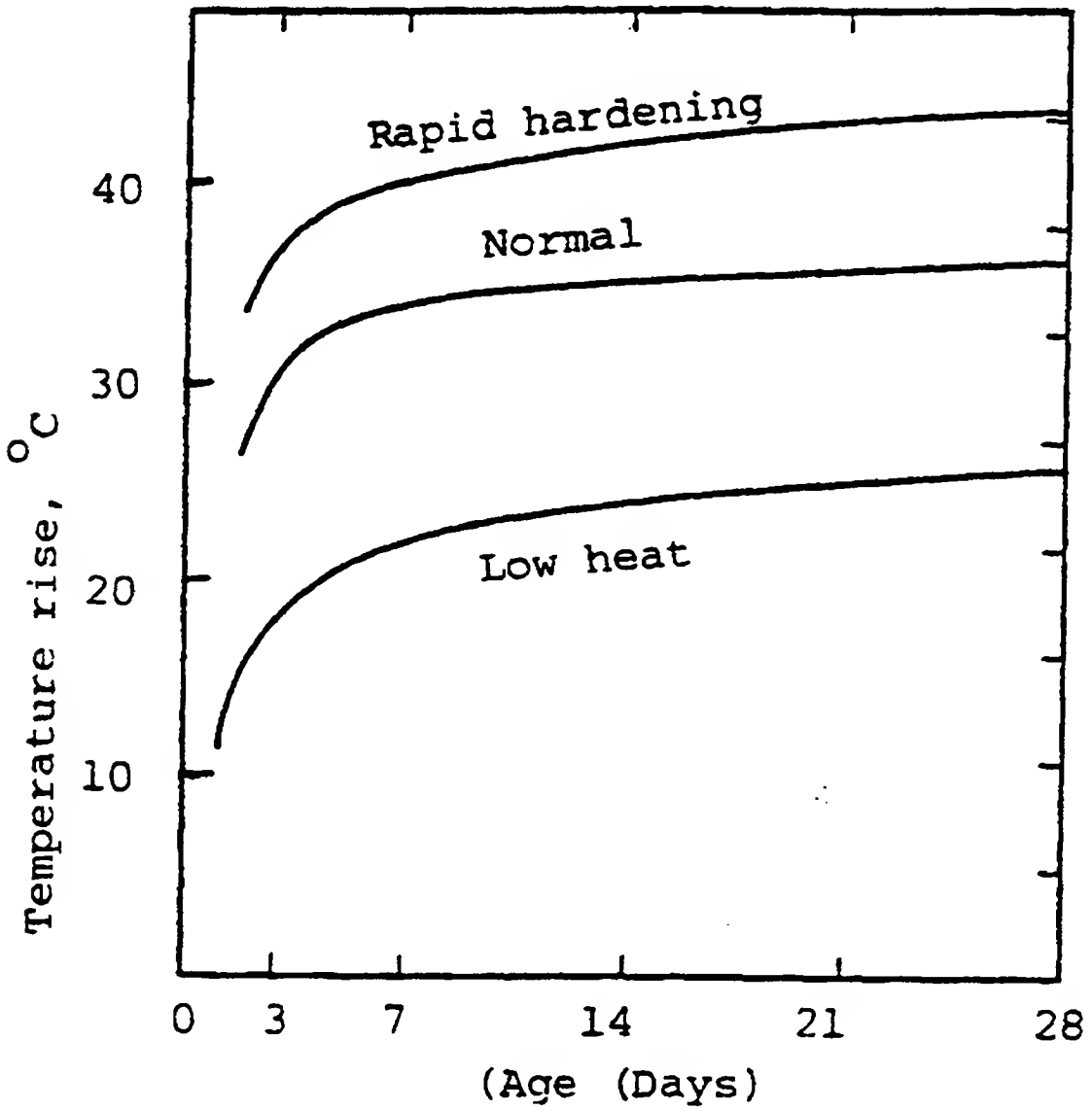
Figure 4: **Temperature rise in 1:9 (weight) concrete under adiabatic conditions for different Portland cement types**



The main measure to control the heat of hydration of Portland cement is the adjustment of clinker composition. The clinker minerals contributing most to the heat of hydration are C_3S and C_3A (see Table 2), so that it is necessary to limit these compounds to reduce heat development during hydration. On the other hand, it is also possible to keep the heat liberation low by not grinding the cement too fine.

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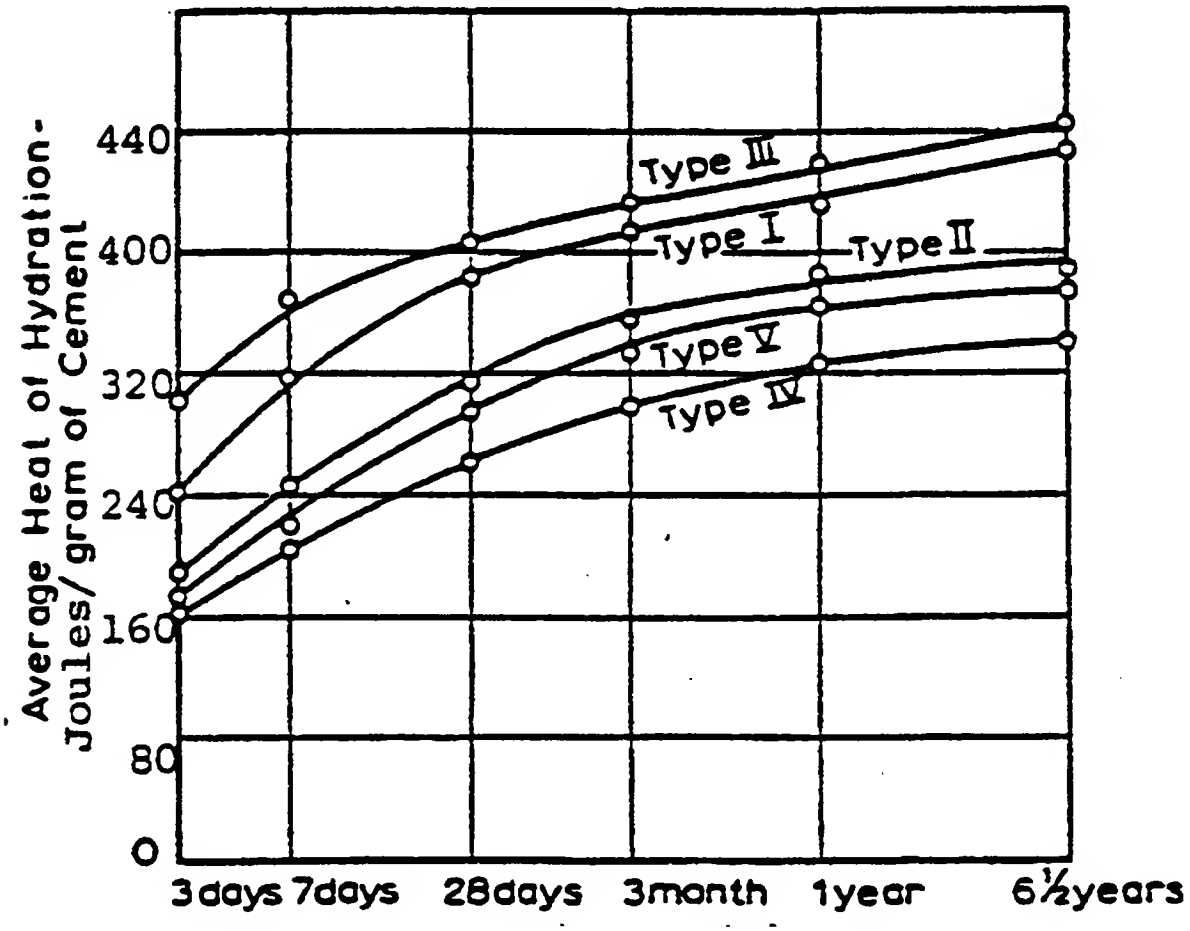
Table 2: Heat of hydration of the clinker components at 21°C (J/g)

Clinker component	3 days	28 days	6 ½ years
C_3S	245	380	490
C_2S	50	105	225
C_3A	890	1380	1380
C_4AF	290	495	495

The effect of clinker composition and cement fineness on the heat of hydration can be well seen in Figure 5 which shows the development of heat of hydration for the different Portland cement types. The low heat cements used for mass concrete - type II and IV ASTM - with low heat of hydration of about 250 to 300 J/g after 28 days contain less C_3S and C_3A and are fairly coarse. The rapid hardening cement - type III ASTM - with increased heat of hydration of about 420 J/g after 28 days is usually richer in C_3S and C_3A and of higher fineness, compared with the ordinary Portland cement - type I ASTM.

Figure 5: Development of heat of hydration for the different Portland cement types cured at 21°C (w/c-ratio of 0.40)



For practical purposes, it is not only the total heat of hydration that matters but also the rate of heat evolution. The same total heat produced over a long period of time can be dissipated to a greater degree, consequently producing a smaller rise in temperature in the concrete. In ordinary Portland cement about one half of the total heat is liberated between 1 and 3 days and about three quarters in 7 days. It is common to specify the heat of hydration after 7 and 28 days, giving a reasonable indication of both total heat of hydration and its rate of liberation.

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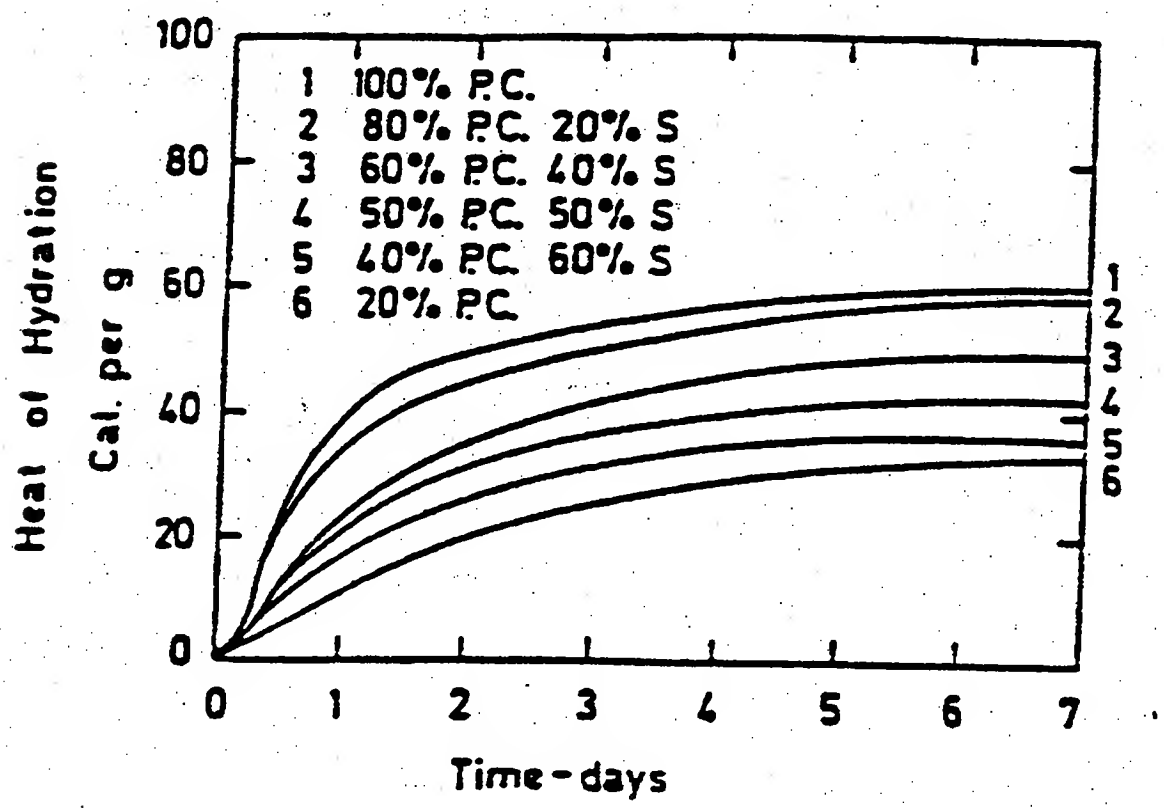
In addition to the composition and fineness of cement, the rate of heat evolution is also greatly influenced also by the temperature of hydration (see Table 3). The ambient conditions thus play also a decisive role with regard to the effects of heat of hydration on the concrete properties.

Table 3: Heat of hydration (J/g) developed after 3 days at different temperatures

Cement type	Temperature of cement hydration			
	4°C	24°C	32°C	41°C
I	154	285	309	335
III	221	348	357	390
IV	108	195	192	214

A very effective means to reduce the amount and rate of heat of hydration of Portland cement is the use of mineral additives in the cement (blended cements). By the replacement of the clinker by the less reactive or inert mineral additives, the heat development of the cement can be easily controlled. As an example, the heat development curves of slag blends with different Portland cement contents are presented in Figure 6. The advantage of the blended cement over the low heat Portland cements is that the same clinker as for the ordinary Portland cement can be used.

Figure 6: Heat of hydration of mixtures of OPC (P.C.) and ground blast furnace slag (S); isothermal method.



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6. ROLE OF GYPSUM IN CEMENT

6.1 General aspects

Ever since cement has been produced in greater quantities, gypsum ($CaSO_4 \cdot 2H_2O$) has been ground into the cement. Gypsum is added to the cement mainly for the purpose of regulating its setting time. It prevents flash setting and makes the concrete workable for hours.

The gypsum influences not only the setting but also other cement properties such as grindability, sensitivity to storage, volume stability and strength. Gypsum is an extremely important part of cement. It is, however, often neglected to pay proper attention to it in the production of cement.

Here, only the effect of gypsum on setting, volume stability and strength shall be elucidated more in detail. For the better understanding of the influence of gypsum on cement, it is first necessary to take a look at the different modifications in which calcium sulphate can be present. In a final part of this chapter, the possible gypsum substitutes shall be discussed.

6.2 Calcium sulphate modifications

In cement, it is possible to find at least five basic modifications of calcium sulphate (see Table 4). The stable modifications, dihydrate (gypsum) and anhydrite II (natural anhydrite), can be found in nature and both are used as additives to clinker in the grinding process. The metastable modifications, hemihydrate and anhydrite III, are not available in nature, but can easily form during grinding and storage of cement at elevated temperatures by dehydration of gypsum. The high temperature modification anhydrite I can form during burning of the clinker in the cement kiln.

Table 4: Modifications of calcium sulphate in cement

Designation	Formula	Crystal water	Density	Range of stability	Solubility at 20°C	Occurrence	
		%	g/cm³	°C	%	nature	cement
Dihydrate	$CaSO_4 \cdot 2H_2O$	20.92	2.32	< 40	0.20	yes	yes
Hemihydrate (α,β)	$CaSO_4 \cdot 1/2H_2O$	6.21	2.70	metast.	0.95	no	yes
Anhydrite III (α,β)	$CaSO_4$	0	2.50	metast.	0.95	no	yes
Anhydrite II	$CaSO_4$	0	2.98	40-1180	0.20	yes	yes
Anhydrite I	$CaSO_4$	0	--	> 1180	--	no	(~clinker)

The different mode of action of the individual modifications on the properties of cement can be mainly attributed to their different solubilities in water. In Figure 7, it can be seen that the solubility of the hemihydrate and anhydrite III is appreciably higher than that of dihydrate or anhydrite II. The high solubility of hemihydrate and anhydrite III can lead to anomalous setting, as discussed later. The solubility of dihydrate and anhydrite II in the ambient temperature range is approximately the same, but their effect on properties of cement can be very different. The reason for this is the different rate of solubility which is greater for the dihydrate than for anhydrite II.

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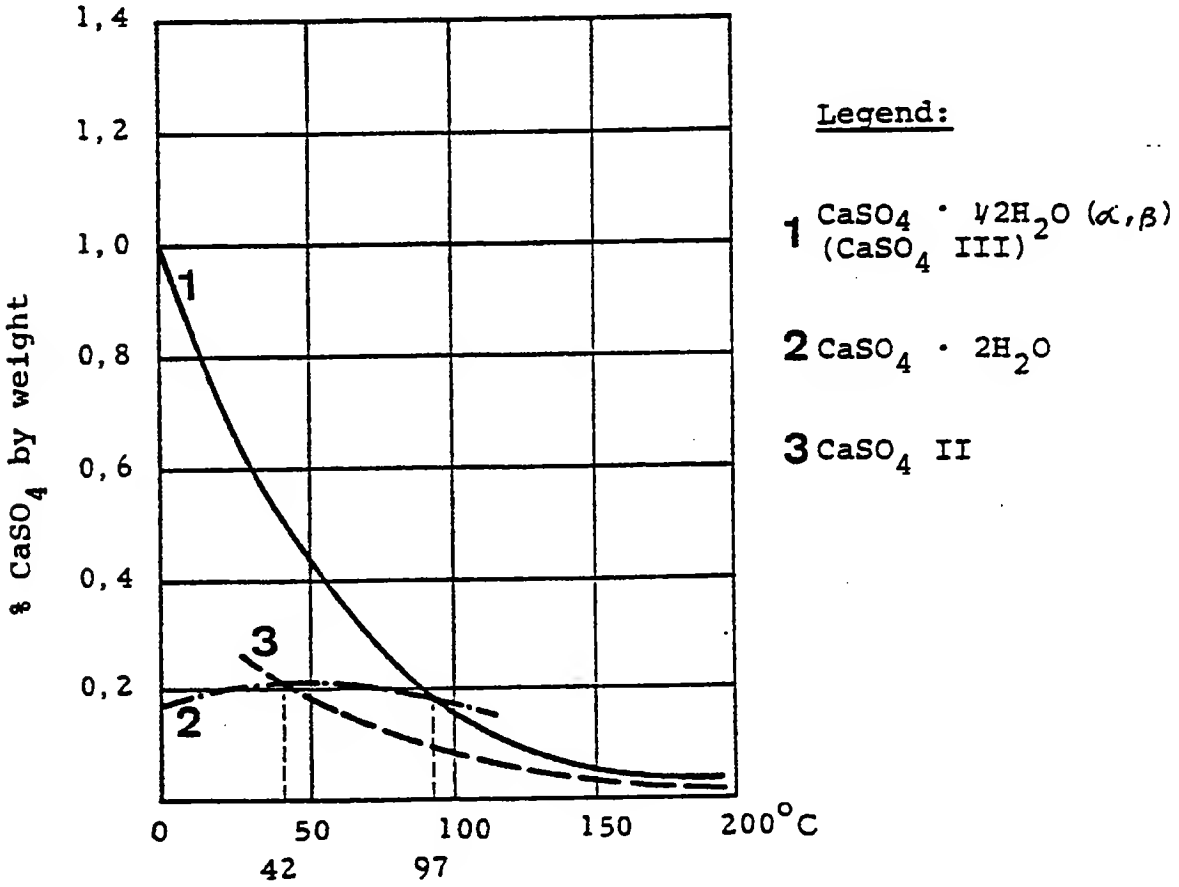
Figure 7: The Solubility of gypsum, hemihydrate and anhydrite



6.3 Gypsum and setting of cement

The reactions which take place immediately after the addition of water to cement are of decisive importance for the setting of cement. Directly after the mixing of cement with water, sulphate dissolves and reacts with aluminate to form ettringite. This vigorous initial reaction ceases after a few minutes and then the so-called dormant period starts.

According to the classical theory, the set retardation is due to the fact that the ettringite forms a cohesive cover around the aluminate particles and in this way inhibits their further reaction (see Figure 8). The reaction inhibiting effect of the ettringite cover is ended only when this cover is broken open by the pressure of crystallisation and not sufficient sulphate is left to close the burst section. Thereafter, the reaction of C_3A can continue unhindered until complete hydration.

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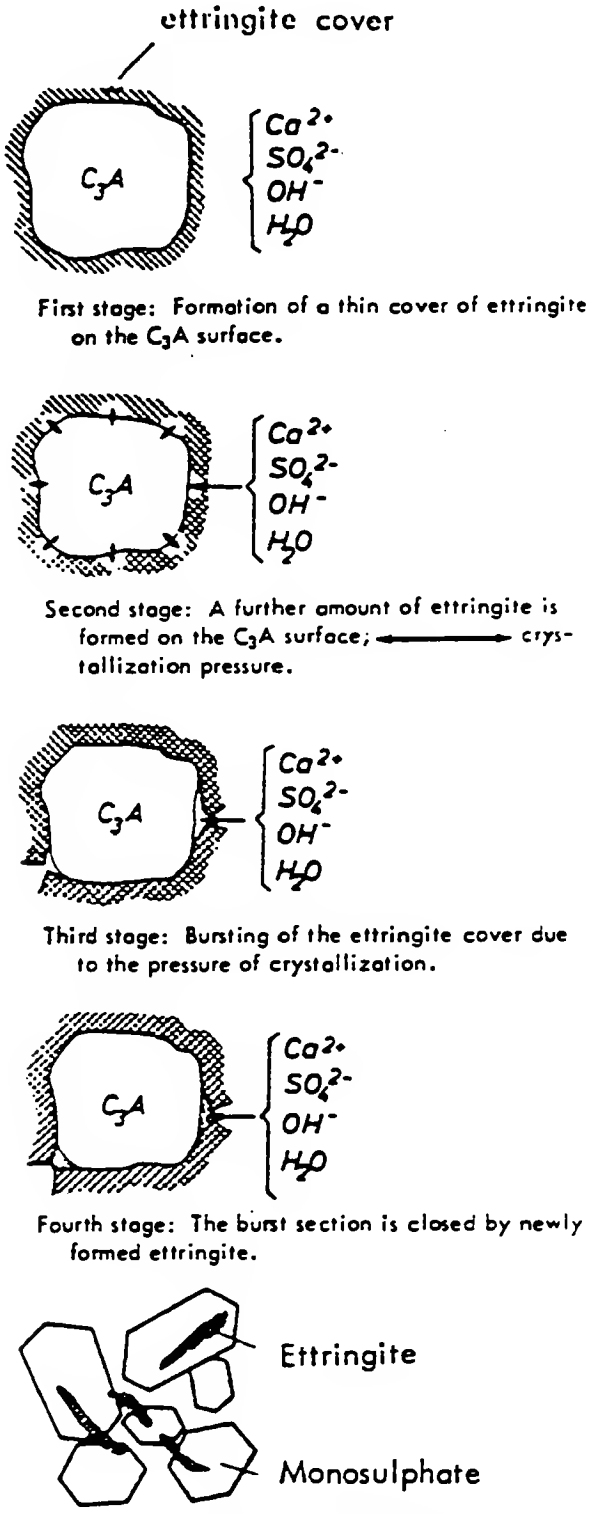
Figure 8: Schematic description of set retardation due to C_3A sulfate interaction



First stage: Formation of a thin cover of ettringite on the C_3A surface.

Second stage: A further amount of ettringite is formed on the C_3A surface; ⟷ crystallization pressure.

Third stage: Bursting of the ettringite cover due to the pressure of crystallization.

Fourth stage: The burst section is closed by newly formed ettringite.

Fifth stage: The SO^{2-} is not sufficient for the formation of ettringite; complete hydration of C_3A with consumption of ettringite to form $C_3AC_3H_{12}$, $C_3A(C_3,CH)H_{12}$.

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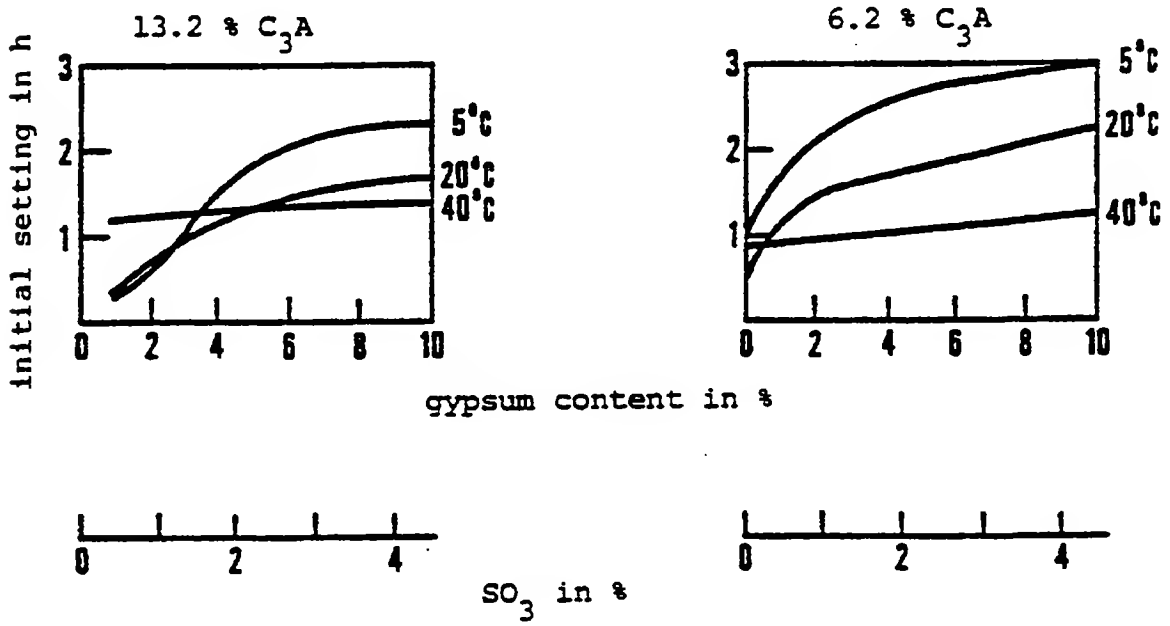
The reasons for the setting of cement are still not exactly known. Some researchers explain the setting of the cement by the recrystallisation of the initially finely divided ettringite into bigger crystals which are building bridges between the cement particles. Other researchers attribute the initial set of cement as corresponding to that stage in the hydration sequence when a sufficient amount of tricalcium silicate has been converted to calcium silicate hydrate. This initial removement of water causes a partial loss of the plasticity of the cement paste.

The setting of cement is mainly influenced by the reactivity of the clinker (i.e. C_3A and alkali content) and by the type and amount of the calcium sulphate added. The cement fineness, the dosage of mineral additives, use of chemical admixutres and the hydration temperature play also an important role. The effect of C_3A content, gypsum dosage and hydration temperature on the initial setting of Portland cement is illustrated in Figure 9.

Figure. 9: Influence of gypsum dosage, C_3A content and hydration temperature on initial setting of Portland cement



In order to obtain normal setting, it is essential to adjust the supply of soluble sulphate to the reactivity of the clinker during the first minutes and hours of hydration (see Figure 10). If there is no proper balance between sulphates and clinker reactivity, abnormal setting can occur:

Flash set is caused by the formation of aluminate hydrate, due to a high content of reactive C_3A and/or a too small amount of easily soluble calcium sulphate.

False set can be traced back to high temperature in the mill, causing a dehydration of the gypsum to an easily soluble hemihydrate and anhydrite III. Hemihydrate and anhydrite III recrystallize to gypsum whose crystals grow into each other and form a solid framework which affects the stiffening of the cement paste. This framework of gypsum crystals is broken down again through reaction with C_3A. For this reason the cement paste can regain its previous plasticity by remixing.

Quick set: The high content of easily soluble sulphate and reactive clinker minerals can also have an accelerating effect on the formation of sulfo-aluminate hydrate and calcium silicate hydrate which can likewise lead to early stiffening of the cement (quick set).

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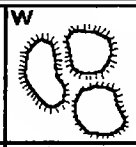
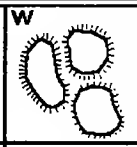
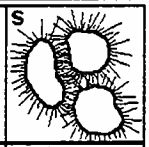
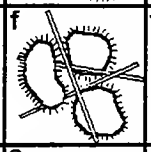
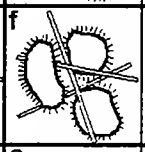
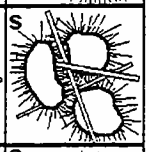
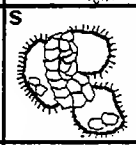
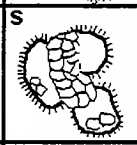
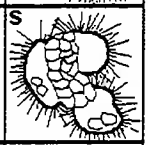
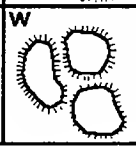
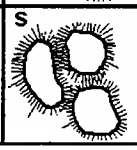
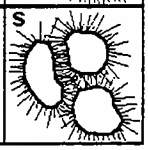
Figure 10: Formation of rigid structure during setting of Portland cement

Clinker reactivity	Sulfate availbility in solution	Type of set	Time of hydration		
			10 min.	1 hour	3 hours
low medium	low medium	regular			
low	high	false			
high	low	flash			
high	high	quick			

w = workable s = set f = false set

Effect of gypsum on strength and volume stability

The addition of gypsum - in addition to its principle purpose of regulating setting - has a significant effect on the strengths and volume stability. Up to a certain limit, which depends on the clinker composition, the gypsum addition increases the strengths and prevents shrinkage. If the gypsum addition surpasses this limit, it causes considerable swelling which can lead to expansion. This is why an upper limit for the SO_3 content is given in the cement standards (see Fig. 11).

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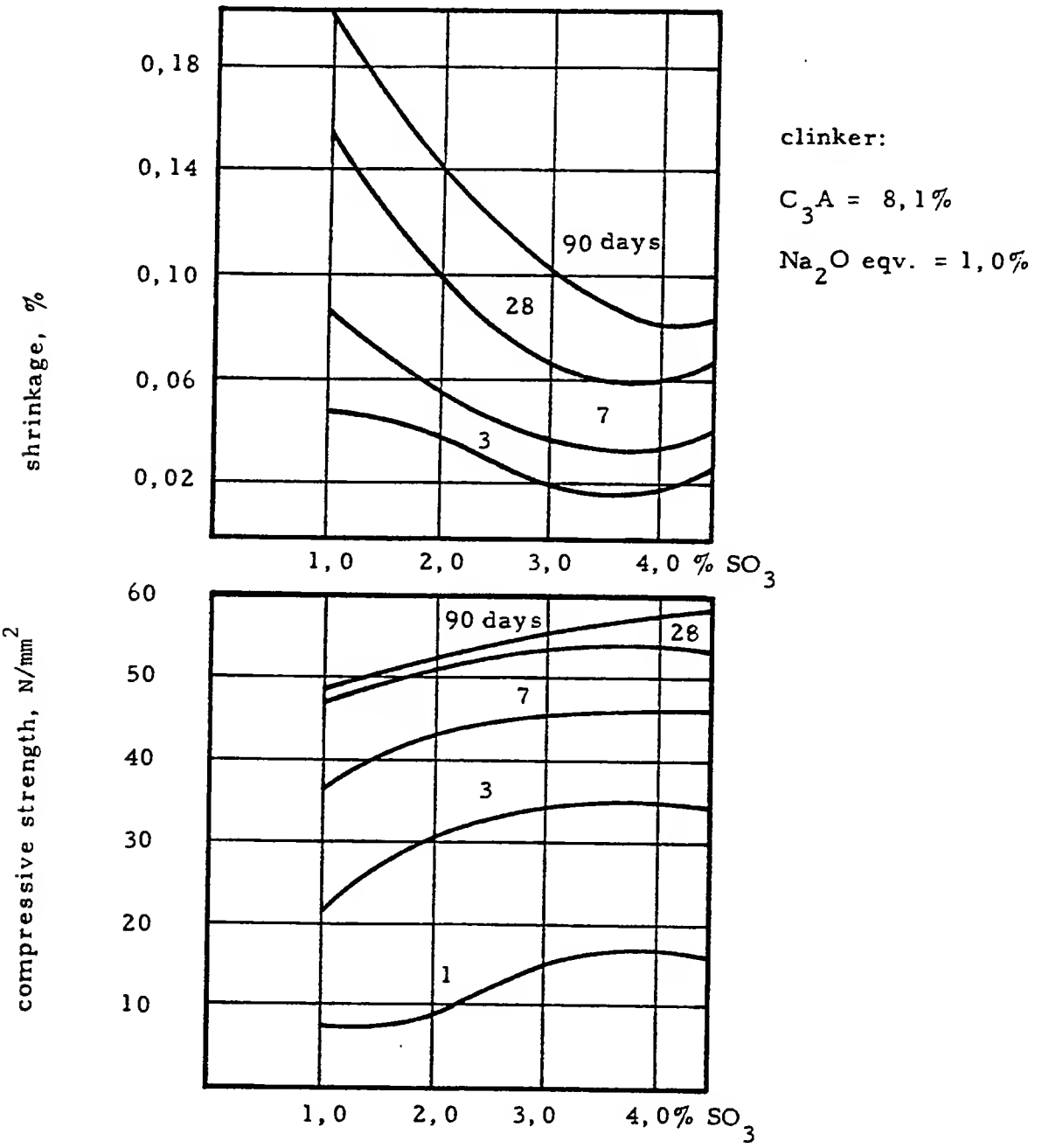
Figure 11: Shrinkage and compressive strength of cements as a function of the SO_3 Content



The gypsum required to obtain optimum strength and volume stability grows with the C_3A content of the clinker, cement fineness, alkali content and application temperature. Other variables influencing the optimum SO_3 content are the form (and solubility) of the calcium sulphate in the cement, the nature and amount of mineral additives and also the presence of chemical admixtures. Often the optimum SO_3 content would be even above the maximum value given by the standards.

6.4 Gypsum substitutes

There has never been lack of effort to substitute gypsum by other set regulators. In particular in countries with few natural gypsum deposits, the cement industry is searching

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intensively for alternative materials. The main alternatives up to date are the use of natural anhydrite and limestone and the application of by-product gypsum coming form other industries.

The use of natural anhydrite as partial substitute for gypsum is already well established in practice. In general, it appears that blends of 40 to 60 percent gypsum and 60 to 40 percent natural anhydrite can be safely used with all clinkers. The advantage of such blends over pure gypsum is the reduction of the risk of false setting and the improved storage stability and flowability of the cement.

In some places, limestone is used as replacement for gypsum. For very reactive clinkers, the replacement is typically limited to 25%. In cements with less reactive clinkers, replacement levels of up 50% may be possible without negative effects on the setting of cement.

With respect to by-product gypsum, there are a number of commercial processes which produce such materials. An overview on the main sources of by-product gypsum is given in Table 5; the most important sources regarding quantities are phosphogypsum and desulphogypsum. Another type of by-product not mentioned in the table are used plaster moulds, which are for instance used in casting certain clay products.

Table 5: Main sources of by-product gypsum

Product/process	By-product	% sulphate*	% impurities
orthophosphoric acid	phosphogypsum	95-98 (G)	0.2-1.5 P_2O_5, up to 1.5 fluoride
hydrofluoric acid	fluorogypsum	80-95 (HH or AH)	2-3 CaF_2, unreacted silicofluorides
citric, formic and tartaric acid	organogypsum	---	---
boric acid	borogypsum	40-50 (G), 20-30 (AN)	7-10 H_3BO_3
flue gas SO_2 removal	desulfogypsum	90-95 (G), sulfite for mod. process	---
titanium white	titanogypsum	90-95 (G)	0.4 TiO_2

* G = $CaSO_4 \cdot 2H_20$, HH = $CaSO_4 \cdot \frac{1}{2}H_2O$, AN = $CaSO_4$

The main problem for the use of such by-products as subsitute for natural gypsum is the presence of impurities, which may have some harmful effect on setting, strength develpment or some other property of the cements. Moreover, the amount of impurities can fluctuate quite a lot so that the variability in cement properties may also be higher.

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7. MICROSTRUCTURE AND PROPERTIES OF HARDENED CEMENT PASTE

7.1 Basic physical features

In this section we deal with the physical structure of hydrated cement paste (HCP), as well as the effect of its morphology on the important properties of hardened concrete.

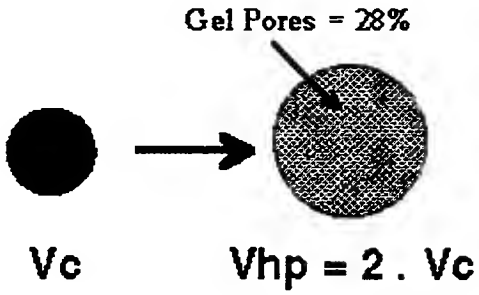
We will be concerned with the main chemical reaction taking place during hydration:

Calcium Silicates + H_2O ---> C-S-H + $Ca(OH)_2$

where C-S-H are the hydrates of the Calcium Silicates (C_3S and C_2S), also called "C-S-H gel".

The most important physical aspect of that reaction is the fact that the total volume occupied by the hydration products is, roughly, twice the volume occupied originally by the unhydrated cement (see Figure 12):

Figure 12: Basic physical feature of cement hydration



It must be mentioned that, out of that total volume, 28 % are very small pores (gel pores) and the rest (72 %) is solid phase:

Gel pores = 0.28 Vhp

Figure 13 shows schematically the growth of crystals during hydration of two pastes, one of low and the other of high water/cement ratio (w/c). The w/c ratio is a measure of the degree of dispersion of the cement grains in water.

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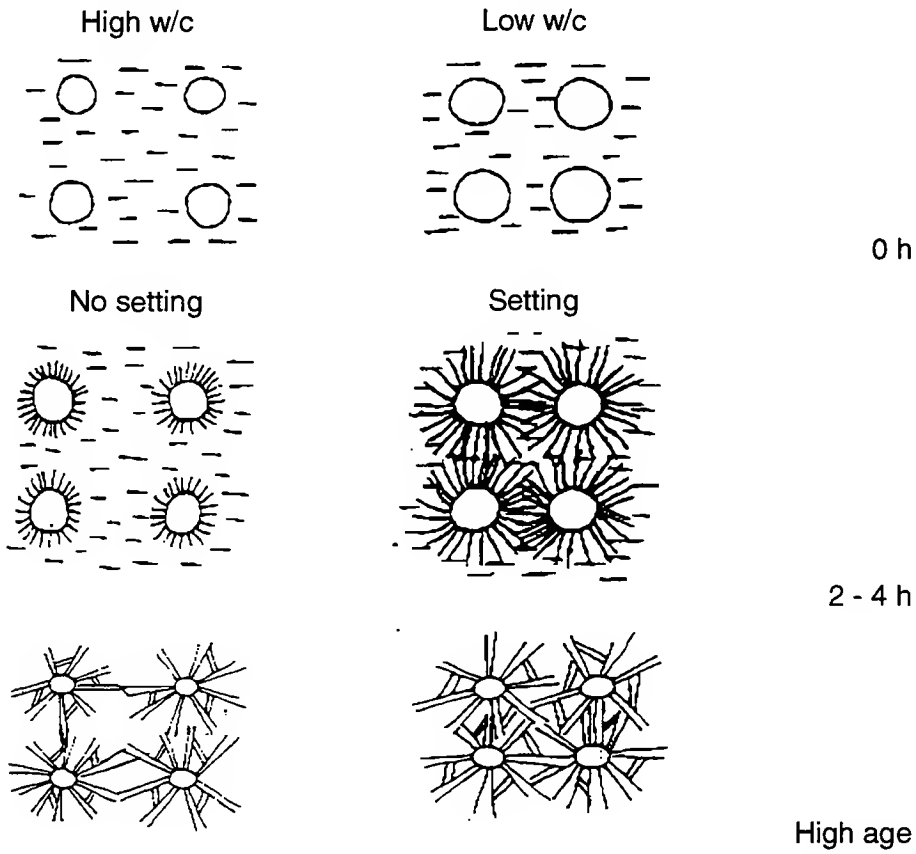
Figure 13: Structure formation in cement paste during hydration



Cement hydration can be seen as a continuous process by which the capillary pores (space originally occupied by water) are being gradually filled with hydration products.

7.2 Morphological features

Figure 14 presents a model (Feldman and Sereda) of the general structure of HCP, showing its main features.

In the hydrated cement paste, there are two distinctive phases:

- Solid Phases
- Pores (at least partially filled with water)

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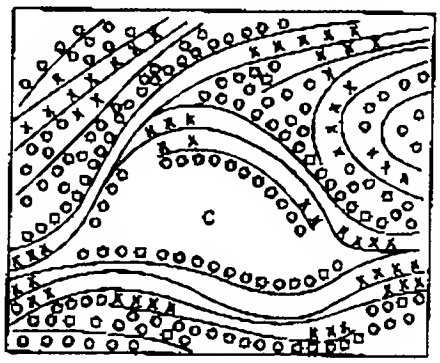
Figure 14: Model of Feldmann and Sereda for the general structure of hydrated cement paste



Legend

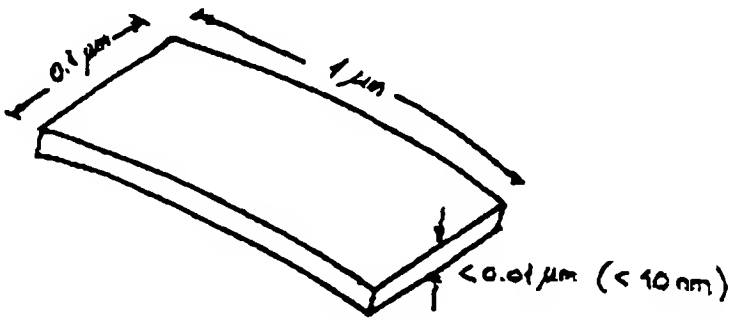
x	Water in interlayer regions
o	Water absorbed on surfaces
C	Capillary pore
—	C-S-H sheets

7.2.1 Solid phases

7.2.1.1 Calcium silicate hydrates-(C-S-H)

The C-S-H particles occupy 50 - 60 % of the total solid volume in hydrated paste; it is also called C-S-H gel. The particles are very small (colloidal dimensions) and sheet-like shaped (see Figure 15)

Figure 15: Basic feature of C-S-H particle



The C-S-H particles are arranged as a highly disordered layered structure (see Figure 14) with very high specific surface ≈ 3.000.000 cm^2/g.

The strength of the HCP is attributed to the attraction forces between the C-S-H crystals over their enormous surface.

The space left between the particles is called gel pores; these pores are extremely small (about 1.5 nm), of the order or magnitude of a water molecule (0.25 nm).

7.2.1.2 Calcium Hydroxide (CH)

Compound with a definite stoichiometry: $Ca(OH)_2$, forming large crystals (0.01 - 0.1 mm) of hexagonal shape (Portlandite).

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These crystals constitute 20 - 25 % of the solid volume in the hydrated paste.

Their contribution to strength is low (low surface area); however, their presence is very important to passivate the embedded steel rebars.

7.2.1.3 Calcium Sulfoaluminates

Occupy 15 - 20 % of solid volume in hydrated paste; they only play a minor role in strength.

There exist two forms:

Ettringite: Elongated needle shaped crystals

Monosulfate: Hexagonal plates or "rosettes"

Iron oxide can replace aluminium oxide in the Crystal structure.

7.2.1.4 Unhydrated Clinker Grains

Smaller grains react faster and coarser grains (> 30 µm) tend to remain partially unhydrated.

7.2.2 Pores

7.2.2.1 Interlayer space in C-S-H (gel pores)

Occupy 28 % of bulk volume of C-S-H gel

Size: 0.5 nm - 2.5 nm (H_2O molecule = 0.25 nm)

They are always present in HCP but, due to their small size do not adversely affect strength and permeability. However, the movement of water - that is firmly held within the gel pores - is the main reason for drying shrinkage and creep.

7.2.2.2 Capillary pores

Represent the space originally occupied by water, not filled with hydration products. Their size and volume depend on the w/c ratio and the degree of hydration:

- well hydrated paste, low w/c: 10 - 50 nm
- at early ages, high w/c: up to 10 µm

7.2.2.3 Macropores

Air bubbles naturally entrapped or intentionally entrained during mixing (10 - 100 µm).

"Compaction" voids: mm or cm

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7.3 Evolution of pastes during hydration

Figures 16, 17 and 18 show the evolution of the structure of cement pastes, made with the same amount of cement (1 kg) but with different w/c ratios, during hydration (under water).

Figure 16: Development of Hydration of Cement Paste made with 1 kg of cement; w/c = 0.70

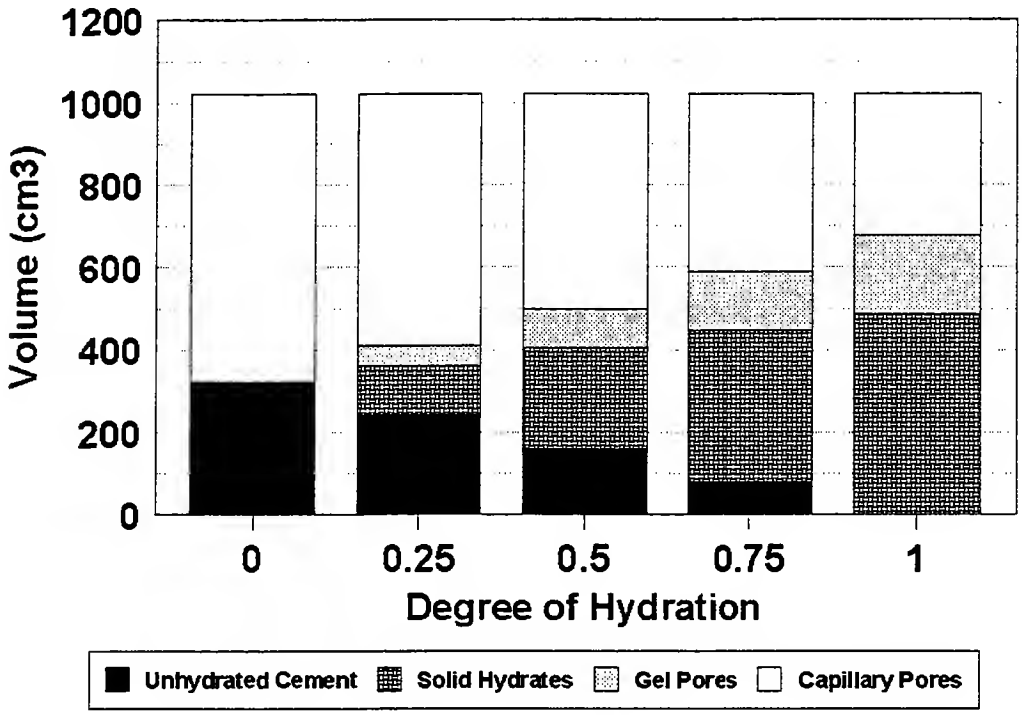

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Figure 17: Development of Hydration of Cement Paste made with 1 kg of cement; w/c = 0.50

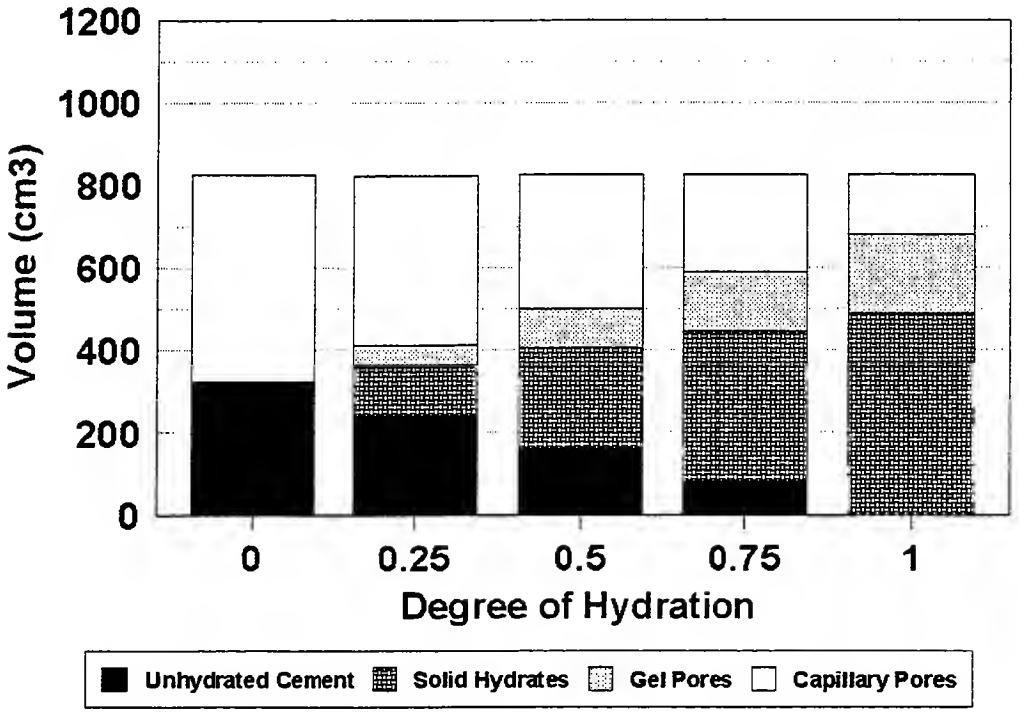
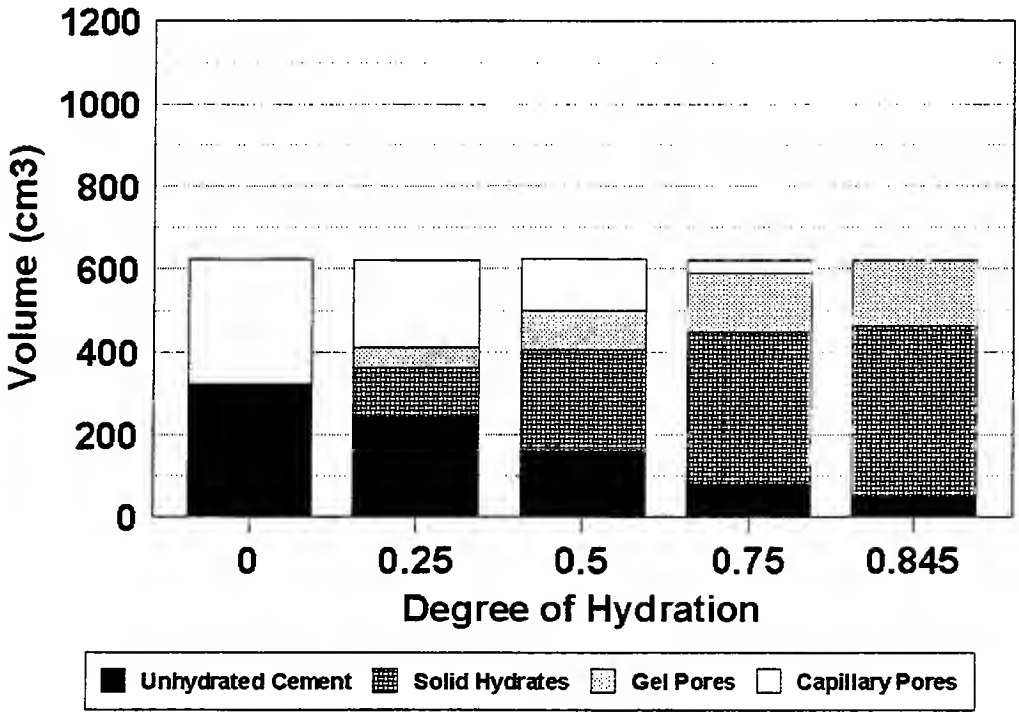


Figure 18: Development of Hydration of Cement Paste made with 1 kg of cement; w/c = 0.30



Initially, all pastes contain about 320 cm^3 of unhydrated cement (1 kg), but different volumes of water, 700, 500 and 300 cm^3 for w/c ratio 0.7, 0.5 and 0.3, respectively. Consequently, pastes with higher w/c ratio will have larger total volumes. The volume of water corresponds to the initial volume of capillary pores.

As hydration proceeds, the total volume remains basically constant, but the amount of unhydrated cement is gradually reduced. At degree of hydration 0.5, half of the original cement has been consumed and replaced by hydration products. The latter occupy a volume about double that of the original cement that hydrated, thus reducing the volume of capillary pores. From the total volume of hydrates, 72% are solid phase and 28% are gel pores.

If we compare the pastes with w/c 0.50 and 0.70 we see that, as they contained initially the same volume of cement, at any stage both pastes contain the same amount of hydration products. However, the paste with higher w/c ratio will contain more pores because the cement grains were more dispersed.

The paste with w/c ratio 0.30 can never reach full hydration because there is no room left to accomodate new hydration products. High-strength concretes, with very low w/c ratios always contain unhydrated cement for that reason.

Clearly, pastes and concretes with lower w/c ratios will have less and smaller capillary pores, thus explaining why they are stronger and less permeable (more durable), see Figures 19 and 20.

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Figure 19: Strength vs. w/c-ratio

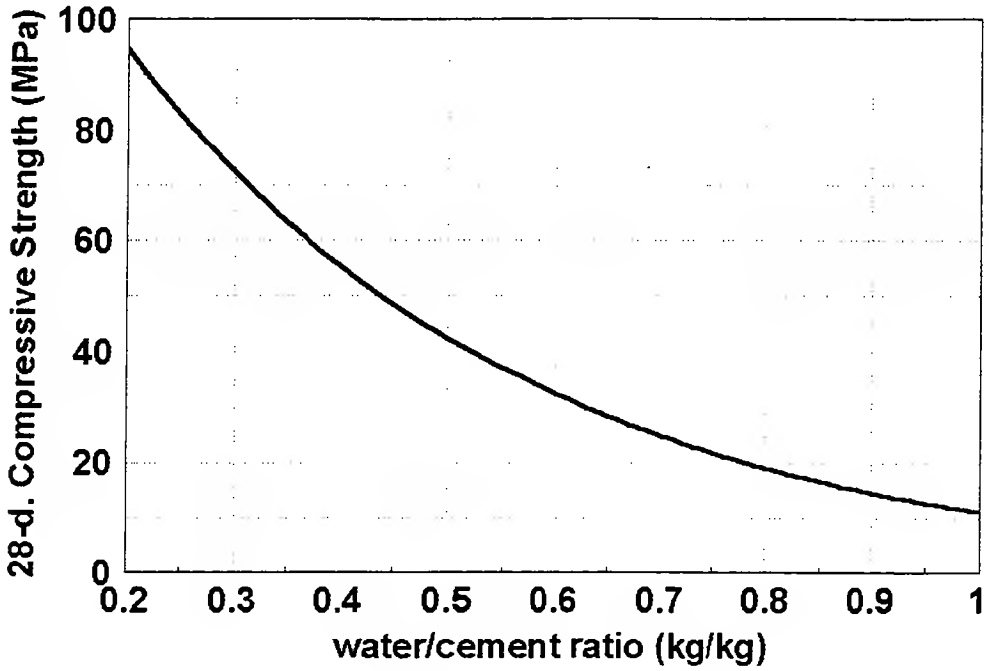

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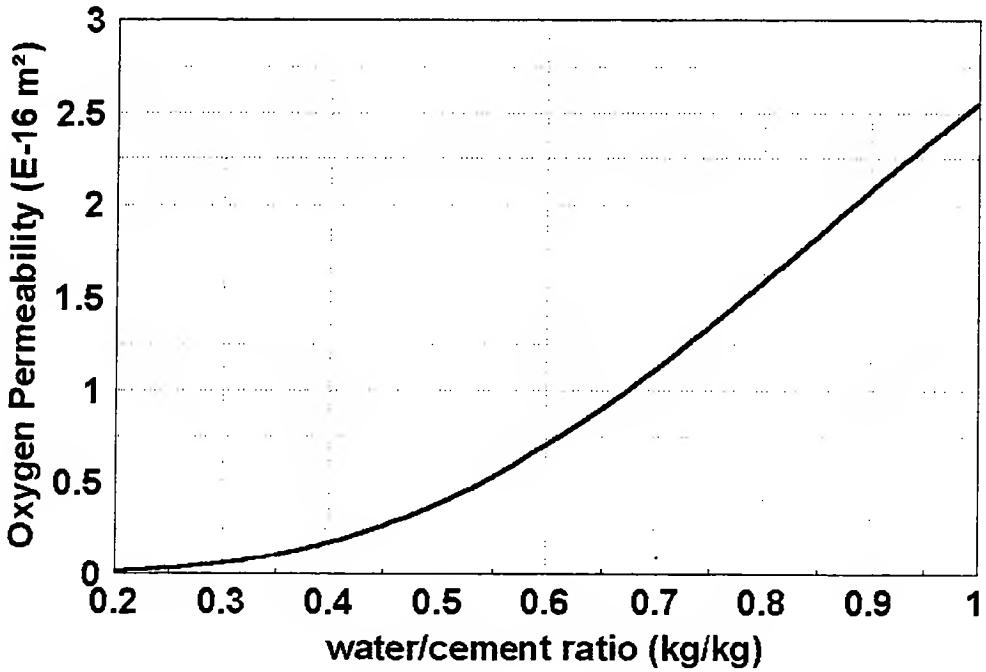
Figure 20: Permeability vs. w/c-ratio



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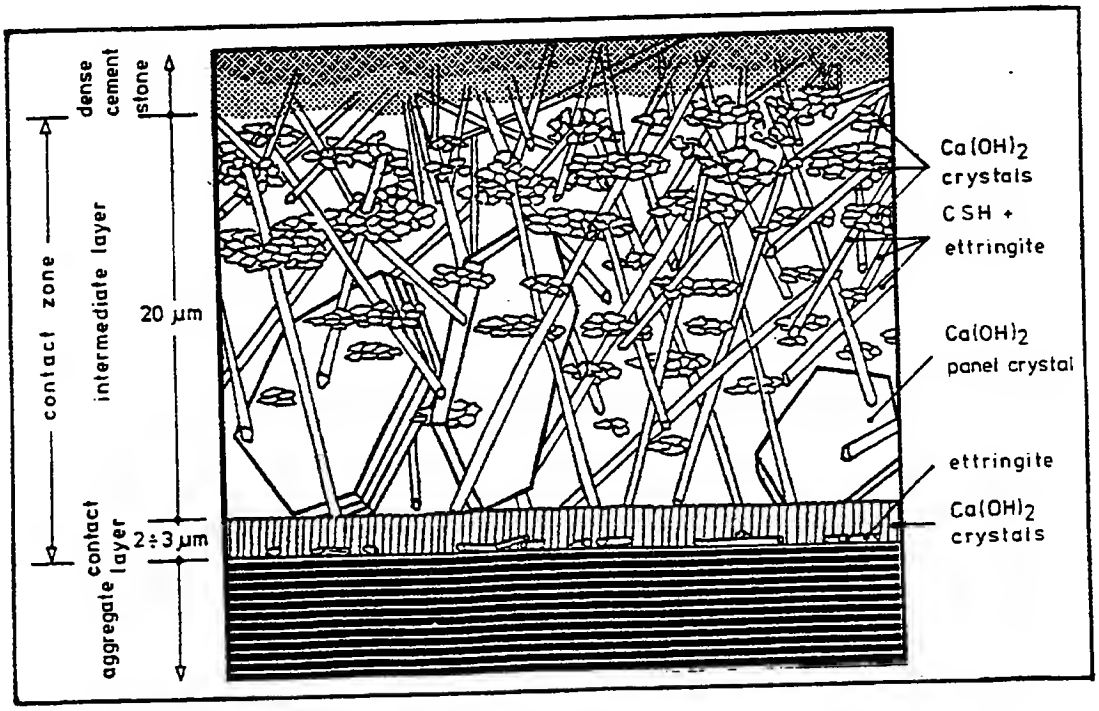
7.4 Influence of the aggregates on the structure

Figure 21 shows that the presence of the aggregates distorts the structure of the HCP in their vicinity. In this contact or transition zone (about 20-25 µm thick), there exists a higher capillary porosity and a higher proportion of $Ca(OH)_2$ than normal, making it weaker than the rest of the paste. This explains why, in the case of rounded and smooth aggregates the failure of concrete tends to happen at the interface between the coarse aggregate and the paste.

Figure 21: Model of the contact zone between cement stone and aggregate



To achieve high-strength concretes it is necessary to overcome this weakness of the contact zone. The increased porosity has to be compensated by using high dosages of a superplastizer and the preferential alignment and presence of $Ca(OH)_2$ crystals can be solved by adding microsilica or fly-ash.

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Chapter 2

Special Cements

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SPECIAL CEMENTS

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1. DEFINITION

Special cements are cements with special properties that meet particular requirements which are not fulfilled by ordinary cements.

These properties refer either to the performance of cement in fresh and hardened concrete as well as in other cementitious blends, or to a special field of application. They can be produced with appropriate selection of clinker raw materials and/or cement constituents, adoption of special measures in manufacturing, tailored cement compositions.

As a consequence of these actions, special cements can still comply with existing standards on common cements, but very often they are better described in appropriate specifications, or otherwise they are produced on the basis of specific agreements between producer and user.

In spite of the several types listed in the following, their application is still limited in quantity and the total amount of marketed special cements can be estimated in the order of 10 to 15% of production. Nevertheless the additional possibilities given by the recently issued new cement standards and the increased severity of the environments where concrete is put into service are going to make this quantity increase in the near future.

2. TYPES OF SPECIAL CEMENTS

Special cements are usually developed and produced to meet performance and durability requirements, in particular

- improved strength development
- increased resistance to chemical attack
- improved compatibility with reactive aggregates
- suitability for use at elevated temperatures and pressures
- suitability for use in special applications
- applicability in architectural purposes

As already discussed in the chapter on special clinkers, special cements may belong to three main categories

- Portland cement or blended Portland cement
- Modified Portland cement
- non-Portland cement.

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2.1 Special Portland Cements for Durability

Their main hydraulic constituent is generally Portland cement clinker, very often tailored to obtain special characteristics that yield the desired properties to the cement. For this reason, particular measures need to be taken in the production of clinker, expecially from the point of view of the selected raw materials to employ. In some cases this could not be enough or the involved costs are much higher than the obtained benefit.

So, the same or even better final properties can be achieved by blending clinker with appropriate mineral components and/or additions, as described in chapter , dealing with design and properties of blended cements.

Main durability characteristics for Portland type special cements are

- resistance to sulfate attack
- low heat of hydration
- resistance to pure water attack
- low alkali content or low reactivity with amorphous silica
- resistance to freeze-thaw cycles.

Before dealing with the different durability aspects that can require use of special cements, a general remark must be done. The main factor influencing durability is the proper design, production, compaction and curing of concrete. Any special cement, designed for the enhancement of durability will fail when used in the production of a poor concrete. A well compacted, high strength, low porosity concrete will be by far less prone to be attacked by external agents, even when produced with ordinary cements.

2.1.1 Sulfate resisting cements

Sulfates can be found in natural and industrial waters, as well as in soils. Soluble sulfates can react with lime and aluminates present in the hardened concrete and form respectively gypsum and ettringite. Both reactions entail expansion and consequent concrete deterioration. Sulfate resisting cements are characterized by a low C_3A content, to minimize the risk of ettringite formation. This type of binders set and harden normally and in fully compacted concrete are not attacked by sulfates in a wide range of concentration. At the same time they possess low-heat properties.

2.1.2 Low heat cements

Hydration reactions of cement develop heat. When cement is used in the production of mass concrete, temperature gradients generate between core and surface of the conglomerate cause strains and may eventually lead to cracking. Low heat cements have a reduced heat of hydration, obtained by altering the chemical composition (low C_3S and C_3A contents). The use of low-heat cement is recommended for mass concrete production or for large structural sections. They usually set and harden at a lower rate than for ordinary cements, especially in cold weather, but ultimate strengths may be higher. They possess also good sulfate resisting properties.

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2.1.3 Leaching resistant cements

Waters with low salinity or a high content of carbon dioxide (CO_2) are capable to dissolve hydration lime present in hardened concrete structures and may subsequently also subtract lime from silicate hydrates, thus causing damage to the hardened cement paste. Cements resisting to leaching have a low development of calcium hydroxide (lime) after hydration of clinker silicates, as a consequence of their low C_3S content. Their use is suggested in hydraulic works like basins, river barriers and sides. At the same time they possess rather low heat evolution characteristics.

2.1.4 Low alkali cements

Some aggregates may contain forms of reactive amorphous silica. In the presence of water this can react with the soluble alkali of cement and form locally an expansive gel that can deteriorate concrete. In such cases, *when the use of reactive aggregates is absolutely unavoidable,* the use of a cement proven to counteract alkali-silica reaction (ASR) is suggested. Such cements can either be low-alkali or blended cements, since the pozzolanic or slag material can immediately react with alkali and prevent subsequent reaction with aggregates in the hardened paste.

2.1.5 Sea-water resisting cements

Deterioration of concrete which is in contact to sea water takes place as a consequence of some of the already described phenomena. Concurring factors are chemical attack by $MgSO_4$, mechanical stress caused by tydal waves, crystallisation pressure due to deposition of salts in the wind and water line. Cold climates add freeze-thaw effects, while in warm areas some reactions are accelerated. Sea-water resisting cements are standardised in some countries, being moderate C_3A content or blended cements.

2.1.6 Freeze-thaw resisting cements

In harsh climates with high temperature differences and repeated freeze-thaw cycles, freezing of water contained in gel and aggregate pores will cause a volume increase of ab. 10% and the consequent development of internal pressure; repeated actions of this type deteriorate the concrete. Entrainment of air in form of microbubbles will produce a closed artificial porosity acting as expansion chambers for the ice generated. Some standards such as ASTM provide for cement types with air entrainment.

2.2 Blended Cements for Durability

In the chapters 2.1.1.to 2.1.5 specifications set limits for clinker components, i.e. maximum C_3A and C_3S content, maximum Na_2O equivalent, as reflected in the **ASTM C150** specification for Portland cement.

But in all of these cases, the simple "dilution" of clinker with a suitable pozzolan or a slag, when not simply the addition of limestone, will yield the same effect in terms of reduction of the content of the sensitive clinker component.

Additionally, the use of a mineral component like slag, pozzolan or fly ash will improve the durability and the pore structure of hardened concrete by lowering its lime content and porosity as a consequence of the enhanced development of calcium silicate hydrates.

Then, the ASTM **C595** and **C1157** deal with blended cements and set limits referred to the performance of cement with respect of sulfate resistance and heat of hydration, without any prescription on the basis of cement composition.

If we look at these durability aspects in the context of the European standard **ENV 197-1**, we can immediately realize that most of the Portland composite cements of the **II/B** type, with addition of 21 up to 35% of non-clinker materials, as well as type **III** (slag), type **IV**

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(pozzolan) and type **V** (composite) cements can be considered as "special" with respect to many properties mainly related to durability.

Nevertheless, a CEN working group is now drafting new ENV 197 parts specifically dealing with special cements, in particular low heat- and sulfate resisting cements.

2.3 High Early Strength Cement, Rapid Hardening Cement

High early strength cement (HES) and rapid hardening cement (RHC) may either be finer than ordinary cement or have a special clinker composition.

Especially in the past, production of HES/RHC cements often required the production of a special clinker, with high C_3S and C_3A contents.

In present times, the improvements gained in grinding technology and in the use of *quality enhancers* as grinding aids, combined with the need of rationalization for storage and transports in cement plants, allow for the production of high early strength cements based on the same clinker used for ordinary binders. The higher grinding fineness and the related enhanced reactivity usually require increased gypsum dosage.

Physical properties of HES/RHC cements in comparison with ordinary Portland cements are shown in table 1. It should be noted that an increase in fineness would often also reflect in higher late strengths.

Table 1 Physical Properties of High Early Strength and Ordinary Portland Cement

Type	Blaine (cm^2/g)	Initial setting time (minutes)	Compr. strength MPa (ISO) 2 days	28 days
OPC	3000 ÷ 3500	150 ÷ 240	10 ÷ 20	38 ÷ 43
High early strength	3800 ÷ 4500	120 ÷ 180	20 ÷ 30	48 ÷ 55
Rapid hardening	4800 ÷ 5500	90 ÷ 150	30 ÷ 40	58 ÷ 67

The use of HES/RHC cements is indicated when a rapid strength development is required, e.g. if formwork has to be removed or re-used after a short time (precast elements production) or when sufficient strength is required for further construction (slipforming).

However, since rapid strength gain is usually associated to high rate of heat devlopment, HES/RHC cements should not be used in mass concrete or in large structural sections. On the other hand, concreting at low temperatures would profit from the use of a high heat cement as a safeguard against early frost damage.

2.4 Fast Setting Cement (Vicat Cement)

A particular type of fast setting cement is the so called Vicat cement from the name of its inventor and main producer. It is obtained by burning at low temperature (1200 to 1250 °C) selected natural marls with rather high alumina contents. The clinker produced is mainly composed of belite and aluminates; after grinding it is capable to set in a few minutes, developing sufficient strength to fulfil requirements for easy and fast repair or small construction jobs.

"Artificial" fast setting cement can be produced by blending OPC and High Alumina cement (see § 2.12) approximately at a 9:1 ratio.

2.5 Sulfo-aluminate Cement

Cements having calcium sulfo-aluminate ($C_4A_3S^*$) as main component. Examples of this binder type are type K cement produced in USA and the so called third cement series (TCS) in India and China. Typical composition ranges are reported in Table 4.

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Characteristics of these cements are

* reduced setting times
* high workability, low water demand
* rapid strength development
* low shrinkage (even expansive)

Hydration of $C_4A_3S^*$ leads to the formation of ettringite, which is mainly responsible for early strength development. Microstructure of ettringite depends on the presence of lime. When ettringite forms in presence of lime it provokes expansion and this property is used to produce no-shrinkage cements. In the absence of lime, ettringite is not expansive and mainly contributes to strength development.

Binders of this type can then be used, according to mineralogy, to prevent shrinkage in concrete or to develop high early strength for special applications.

The blend of TCS cements with OPC yields to flash setting.

Table 4 Chemical and mineralogical composition of Sulfo-aluminate Cements

	Sulfo-aluminate clinker (type K)	**TCS sulfo-aluminate cement (SAC)**	**TCS ferro-aluminate cement (FAC)**
SiO_2	2 - 4	3 - 10	6 - 12
Al_2O_3	45 - 49	28 - 40	25 - 30
Fe_2O_3	1 - 2	1 - 3	5 - 12
CaO	37 - 39	36 - 43	43 - 46
SO_3	7 - 10	8 - 15	5 - 10
CaO free	0 - 0.3	0 - 0.3	0 - 0.3
mineralogical composition			
$C_4A_3S^*$	55 - 70	55 - 75	35 - 55
CA	15 - 20	-	-
C_2AS	15 - 20	-	-
C_4AF	0 - 5	3 - 6	15 - 30
C_2S	-	15 - 30	15 - 35

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2.6 Regulated Set (Regset) Cement

Modified Portland cement type with rapid setting and hardening characteristics. The active component is a calcium fluoroaluminate (11 CaO. $7Al_2O_3$. CaF_2).

The regset cement is characterized by a setting time of 15 to 60 minutes. The setting is controlled by the addition of retarders (citric acid). Additional characteristic property is its unusually high early strength. A comparison between strengths achieved with Regset and HES cement is reported in Table 2.

Table 2 Properties of Regset and High Early Strength Cement

EN method	Regset Cement	H.E.S. Cement
Compr. Strength MPa		
2 hours	6	0
8 hours	8	2
16 hours	9	15
2 days	10	35
7 days	18	47
28 days	35	60
Setting time minutes		
Initial	45	120
Final	75	180

The very early strength gain makes this cement useful for all applications where there is a vital need for short times between placing and hardening of concrete, i.e. shotcrete, repairs on airport runways or highways, pavements.

Regset cements are batched, handled and mixed in pretty well the same way as Portland cements, making however the necessary provisions for rapid handling and placing of concrete. Although production is as cheap and easy as for Portland cement, the fast and often unpredictable setting behaviour puts serious limits to the use in constructions and the total world production is estimated in the order of 10000 tons/year.

2.7 Oil-well cements

Oil-well cements are developed for use in oil and gas wells and are designed to set and cure at high temperatures and pressures in well grouting. They can also be used for sealing water wells, waste disposal wells and geothermal wells. Cement plays an important part in the successful drilling of a well. It is used primarily to seal the annulus between the walls of the borehole and the steel casing, to isolate the pressured or weak zones encountered whilst drilling.

The oil-well cement must possess the following properties:

* low permeability
* form a good bond between rock and casing
* maintain these properties under downhole temperature and pressure conditions
* protect the casing against corrosion and collapse

To achieve these aims, the cement slurries must stay pumpable for sufficient time to permit placement, give stable suspensions, harden rapidly once in place and retain high strength and low permeability during well lifetime.

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API specification 10 from American Petroleum Institute classifies eight different oil-well cements. Other national standards exist, e.g. in Russia, China, India, most Eastern European countries, but so far no EN standard has been drafted. In Table 3 an overview of API 10 cements is compiled.

Table 3 Overview of oil-well cements

Class	Typical use
A	from surface to 1830 m, special properties not required, ASTM type I
B	from surface to 1830 m, moderate or high sulfate resistance
C	from surface to 1830 m, high early strength
D	between 1830 and 3050 m, moderately high T and p conditions, moderate or high sulfate resistance
E	between 3050 and 4270 m, high T and p conditions, moderate or high sulfate resistance
F	between 3050 and 4880 m, extremely high T and p conditions, moderate or high sulfate resistance
G and H	from surface to 2440 m, for use with accelerators or retarders, moderate or high sulfate resistance, class H coarser than G
J	between 3660 and 4880 m, pure OPC, extremely high T and p conditions

Such cements need special test procedures for suitable characterisation and additional care during manufacture so as to ensure consistency of quality between batches of the same plant.

In use, they are frequently mixed with additives in various proportions to produce satisfactory slurry performance for given well conditions, so they should additionally be compatible and responding to these additives (Table 4).

Table 4 Common types of additives for Oil-well Cements

Classification	Function	Example
Accelerator	Reduces thickening time of cement	$CaCl_2$, sea-water
Retarder	Lengthens thickening time	Lignosulfonates, sugars
Dispersant	Improves flow properties of cement	Superplasticizers
Lightweight extender	Improves stability of suspension	Bentonite, various clays
Weighing agent	Improves density of slurry	Haematite, barite, sand
Lost circulation controller	Prevents cement losses through strata	Walnut shells, cellophane flakes, expanded clay
Inhibitor of strength retrogression	Prevents loss of strength and formation of low strength silicate hydr.	Silica flour, silica sand
Fluid loss controller	Controls rate of water loss	polymers, cellulose
Defoamers	Removes foaming during mixing	Lauryl alcohol, glycols

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2.8 White cement

One of the most important special Portland cements. It is characterized by a white colour, obtained with suitable selection of raw materials, in which the colouring elements iron, chromium and manganese must be kept at the lowest possible level. Reduced formation of melt phase during burning is sometimes compensated by addition of fluoride as mineralizer.

Mechanical properties of white cements are comparable to those of OPC. Lower strength binders are produced intergrinding (up to 35%) high purity limestone. To improve whiteness, use of "optical whiteners" such as methylene blue in cement grinding can be applied.

White cement is used for architectural purposes in white or coloured concrete. To achieve best results, a properly coloured aggregate has to be used.

2.9 Hydrophobic cement

A small amount of a water-repellent agent (stearic acid, oleic acid, a.s.o.) is added to Portland cement during grinding. This forms a protective coating around each cement particle that retards hydration until cement is mixed with water and prevents deterioration during storage, especially in humid countries.

2.10 Masonry Cements

Masonry cements are successfully used in the production of mortars and plasters for most non-structural building purposes.

Their main characteristics are: low early- and late strengths, low shrinkage, limited water permeability, high water retention, excellent plasticity and cohesiveness in the fresh state, frequently also air entrainment for better workability and freeze-thaw resistance.

These properties are achieved by a low to medium clinker content, use of a suitable limestone and/or other mineral additions, use of air-entraining agents.

MC's are mostly produced by intergrinding Portland cement clinker, gypsum, precrushed limestone and/or other mineral additions. When the grindability of the constituent materials is much different, it might be opportune to grind them separately, especially if the following blending facilities are available.

The main advantage from the production point of view is the limited clinker content (may be as low as 25%) and the availability of the limestone in the plant. Production

25 Holderbank Group plants produce masonry cements in an extremely wide range of compositions and properties, reported in the following table 5 (based on 1996 Annual Technical Report).

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Table 5 Characteristics of Masonry Cements in the Holderbank Group

Masonry Cement Characteristics	avg.	min.	max.
Clinker content, % [1)]	54.2	24.0	81.5
Specific surface acc. to Blaine, cm^2/g	6470	3765	9420
Residue on a 45-µm sieve, %	8.9	1.7	25.7
Residue on a 90-µm sieve, %	5.5	1.1	9.5
Air entraining agent, dosage g/t	1210	91	4000
Sulfate content, % SO3	1.93	0.62	3.00
Water demand, % [2)]	26.3	21.5	30.2
Initial setting time, minutes [2)]	190	113	383
Compressive strength 7 days, MPa [2)]	12.8	3.7	25.6
Compressive strength 28 days, MPa [2)]	15.2	3.4	29.5
Specific milling energy consumption, kWh/t	65.6	34.3	132

[1)] main other constituent is limestone, but also fly ash and pozzolana are used

[2)] figures related to both ASTM and EN test methods are considered

2.11 Ultrafine Cements (Microcements)

These binders are characterized by a narrow and steep particle size distribution (see figure 1), specially designed for injection grouts used for sealing and improving mechanical properties of porous systems (rocks, damaged concretes, soils).

Due to the reduced particle size, water-microcement suspensions are highly stable and penetrating than OPC grouts. Results obtained using microcement injections are comparable to those of chemical products such as resins and soluble silicates, with the advantage of being more environment friendly.

Microcements are usually Portland or blast furnace slag cement based and are produced by ultrafine grinding or subsequent efficient separation of finer fractions from common cements.

Particle size distribution may range between 0 and 10 -15 µm. Due to this, they also develop much higher early strength than the corresponding common cement , but usually require higher water demand. This is not a drawback in most applications, since for injection grouts a W/C ratio of 1.5 to 3 is quite in the normal range.

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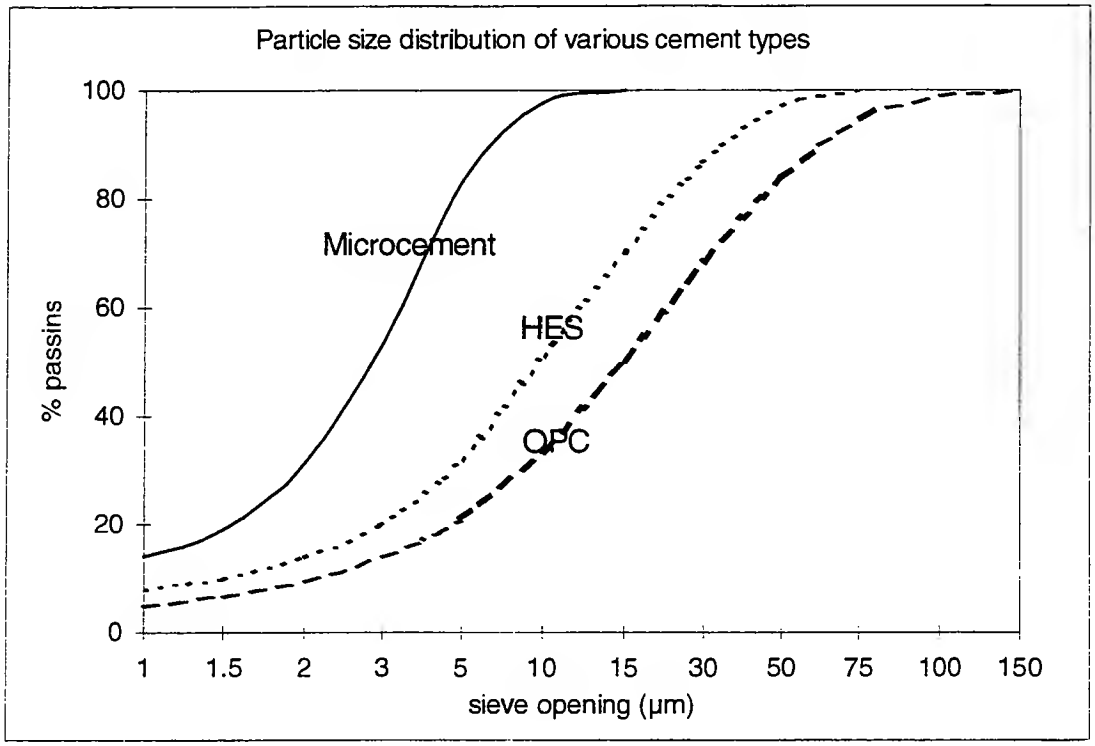
Figure 1 Particle size distribution of various cement types



As mentioned, their main use is in fields where common cements cannot be used due to their limited fineness characteristics

- agglomeration of loose soils
- sealing of microcracked rocks
- restoration of foundations, tunnels, leaching dams, historical buildings.

2.12 High Alumina Cement

Typical non-Portland cement, high-alumina cement (HAC) produces a concrete which has an exceptionally fast rate of hardening and is resistant to attacks by most sulfate solutions. It also has a higher resistance against acidic solutions than Portland cement based concretes, but it does not resist the attack of caustic alkali.The strength development of HAC is demonstrated in Figure 2. About 80% of its ultimate strength is achieved after only 24 hours. The high rate of strength gain of HAC is due to the rapid hydration of the anhydrous calcium aluminate (CA).

Rapid hardening is not accompanied by fast setting. Initial setting takes place between two and six hours after mixing, becuase of the slow setting pattern of hydration of the main compound CA.

Setting time can be shortened by addition of OPC. HAC/OPC blends are used in applications where rapid setting is compulsory, but lower ultimate strengths are then obtained.

Inversely, the same effect on setting is achieved by replacement of 10 to 20% HAC to OPC.

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The high rate of heat evolution (Figure 3) of high-alumina cement (ab. 40 J/g.hour in the first 24 hours) makes it necessary for HAC concrete to be placed in thin sections and never in large mass. The rise in temperature causes cracking and adversely affects strength. HAC concrete needs also more water for hardening and better curing during the first two days than OPC concrete.

The main drawback of high-alumina cement concrete is the loss of strength associated with expansion due to the conversion of the aluminate hydrates under moist conditions at elevated temperatures. The hexagonal hydrate CAH_{10} is converted into the cubic aluminate C_3AH_6 as from the reaction

$$3\ CAH_{10} \rightarrow C_3AH_6 + 2\ AH_3 + 18\ H$$

This means that concrete which is properly placed and has developed a high strength will lose a considerable proportion of its strength upon exposure to temperatures over 30 °C and moisture.

Figure 2 Strength development of different cements

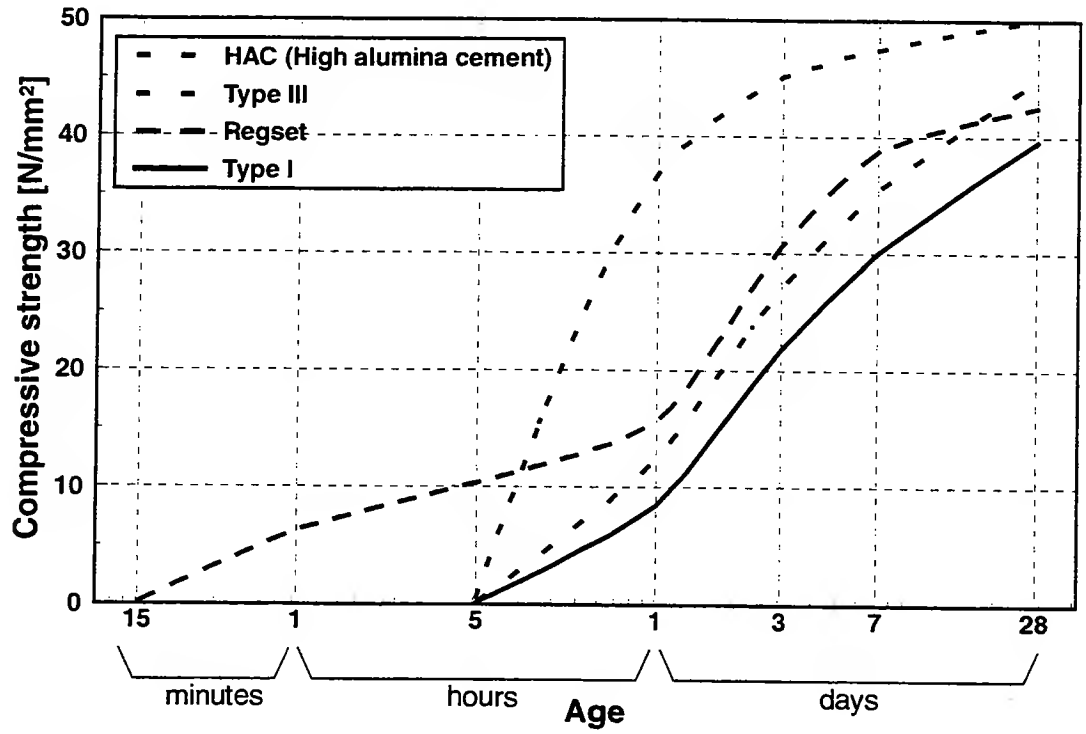

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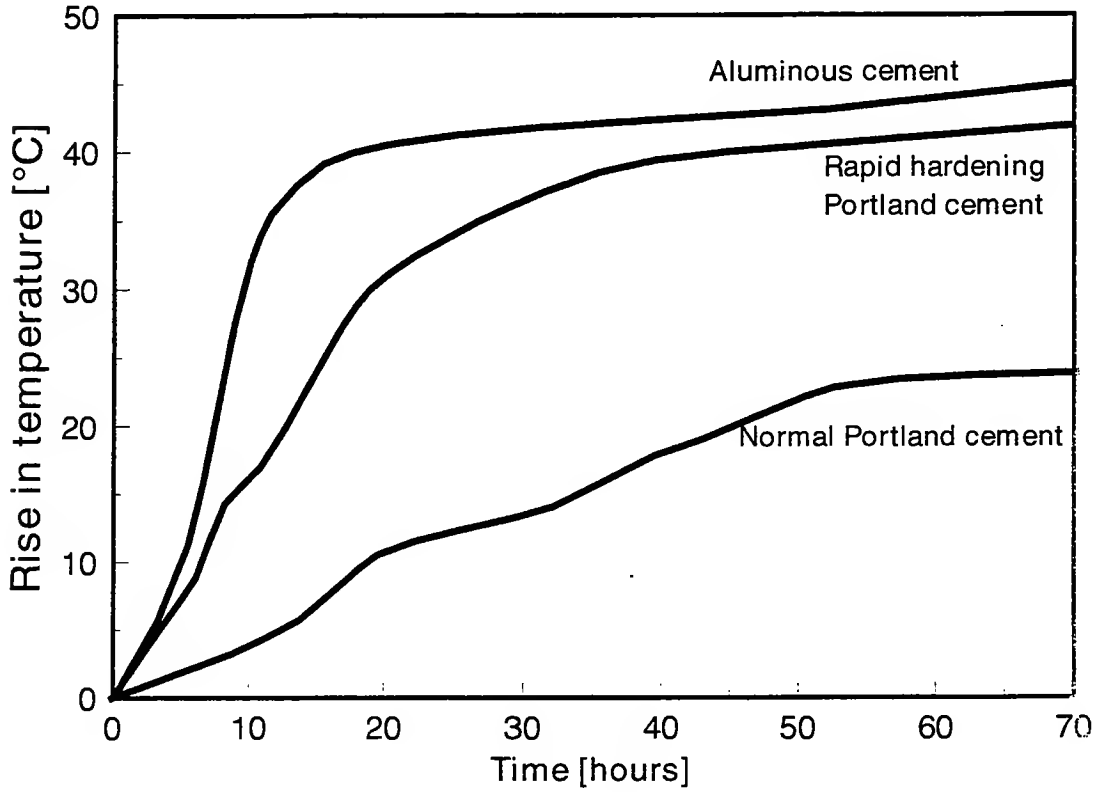
Figure 3 Heat evolution of different cements



2.13 Phosphate cements

A mixture of tetracalcium phosphate and dicalcium phosphate, when mixed with a dilute phosphoric acid or other aqueous solutions will harden like a cement-producing hydroxyapatite, $Ca_{10}(PO_4)_6HOH$, as the final product. The setting property, combined with biocompatibility, makes calcium phosphate cements useful in many applications in dentistry and medicine.

Pastes prepared from diammonium orthophosphate, as well as NaH_2PO_4 or Na-polyphosphate, and calcined MgO exhibit a fast setting and hardening at room temperature associated with NH_3 liberation. MgO-phosphate binders may be used for high temperature applications, since strength remains preserved up to 1000 °C.

2.14 Other non-Portland cements

2.14.1 Alkali-activated cements

"Pyrament " (developed in USA) belongs to this type of binders; they are mainly composed of pozzolans and possibly other active silica-based materials such as silica fume, fly-ash, slag or Portland cement, and alkali compounds (alkali silicate, hydroxide or carbonate); water reducers and retarders are also added to set regulation. They display excellent early strength properties as well as high density and strength at later age, but the complicated multicomponent formulation indicates a very sensitive system, difficult to control under field conditions.

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Development of "Geopolymers" is based on the work of Davidovits. The main reaction takes place between sodium hydroxide and kaolinite to form hydrosodalite at room temperatures or preferably between 150 and 180 °C. This reaction is the typical hydrothermal synthesis of zeolites.

2.14.2 Alkali-activated slags

The ability of activating blast furnace slags (BFS) by the addition of alkalis has been known for many decades. Hydration of slag requires the breaking of bonds and dissolution of the three-dimensional structure of glass and this is easily achieved in the high pH environments produced by alkali. Research on slag activation dates back to the early 50's; interest in alkaline activation has grown markedly and in recent years alkali-activated cement and concrete have received greater attention worldwide.

Compared with ordinary Portland cement and interground slag cement, alkali activated slag cement has some advantageous properties, including rapid and high strength development, good durability and high resistance to chemical attack.

Finely ground, well granulated BFS can be utilized in the production of cements suitable for the precast industry, after addition of alkaline activators and superplasticizers, as for the "F" cement (Finland). When used with thermal curing, these cements can develop strengths 30% higher than normal Portland cements.

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3. OVERVIEW OF PRODUCTION IMPLICATIONS IN SPECIAL CEMENTS

In the following table 5, the implications related to the production of special cements are summarized. In particular, after highlighting the main properties and applications of each special cement type, a list of special measures that need to be taken at the plant to achieve the expected properties is compiled.

Table 4 Properties , application and measures for proper production of special cements

Type	Properties	Application				
			Raw mat	Man ufact	Addi tions	App licat ion
OPC	normal workability and stength	general application in building and construction	-	-	-	-
ASTM II / SRC/LHC	moderate sulfate res. low heat of hydration	drainage, large piers and retaining walls, sea water	x	-	(x)	-
ASTM III / HES-RHC	high early strength, rapid hardening	precast concrete, repairs, cold weather concreting	-	x finer	-	-
ASTM IV / LHC	low heat of hydration	mass concrete, large dams	x	-	x	x
ASTM V / SRC	sulfate resistant	severe sulfate action in (soils, ground water)	x	(x)	x	x
Air entrain.	frost resistance	roads, freeze-thaw action	-	-	x	-
Low alkali	low reactivity with amorphous silica	in concrete with reactive aggregates	x	x	x	-
Leaching resisting	low content of hydrat. lime	in presence of pure waters or high CO_2 solutions	x	-	x	-
Oil well	retarded setting, moderate sulfate res.	in oil, gas and other types of wells	x	-	(x)	x
Regset	fast setting time	road repairs, dry mortars	x	x	-	x
White cem	white colour	architectural concrete	x	xx	(x)	x
Sulfoalum. cement	fast setting, high workab., (expansive)	precast concrete, repairs, no-shrinkage concrete	x	x	-	x
Ultrafine	max size < 15 µm	injection grouts	-	xx	(x)	xx
High alum. cement	high early strength, rapid strength devel.	demand of very high early strength, refractories	x	xx	-	xx

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Chapter 3

Blended Cement

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BLENDED CEMENT

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1. INTRODUCTION

A blended cement is a binder that contains - in addition to Portland cement clinker and calcium sulfate - one or more of the following materials:

- a latent hydraulic material (granulated blast-furnace slag)
- a pozzolanic component (natural pozzolana, fly ash, silica fume, burnt clay, a.s.o.)
- a filler (usually limestone).

1.1 Latent Hydraulic Materials

Latent hydraulic materials have a natural hydraulic potential (i.e. they can develop silicate hydrates by hydration reactions), but require an activator such as:

- Lime
- Portland cement clinker
- Gypsum
- Chemical activator

in order to accelerate the reaction with water. Blast-furnace slag is the most typical latent hydraulic material.

1.2 Pozzolanic Materials

They can give rise to the formation of silicate hydrates only in presence of lime (calcium hydroxide, $Ca(OH)_2$), as a consequence of its reaction with the amorphous (glassy) silica contained in the pozzolana. The hydration reaction of Portland cement clinker with water will provide the necessary lime to make the pozzolanic reaction happen.

Natural pozzolanas, generally of volcanic origin, and low-calcium fly ashes are the most commonly used materials belonging to this group.

1.3 Inert Additions (fillers)

Inert additions possess no hydraulic or pozzolanic activity, but contribute to cement properties other than strength development.

Limestone filler is the most commonly used, particularly in the production of fillerized cements, with up to 35% clinker replacement, and in the manufacture of masonry cements (up to 75% limestone).

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1.4 Alternative terminology

Depending on the idioms of the place of use, the reference standards and the materials involved, blended cements can be also named differently, e.g.

* composite cements
* fillerized cements
* slag cements
* pozzolanic cements, a.s.o.

The materials, besides clinker and gypsum, used as constituents for blended cements can also be named, according to the nature and performancein cement, as

* mineral components
* mineral additions
* mineral additives
* mineral admixtures
* fillers
* cement extenders.

2. SIGNIFICANCE OF BLENDED CEMENTS

The principal reason for the production of blended cements is economic and environmental, although in most cases blended cements perform better than Portland cement.

Blended cements allow an economy of fuel and an increase of cement production due to the lower clinker/cement factor. During the last thirty years, a significant increase in the use of alternative materials in the manufacture of blended cements, or as additions in concrete, has been recorded owing to unquestionable technical advantages, such as:

* improved durability (beneficial for service life of concrete structures)
* reduced heat of hydration (beneficial for casting of thick sections)
* reduced alkali level of the blend (to minimize the risk of alkali-silica reaction)
* improved characteristics of fresh concrete.

Possible disadvantages, such as

* slower rates of strength gain at early ages (and at lower temperatures)
* higher sensitivity to poor curing

are largely overcome by the advantages gained in the overall performance.

Production of blended cements, then, has effect on the

- reduction and optimization of production costs by
  - improved clinker factor
  - saving on thermal and electrical costs
  - increased cement output,
- activities related to environment protection and preservation by
  - conservation of natural fuels and mineral resources
  - use of by-products as mineral components
  - reduction of CO_2 and heat emission,
- differentiation of products and applications by
  - opportunity to produce tailored products
  - improvement of cement properties for special applications
  - fulfillment of durability issues in concrete construction.

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2.1 History

Slags have been known since ancient civilizations. It has been illustrated that ancient Egyptians produced glass slags, rich in vesicles (air bubbles) and containing crystals of the minerals melilite and tridimite, by burning wheat straw. This contains opal, an amorphous form of silica, that combines during burning with alkali to form glassy slag.

More than 2000 years ago, the Romans discovered the activity of the volcanic ash from Pozzuoli when it was mixed with lime. This was used in many structures that lasted through the centuries and are still existing today, such as aqueducts, bridges, historical buildings. Vitruve, architect and engineer in the 1st century AC was the first who tried to explain the reactions that transform some rocks into cementitious materials.

More recently, the production of blended cements has been practiced for many decades in many countries. History of use of blast furnace slag (BFS) goes back more than 100 years when the iron and steel industries in England, Germany, Czechoslovakia, etc. operated their own cement factories. Cements with natural Pozzolanas as an addition have been firmly installed in the Italian and Greek markets for decades, and are used in practically all applications involving concrete construction.

2.2 Present situation

The observed growth of the production of composite cements in the recent years is roughly indicated in the following table 1:

Table 1 Production of Composite Cement in the World

Addition → Situation	**Slag**	**Ash**	**Pozzolan**	**Limestone**
established	EC countries Latin America East. Europe Japan South Africa	Europe Oceania South Africa	Italy Turkey Latin America Greece	Europe Morocco
increasing	Australia Brazil Canada India	Canada Far East	Middle East Far East	East. Europe New Zealand

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3. OVERVIEW ON MINERAL COMPONENTS

3.1 Classification of mineral components

As already mentioned in the definitions, secondary constituents of a blended cement can be

- (latent) hydraulic materials
- pozzolanic materials
- inert materials (fillers)

The following Table 2 summarizes the types of constituents used in the production of blended cements.

Table 2 Constituents of Cement

1.	**Principal constituents**		
	. clinker	-	various clinker types
	. set regulator	-	gypsum, anhydrite
		-	gypsum by-products
		-	limestone
2.	**Secondary constituents**		
	. hydraulic materials	-	blast furnace slag
		-	other by-product slag
	. pozzolanic materials	-	natural pozzolan
		-	fly ash, bottom ash, silica fume
		-	artificial pozzolan
	. fillers	-	limestone, marl, siliceous sand/ rock
		-	kiln dust
3.	**Chemical additives**	-	grinding aids
		-	quality improvers

3.2 Types of Mineral Components

Suitable materials for blended cement production can be natural or artificial. An artificial component is either derived from industrial production as a by-product, or can be obtained by thermal activation of natural materials or by-products.

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The following Table 3 gives a concise overview of these materials.

Table 3 Different types of mineral components

SLAGS	POZZOLANAS	
artificial	**natural**	**artificial**
* granulated blast furnace slag	* volcanic ashes	* activated natural materials
* all other rapidly cooled slags	* tuffs	* activated industrial by-products
	* opaline cherts and shales	
	* diatomaceous earth	
	* rhyolites, etc.	

Activated natural materials	Activated industrial by-products
- calcinated clays	- pulverized ashes of lignitic or anthracitic coal
- calcinated volcanic rocks	- bottom ash
- metakaolins	- industrial SiO_2 rich residue (silica fume)
- Kalsin	- calcinated red muds
- calcinated shales	- calcinated wastes
- etc.	- pulverized clay bricks or tiles

3.3 Chemical Composition

- Although these materials have very different properties in many respects, they have one thing in common: they consist essentially of the same major chemical elements as PC clinker: **SiO_2 - Al_2O_3 - CaO**.

The respective proportions regarding the CaO - SiO_2 - Al_2O_3 diagram are shown in Table 4 and represented in Fig.1.

Table 4 Chemical Composition of Hydraulic Additions (on the basis of dried and calcinated materials)

Elements	Blast Furnace Slag	Natural and/or Artificial Pozzolan	Class F Fly Ash
SiO_2	30 - 40	50 - 75	40 - 65
Al_2O_3 (TiO_2)	8 - 25	15 - 25	15 - 40
Fe_2O_3	0.5 - 1.5	3 - 10	3 - 17
CaO	35 - 45	1 - 15	1 - 10
MgO	1 - 18	< 5	- 3
SO_3	1.5 - 6	0.1 - 1.5	0.3 - 3
K_2O	0 - 0.5	0.5 - 4	2 - 3
Na_2O	0 - 0.5	0.5 - 4	0.4 - 2
MnO	0.2 - 2	-	< 0.3

Most slags, Pozzolanas and fly ash must be dried before utilization.

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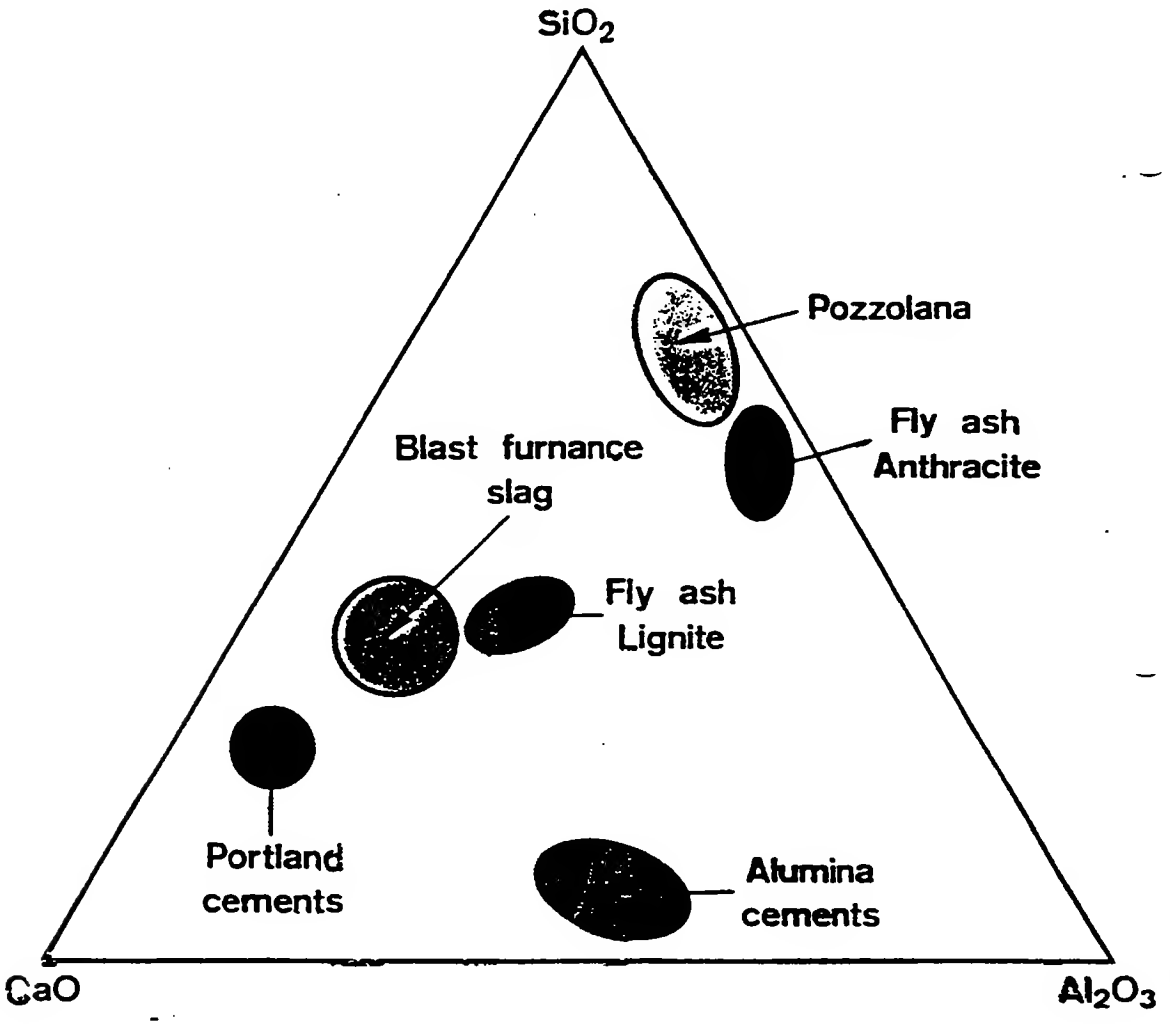
Fig. 1 CaO-SiO-AlO-System



3.4 Hydration Aspects

Both slag and pozzolan must be activated for adequate hardening.

Slag has a high silica and calcium oxide content. If activated by OH^- or SO_4^{--} ions, it displays hydraulic activity (cementitious properties). In practice, the activation is realized by addition of lime, Portland cement clinker (OH^-) or gypsum (SO_4^{--}).

Silica, and alumina at a lower degree, are the main components of pozzolan, whereas the calcium oxide is low. Pozzolan supplies the silica (or alumina) that reacts with added lime or with the lime (Portlandite) produced by hydration of Portland cement clinker.

When analyzing the calcium hydroxide content on mortar as a function of time, it can be observed (see Fig. 2) that the quantity of CH decreases significantly for a Portland pozzolan cement containing 20% of pozzolanic material, compared to pure Portland cement.

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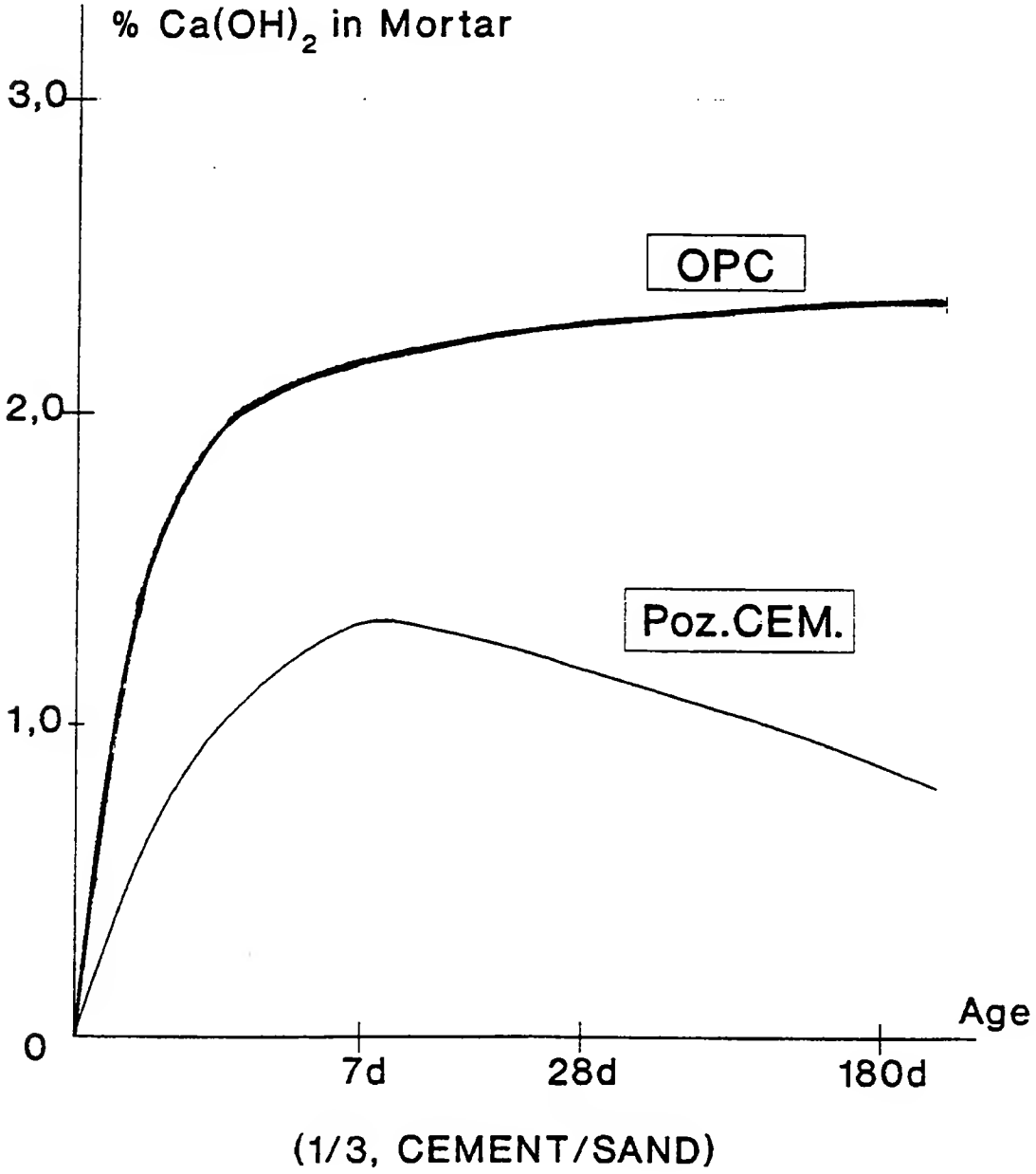
Fig. 2 **Calcium Hydroxide Content of Pozzolanic Cement and Portland Cement Mortar**



3.5 Standards for mineral components

The increased use of mineral components in cement and concrete brought to the development of specifications describing the characteristics of the different materials and the testing methods applied to evaluate suitability of use in cementitious blends.

These features are usually enclosed in the main cement standards (e.g. ENV 197/1 on common cements, ASTM C-595 on blended cements). When the mineral component can be directly used in the mix design of concrete, specific standards are usually drafted.

A comprehensive description of national and international specifications for blended cements and their mineral constituents is given in the chapter on cement standards.

As an example, some of them are listed below.

- **ASTM C 311** "Sampling and Testing Fly Ash or Natural Pozzolanas for Use as a Mineral Admixture in Portland-Cement Concrete"
- **EN 450** "Fly Ash for Concrete"
- **ASTM C 989** "Specification for Ground Granulated Blast Furnace Slag for Use in Concrete and Mortar"
- **ASTM C 1240** "Specification for Silica Fume for Use in Hydraulic Cement Concrete and Mortar".

As concerns the question whether it is more advisable to use mineral components in the manufacture of blended cements rather than in the preparation of concrete mixes, the theoretical answer is quite simple. The important thing is that both concretes, the one prepared with a blended cement and the other prepared by addition of the mineral component directly into the concrete mixer, prove to be equivalent in performance. From a practical point of view, facts in favour of the use of a blended cement are a well established quality control on materials, availability of large storage facilities that smoothen quality fluctuations, possibility to adjust the cement composition and production parameters to the desired final binder performance.

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3.6 Testing methods

The most currently used testing methods for the determination of the quality of an addition are:

- Chemical analysis including heavy metals and insoluble residue
- Mineralogical analysis by means of XRD and microscopy
- Calorimetric methods such as DTA, DSC or TGA
- Particle shape (microscopy)
- Density
- Fineness and particle size distribution (Blaine, sieve analysis, laser granulometry)
- Grindability ('Holderbank' grindability test)
- Activity index and pozzolanic activity determination:
  - Keil Index (hydraulic)
  - Strength Activity Index (SAI) according to ASTM C 311
  - CEN pozzolanic activity test
- Test on standard laboratory cement
- Specific accelerated mortar test, etc.

A combination of 2 or 3 of the above methods is generally necessary for the current quality control of acceptance of an addition.

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4. MAIN MINERAL COMPONENTS FOR PRODUCTION OF BLENDED CEMENTS

4.1 Blast furnace slag - other slag types

Slags are obtained as by-products in industrial processes of the metallurgic industry.

When metal is extracted from its ore and refined by a further process, slag of various qualities are obtained. The most important slags are those of the iron-melting operations, above all the blast furnace slag. These slags can be considered as the molten silicate complex, formed by a combination of agglomerated earthy matter with the ore, fuel and fluxes either in, or added to the charge. All the slags are molten in the furnace hearth. The chemical and mineralogical composition of the slag varies according to the raw materials and the refining process.

4.1.1 Production of Blast Furnace Slag (BFS)

In the blast furnace, iron ore is converted into pig iron. The slag floats on the top of the liquid iron and is drawn off through a notch (see Fig3).

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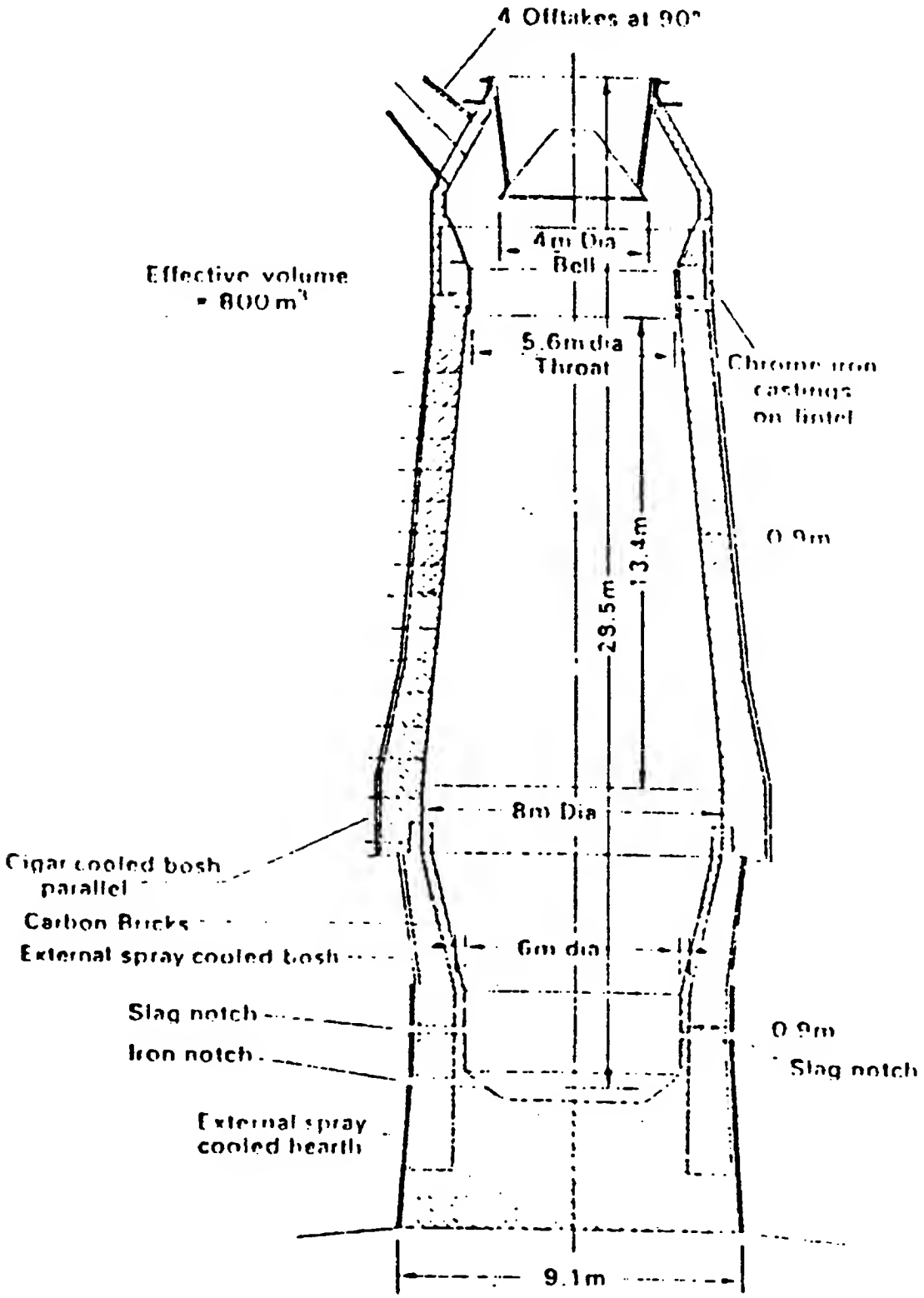
Fig. 3 Sectional Diagram of a Modern Blast Furnace



Blast furnace slag consist of secondary constituents of the iron ore, the combustion residues of coke and the necessary additions such as limestone or dolomite.

The iron ore is mainly a mixture of oxides of iron, silica and alumina. The chemical reactions within the blast furnace reduce the iron oxide to iron. The silica and alumina compounds react and combine with the calcium added as a fluxing stone (limestone and dolomite) to form the slag. The chemical reactions occur at temperatures between 1300 and 1600°C.

The limestone and dolomite are calcinated at this temperature, and the resulting calcium and magnesium oxides are combined with the silica and alumina of the iron ore to form a slag. Compounds of lime-silica-alumina and magnesia are formed in a similar way as during the clinkering process in a cement kiln.

To ensure a constant quality of iron, the feed of the furnace must be uniform and therefore the composition of the slag varies only slightly as well as in a good conducted melting process. For an economic production of iron, it is essential that the ratio of slag volume to pig iron volume is as low as possible. Modern plants should have an average ratio of 200 to 300 kg slag per ton of iron.

The density and porosity of the slag depends on the conditions of cooling, as well as on its chemical composition. According to the cooling process, different slag types can be distinguished (see Fig.4).

Fig. 4 Forms of Blast Furnace Slag

Air-cooled, crystalline BFS is hydraulically not active and therefore is used as an aggregate or filler.

The high energy of the slag melt is completely dissipated in the case of slow cooling, and a stable crystalline product is formed with properties similar to those of basalt. It is used in road base constructions or as concrete aggregate. Crystalline slags have very weak or no cementitous properties.

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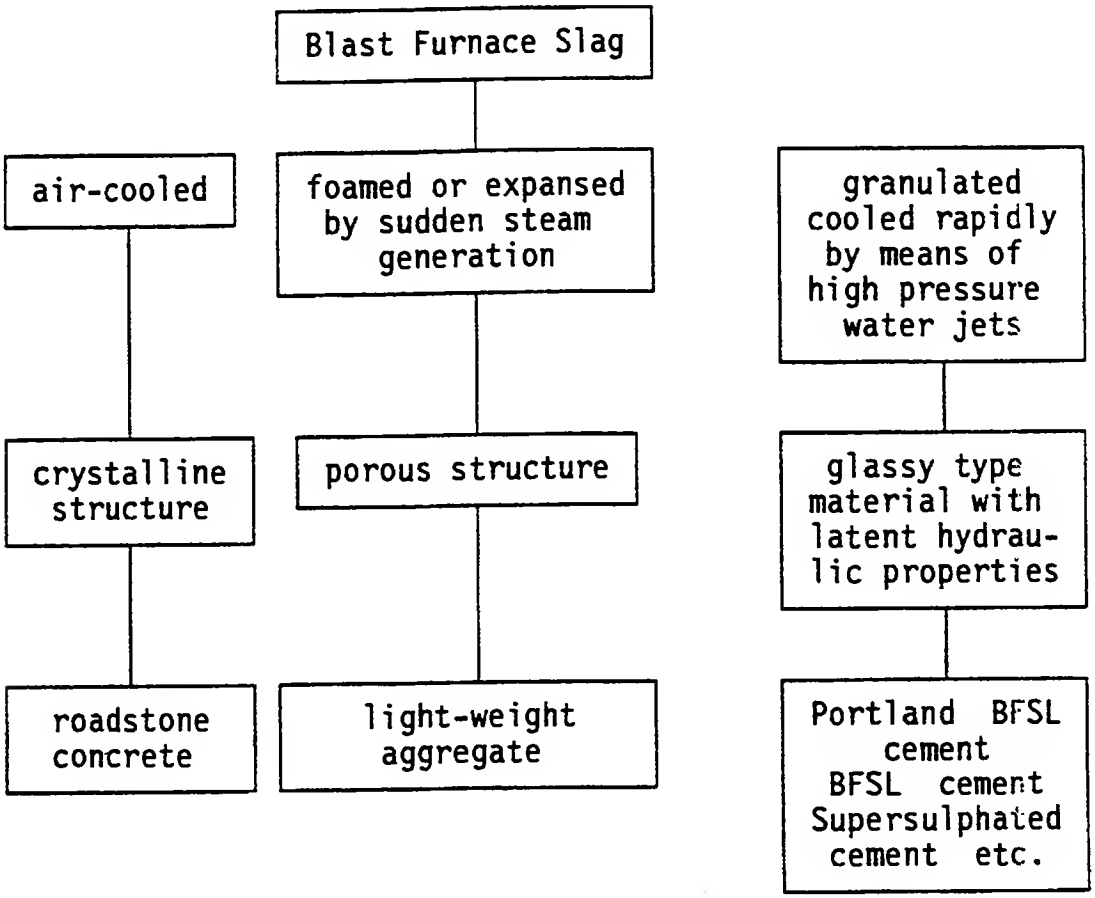
Granulated, rapidly cooled, glassy slag has latent hydraulic properties, i.e. it can be activated so that a cement is obtained.

If the slag is cooled rapidly, a slag glass, richer in energy and with cementitious properties is obtained.



Today, there are basically two methods of rapid cooling of slag:

- Granulation
- Pelletization

4.1.1.1 Granulation Method

By the traditional well-established granulation method, the slag is first divided into small droplets, up to 5 mm in size, which subsequently are brought into contact with air or water. The use of compressed air has the advantage for the cement industry that a dry product is obtained, however, a lower glass content in slag results. Therefore, granulation is mostly done with water, whereby a quantity of up 10 m^3/t of slag is required. By applying spray jets, only about 3 m^3 water per ton of slag are necessary. It is important that the cooling rate is high (down to 800°C) in order to maintain the glassy state of the slag. In the granulation process, the water content may be undesirably high (up to 30%), the excess water is then partially removed during storage in gravel filter basins. The final moisture content is usually still too high (≥ 10%) for direct grinding with clinker at proportions above 20%, thus a slag drier has to be installed.

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4.1.1.2 Pelletization Method

The pelletization method was advanced in Canada. In this procedure the molten slag is pre-cooled with water and hurdled into the air by a rotary drum to separate it into two different fractions. The larger pellets (up to 15 mm) which have a lower glass content are used as light-weight aggregate.

The smaller fraction (up to 5 mm) with a higher glass content is reported to be suitable for the production of blended cements. The advantages of this process are the lower water consumption and the lower residual water content of the slag (less than 10%). The hydraulic activity and the grindability do not differ from that of granulated slag.

4.1.2 Properties of Slag

It is generally agreed that in the manufacture of slag for cementitious purposes high CaO content is beneficial to the development of strengths. However, the CaO/SiO_2 ratio can only be increased to a limited degree, since under constant operating conditions, increased CaO content results in an increased slag viscosity necessitating higher operating temperature, which in turn affects costs.

At constant basicity, the higher alumina content has a favorable effect on the strength development of activated granulated slag. Iron and manganese oxide and sulfides generally have an adverse effect on strength development.

Magnesium is not present in form of periclase, in either of the slag types, and therefore does not affect the soundness of the hardened concrete. MgO content up to 20% are permissible in slag.

The increase of silica content reduces grindability, and the presence of metallic iron increases the milling time.

4.1.3 Chemical Characteristics and Properties

Many researchers have attempted to define the hydraulicity of slag in terms of chemical formulae based on the observed effects of the various oxides on the compressive strengths of slag cements.

The usefulness of calculated empirical formulae is questionable and the only fixed line guide is the strength development in cement.

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The main proposed moduli relating to BFS in the manufacture of Portland cement are:

- I - according to DIN 1164

$$\frac{CaO + MgO + Al_2O_3}{SiO_2} \geq 1$$

- II - EN Standard

$$\frac{CaO + MgO}{SiO_2} \geq 1$$

- III - Keil

$$\frac{CaO + CaS + ½\,Mg) + Al_2O_3}{SiO_2 + MnO} \geq 1.5$$

$1.5 \leq F \leq 2.0$ good hydraulic properties

$1.0 \leq F \leq 1.5$ poor hydralic properties

- IV - Langavant Index

$$i = 20 + CaO + Al_2O_3 + ½MgO - SiO_2$$

$i \leq 12$ poor hydraulic properties

$12 \leq i \leq 20$ acceptable hydralic properties

$i \geq 20$ excellent hydraulic properties (slag difficult to granulate)

- V - Basicity Index

$$1.20 \leq CaO / SiO_2 \leq 1.5$$

- VI - Silica-alumina Ratio

$$1.8 \leq \frac{SiO_2}{Al_2O_3} \leq 1.9$$

Lea has commented on the usefulness of such empirical formulae: 'Chemical composition moduli are convenient for the rapid control of slag quality, but the only fixed guide is the strength development in cement'.

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4.1.4 Mineralogical Composition

The chemical composition of slag is not significant enough to predict their behaviour.

Much more important than chemistry, is the mineralogy of slag, and principally the proportion between the crystalline and the glassy phase. A good slag shall contain at least 2/3 by mass of glassy slag, according to EN standard.

The following methods are applied to determine the mineralogical properties:

- Microscopy
- X-ray diffraction (XRD)
- Differential thermal analysis

By differential thermal analysis, vitreous slag shows irreversible exothermic peaks at about 860°C, when heated between 800 and 900°C. These peaks are mainly due to the heat of devitrification of the glassy part of the slag. Some authors tried to find a correlation between the areas under the peaks and the slag content.

By X-ray diffraction, crystalline materials produce strongly marked peaks. The following Fig.5 gives the pattern of a crystalline and an amorphous slag.

The most commonly occurring minerals found in blast furnace slag are given in Table 5.

Table 5 Commonly occurring Minerals in Blast Furnace Slag

Mineral	Formula	Analysis (% by mass)			
		SiO_2	Al_2O_3	CaO	MgO
Gehlenite*	$2CaO . Al_2O_3 . SiO_2$	21.9	37.2	40.9	
Akermanite*	$2CaO . MgO . 2SiO_2$	44.1		41.1	14.8
Wollastonite	$CaO . SiO_2$	51.7		48.3	
C_2S	$2CaO . SiO_2$	34.9		65.1	
Rankinite	$3CaO . 2SiO_2$	41.7		38.3	
Merwinite	$3CaO . MgO . 2SiO_2$				
Anorthite	$CaO . Al_2O_3 . 2SiO_2$	43.2	36.6	20.2	
Monticellite	$CaO . MgO . SiO_2$	38.4		35.8	25.8
Spinel	$MgO . Al_2O_3$				

* These compounds form a continuos series of solid solutions known as melilite

The most common mineral is melilite, a name applied to any of a series of solid solutions from akermanite, which is represented by 2CaO . MgO . 2SiO, to gehlenite 2CaO . AlO . SiO. In slag of low lime and high alumina content, the mineral anorthite CaO . AlO . 2SiO may appear.

In slag with a high lime content, the CS may be formed: This compound occurs in three different forms: α, β, γ. The α-form occurs at temperatures above 1420°C.

When the temperature drops to about 675°C, the CS occurs in the β-form. At atmospheric temperature, the β changes into the γ-form, increasing a volume of about 10%. If the silicate is present in sufficient quantities, this expansion causes disintegration of the slag matrix. To avoid this, the lime has to be limited (generally, limitation of the CaO/SiO ratio to 1.4 in some countries).

The specific gravity of crystalline slag lies between 2.38 and 2.76, the bulk density between 1150 and 1440 kg/m^3.

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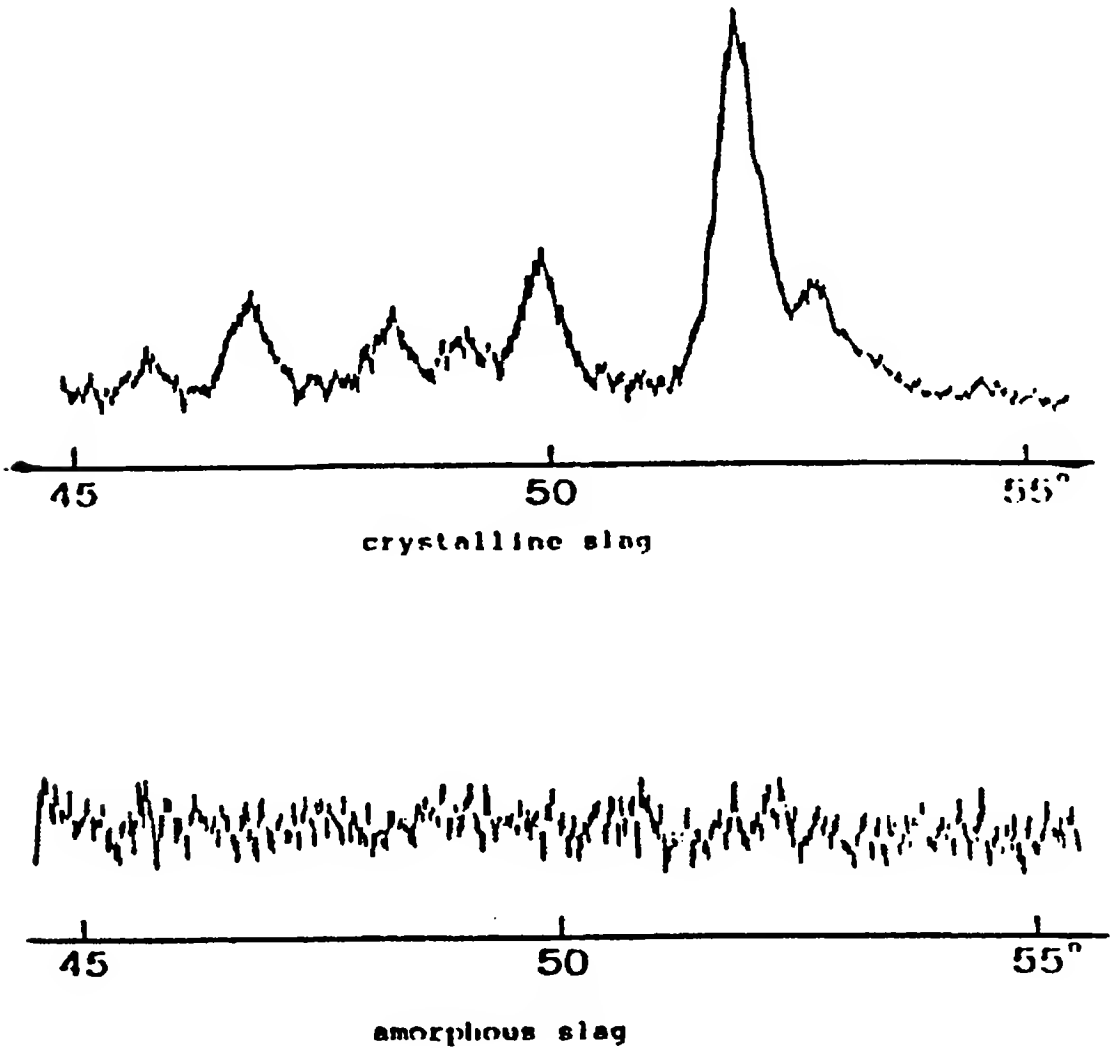
Fig. 5 XRD of Slag



4.1.5 Other Types of Slag

Other types of slags than iron blast furnace slag have been introduced in the EN standards and are admitted for the production of Portland blended cements, with a maximum content of 15%.

The chemical composition of slags produced as a by-product in the metallurgical industry varies within a very wide range and depends on their origin. Typical analysis from different metallurgical industries have been compiled by Smolczyk and are given in Table 6.

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Table 6 Characteristic Composition of Metallurgical Slags

	Non-ferrous Slags				Ferrous Slags	
	Lead Zinc U.K.	Nickel Canada	Copper S. Africa	Phosphorus Furnace USA	Steel BOF (LD Slag) Germany	Iron Blast Furnace Europe
SiO_2	18	29	34	41	13	34
CaO	20	4	9	44	47	41
MgO	1	2	4	1	1	7
Al_2O_3	6	1	6	9	1	13
$FeO_x + MnO_x$	38	53	41	1	31	1
CaO/SiO_2	1.1	0.1	0.3	1.1	3.6	1.2

4.2 Fly ash - Coal ash

4.2.1 Production of fly ash

Fly ashes are obtained by electrostatic or mechanical precipitation of dust-like particles from the flue gases from furnaces fired with pulverized coal.

The pulverized coal is injected into the furnace in a stream of air sufficient for the combustion of coal. The remaining particles after burning of the coal are collected in the electrostatic precipitator in form of a cement like powder as fly ash. Part of the ash is collected at the bottom of the furnace as bottom ash (see Fig. 6).

Fly ashes may be silicoaluminous or silicocalcareous in nature. The former have pozzolanic properties, and the latter have, in addition, hydraulic properties.

Fly ash is a mixture of mineral components which have undergone thermal transformations and which contain still unburnt materials. The main component is an aluminium-iron melt, in which the silica and other oxides are dispersed in a molecular form.

According to the burning temperature of the coal, three types of fly ash may be distinguished:

- Type I: Burning temperature 800 to 1000°C
  - crystal structure remains
  - about 20% of the fly ash is melted
  - poor pozzolanic properties
- Type II: Burning temperature 1200 to 1400°C
  - most minerals are melted (about 70%)
  - about 60% of melted particles are deposited before reaching the precipitator, only 40% get to the filter
  - slow reaction with Ca(OH), only medium pozzolanic activity
  - grain size distribution similar to cement
  - can replace 10 to 30% of cement
- Type III: Burning temperature 1500 to 1700°C
  - all minerals are melted
  - grain size distribution similar to cement
  - rapid cooling of the fly ash induces the production of 60 to 80% of glassy particles

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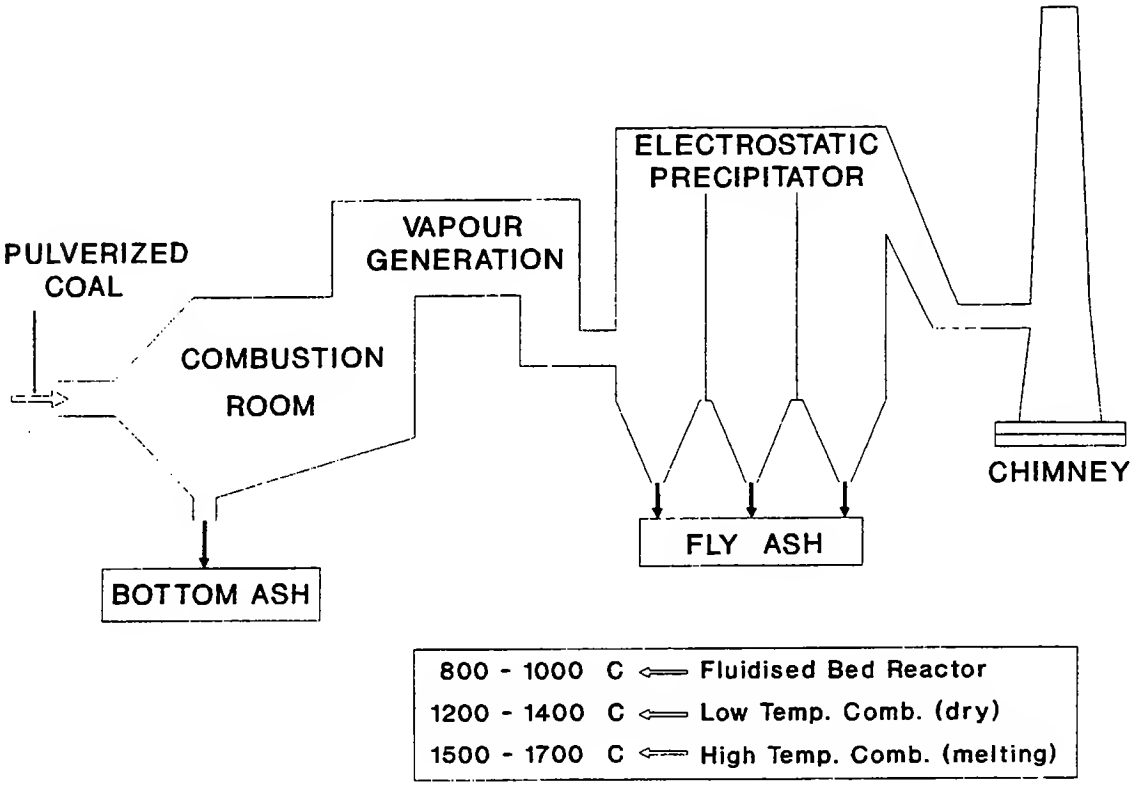
- development of good pozzolanic properties

It is generally known that the volume of fly ash produced in most countries exceeds largely the volume that can be utilized, but statistics are generally unavailable.

Fig. 6 Coal Fired Power Station



4.2.2 Properties of Fly Ash

The ash quality depends greatly on the type of coal, type of process and actual operation of the furnace. The operating conditions as well as the coal quality strongly influence the variability of the ash with respect to residual carbon content, composition and pozzolanic activity.

Fly ash burnt at high temperatures are best suited for the production of blended cements, where the pozzolanic activity is important. Other uses of fly ash, where pozzolanic properties are less important, are:

- Light-weight aggregate
- Foamed concrete
- Macadam, etc.

4.2.3 Chemical and Physical Properties

The main chemical components of ashes are SiO_2, Al_2O_3, Fe_2O_3 and CaO. Substantial amounts of alkalis and sulfate may also be present. In the case of lignite or brown coal, the observed amount of free lime as well as of anhydrite is generally high. Significant amounts of periclase can also be noticed in lignitic fly ash. The typical aspect of fly ash observed by microscopy is given in Phototable 1, while Fig. 7 shows the grain size distribution obtained by laser granulometer.

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Phototable 1 Thin Walled Hollow Sphere of Fly Ash (Scanning Electron Micrography)

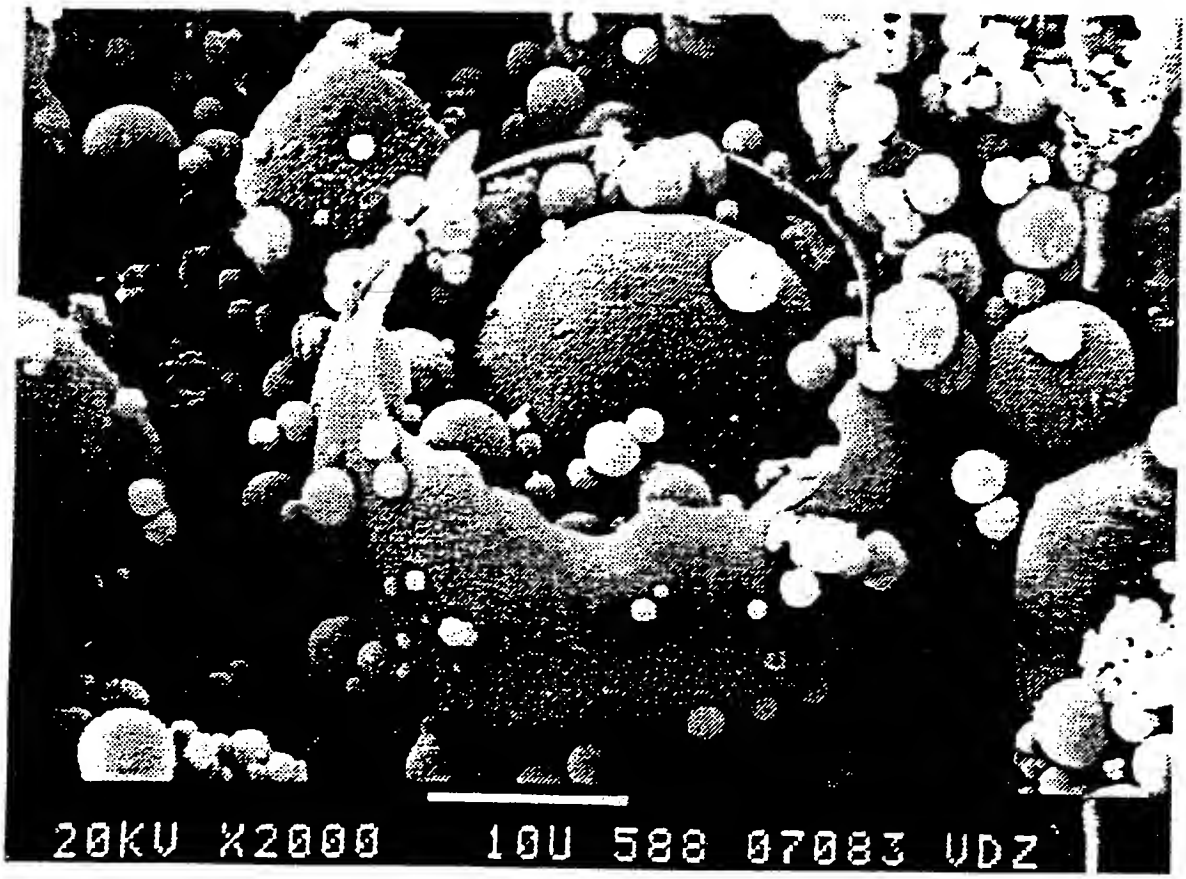
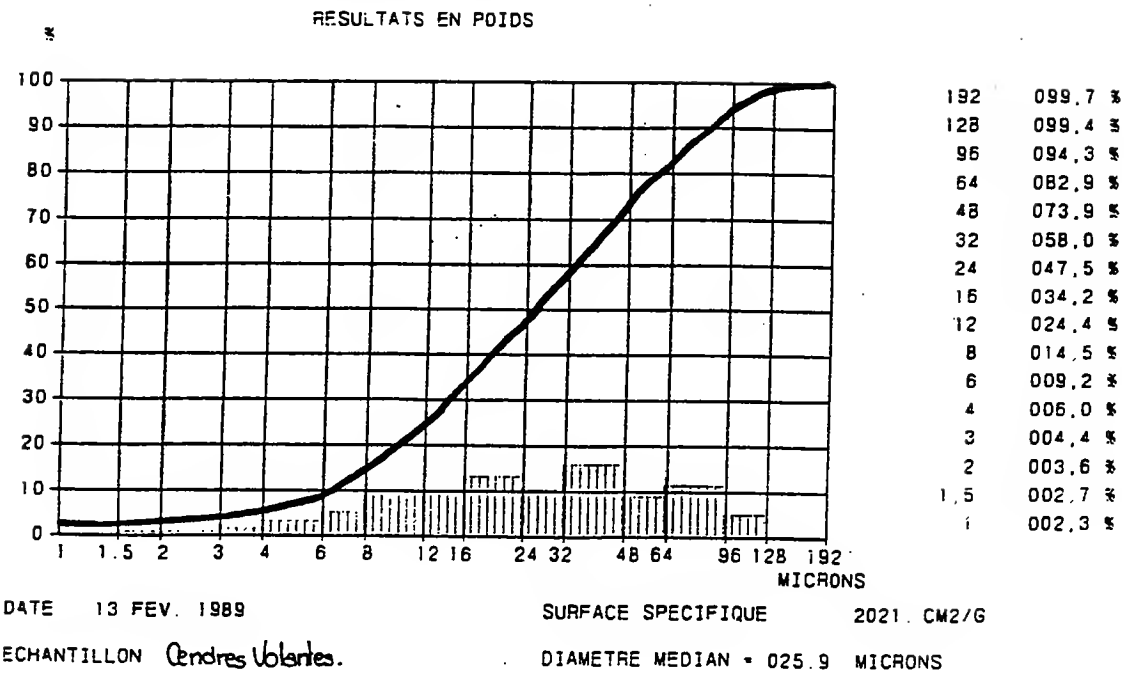


Fig. 7 Granulometry of fly ash



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4.2.4 Mineralogical Properties

The mineralogical composition of fly ashes depends on the amount of glassy material. The following minerals can be observed (Table 7), depending on the operating conditions and type of coal. Moreover, calcareous fly ash possess hydraulic properties, due to a more or less greater amount of dicalcium silicate CS.

Table 7 Mineralogical Composition of Fly Ash

Minerals	Quantity % in weight
Glass	70 - 98
Quartz	1 - 15
Mullite	1 - 10
Magnetite	1 - 8
Hematite	1 - 5
Calcite	< 2
Plagioclase	< 2
Merwinite	< 2
Melilite	< 2
Anhydrite	1 - 8
Free lime	1 - 8
Periclase	1 - 6
Wollastonite	1 - 5
Dicalcium silicate	1 - 15

4.2.5 Carbon Content

The carbon content is an important criterion to assess quality of fly ash. Residual carbon is due to the non-burnt organic matter and lies generally in the range of:

- 1 to 12% in anthracitic fly ash
- 1 to 4% in lignitic fly ash

The variations in C-content are mainly due to instabilities of the burning system, to variations in burning intensity or to changes in coal supply.

A high carbon content (> 5%) is considered detrimental for the quality of concrete since it might

- impair the air-entrainement in freeze/thaw resistant concrete
- have a negative effect on workability and strength development of concrete

From the manufacturing point of view, it is worthwile mentioning that most of the unburnt material is agglomerated and concentrated in the coarse fraction of fly ashes. Recent developments in the technology of fly ash benefication allow to improve quality by mechanical or electrical separation of the organic matter, making it possible to use high-carbon fly ashes (separated unburnt coal can be used as additional fuel for clinker burning).

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4.2.6 Classification

Fly ash can be classified according to EN 197 or to ASTM C 618 in the following classes of Table 8.

Table 8 Classification of Ashes

Characteristics	ASTM	EN
Fly ash from anthracite or bituminous coal	Class F	
Siliceous fly ash		Class V
Fly ash from lignite or subbituminous coal	Class C	
Calcareous fly ash		Class W

4.2.7 Requirements

The requirements does not significantly differ between ASTM and EN standards. Table 9 summarizes the ASTM chemical requirements, whereas those for EN are given in Table 10.

Table 9 ASTM Chemical Requirements

Specifications	Class F	Class C
- Silicon dioxide (SiO_2) + Aluminium oxide (Al_2O_3) + iron oxide (Fe_2O_3), in % weight	70.0	50.0
- Sulfur trioxide (SO_3), max. %	5.0	5.0
- Moisture content, max. %	3.0	3.0
- Loss on ignition, max. %	12.0	6.0
Supplementary optional chemical requirements (apply only when specifically requested)		
- Magnesium oxide (MgO), max. %	5.0	5.0
- Available alkalis, as NaO_2 equiv., max. %	1.5	1.5

Table 10 ENV 197-1 Requirements

Specifications	Class V	Class W
- Loss on ignition, max. %	5.0	5.0
- Reactive CaO, %	< 5.0	> 5.0
- Reactive SiO_2 and Al_2O_3, %	> 25	
- Expansion (Le Châtelier)		< 10 mm
- Compressive strength: 30% W FA + 70 OPC		> 10 N/mm^2

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4.3 Pozzolanas

4.3.1 Definition

Pozzolanic materials are natural or industrially activated products.

Pozzolanas are siliceous or siliceous and aluminous materials which in themselves possess little or no cementitious properties. When finely ground, they react in the presence of water at ambient temperatures with dissolved calcium hydroxide $Ca(OH)_2$ from lime or Portland cement clinker to form strength developing calcium silicate and calcium aluminate compounds.

Natural Pozzolanas are mostly of volcanic origin and they can be found in various parts of the earth. The Pozzolanas where named after the Italian town Pozzuoli, near Mt. Vezuvius. Other well-known Pozzolanas are the German 'trass' and the Greek 'Santorin earth'. But Pozzolanas are obviously available in a great number of other countries.

The Roman used Pozzolanas together with burnt lime as building materials, hardening either in air, or even under water. Many remainders from that time prove the good quality of these materials. When Portland cement was invented (in the 19th century), the Pozzolanas fell into oblivion, but today, due to the high cost of fuel, the pozzolan regained their importance.

There are two great families of Pozzolanas:

- Natural Pozzolanas
- Artificial Pozzolanas

The natural Pozzolanas are mainly of volcanic or metamorphic origin or of other highly siliceous materials such as diatomaceous earth.

Typical examples of natural Pozzolanas are:

- Volcanic ashes or pumicites
- Diatomaceous earth
- Opaline cherts and shales
- Tuffs, phonolites, zeolites, etc.

The artificial Pozzolanas are produced by means of thermal treatment from non-pozzolanic material (e.g. marl, shale) or from material with very low pozzolanic properties (e.g. phonolite). When exposed to a rise of temperature, depending on the nature of the raw material (150°C to about 1500°C), they are activated (crystalline structure is modified and glass if formed) and therefore develop pozzolanic properties.

Clay-bearing Pozzolanas, altered volcanic ashes and tuffs as well as shales needs a calcination at temperatures between 500 and 1000°C, in order to be activated.

4.3.2 Classification of Natural Pozzolanas

A tentative classification of the natural Pozzolanas according to the nature of the reactive constituents is difficult. They can, however, be distinguished in four great groups. An aspect of natural pozzolan by SEM is given by Phototable 2.

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4.3.2.1 Glassy Volcanic Rocks

Those rocks consist of loose or consolidated rocks of volcanic origin, the predominant component being unalterated glass. The reactive glass originates from explosive eruptions where abrupt cooling took place preventing crystallization (e.g. pumice). The vitreous particles have a very high internal surface promoting reactivity. The presence of alkalis favors a more open glass structure and thus, a more reactive type of pozzolan. Volcanic ashes also belong to this group, the minute particles having been quenched in the atmosphere.

4.3.2.2 Zeolitic Volcanic Rocks

Zeolitic volcanic rocks consist of loose or consolidated materials of volcanic origin. The reactivity is mainly due to the inclusions of zeolites or to the partially zeolized glass which is very sensitive to any type of reaction.

This group comprise also the highly siliceous material which has suffered considerable chemical alterations due to circulation of gases and aqueous solutions in the deposit. Natural leaching causes a very high internal surface.

4.3.2.3 Mixed Pozzolanas

Mixed Pozzolanas consist of one of the above mentioned types, mostly mixed with clayey material and other inert minerals.

Gaize and Moler are typical representatives of this group. An increase in their reactivity is mostly obtained by a thermal treatment at low temperature.

4.3.2.4 High Silica Pozzolanas

High silica Pozzolanas are a reactive form of hydrated silica containing mainly skeletons of living organisms such as radiolarites or diatoms. They lead to an extremely porous structure. They are often the host rocks for petroleum and are used as hydraulic additions after extraction of the fuel. A thermal treatment at low temperature is generally recommended for a considerable increase of their reactivity.

4.3.3 Chemical and Mineralogical Composition

Pozzolanas - natural as well as artificial - consist essentially of reactive SiO_2 and Al_2O_3. The remainder contains Fe_2O_3 and other oxides. They generally contain also a significant amount of alkalis.

It is not possible to calculate meaningful modulis on the basis of chemical analysis, because the chemical parameters which determine the pozzolanic activity vary in a wide range.

The current control of the quality of a pozzolan can be achieved by applying a combination of three different methods:

- Chemical composition
- Mineralogical composition
- Physical properties

A strict classification is difficult and the Streckeisen-diagram (see Fig.) is the most appropriate document allowing a classification and nomenclature of the different natural Pozzolanas. Table shows the great variety of minerals found in the different types of Pozzolanas tested by 'Holderbank' and used in the different plants of the Group.

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Pozzolanic activity of natural pozzolan of volcanic origin depends mostly on their:

- Glass content
- Content of zeolitic minerals.

The best results were obtained with Pozzolanas containing:

- Silica rich glass
- Zeolites such as chabazite, heulandite, natrolite, stibite and wairakite

4.3.4 Standard Specifications

It is obvious that no basis exists for taking the content of any single constituent, or a quantitatively defined physical or chemical characteristic of such a constituent, as a criterion for judging the pozzolanic reactivity of a material.

The main specifications are summarized in the following Table 11 for ASTM. The determination of pozzolanicity of a pozzolanic material is carried out by applying the ASTM C 311 and in Europe the EN 196-5.

Table 11 ASTM Chemical Requirements

Specifications	**Class N**
- Silicon dioxide (SiO_2) + Aluminium oxide (Al_2O_3) + iron oxide (Fe_2O_3), min % in weight	70.0
- Sulfur trioxide (SO_3), max. %	4.0
- Moisture content, max. %	3.0
- Loss on ignition, max. %	10.0
Supplementary optional chemical requirements. These optional requirements apply only when specifically requested.	
- Magnesium oxide (MgO), max. %	5.0
- Available alkalis, as NaO_2 equiv., max. %	1.5

Several tests have been proposed to determine the activity of pozzolan. These tests are based on an extraction with a mineral acid or an activation with lime or Portland cement. None of them can be considered as a prediction of the compressive strength of concrete containing pozzolanic material.

The conclusion is that the pozzolanic activity is measurable only by strength development of a blended cement or of a concrete mix in the provided proportions.

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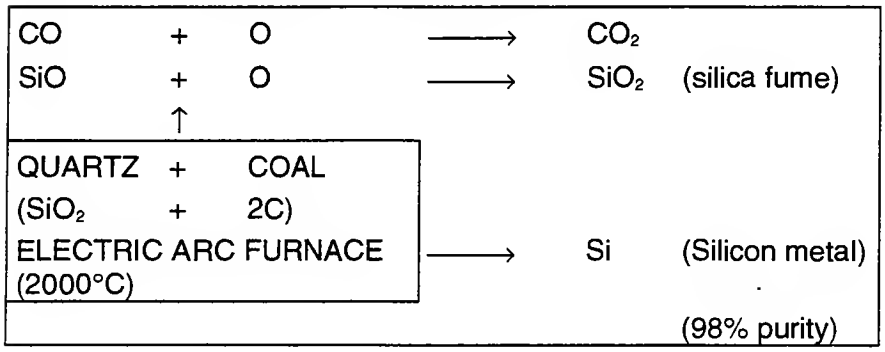
4.4 Silica fume - Other specific pozzolanic materials

As previously indicated, a great number of other industrial by-products or activated materials can be used for the production of blended cements. This paragraph shortly reviews some of them.

4.4.1 Production and Characteristics

Silica fume is a by-product of the manufacture of silicon (see Fig. 9). Silicon is used for the production of ferrosilicon, which is required by the steel industry for the improvement of its products.

Fig. 9 Production of Silica Fume



Condensed silica fume consists of very fine spherical particles with a high content of amorphous silica. It is an extremely fine powder, very much finer than Portland cement or fly ash, its fineness being roughly comparable with the finest particles in tobacco smoke (Table 12).

Table 12 Fineness of Different Materials

Type of Material	Fineness (cm^2/g)
Silica fume	approx. 200'000
Tobacco smoke	approx. 100'000
Fly ash	2'000 to 5'000
Normal Portland cement	approx. 3'000

The grain size distribution is shown in Fig. 10. The extreme fineness gives rise to a number of problems in handling and transporting this material. Its bulk weight is not higher than 300 kg/m^3. Exposed piles are stirred by the least breath of wind. Silica fume is either packed in bags as such, or densified and packed in bags, as well as in the form of a 50% water slurry. Transport and transshipment facilities clearly necessitate efficient dedusting systems.

The chemical composition (see Table 13) may vary somehow depending on the impurities of quartz, the features of the manufacturing process, the dust precipitation facilities and the coal used. The effect of different compositions can become quite significant depending on the particular type of application.

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Table 13 Chemical Composition of Silica Fume

Elements	Range	
	Min.	Max.
Loss on Ignition	0.7	2.5
SiO_2	90.0	96.0
Al_2O_3	0.3	3.0
Fe_2O_3	0.2	0.8
CaO	0.1	0.5
MgO	0.5	1.5
NaO	0.2	0.7
HO	0.4	1.0
C	0.5	1.5
S	0.1	0.4

4.4.2 Properties and Requirements

4.4.2.1 Hydraulic Properties

Owing to the great fineness, its completely amorphous state and its high content of SiO_2, silica fume exhibits properties closely resembling those of pozzolan. *However, the usual methods for characterizing the pozzolanic activity are not suitable and they had to be first adapted to suit its unusual properties.*

4.4.2.2 Requirements

Only silica fume that meets the following requirements shall be used:

- amorphous silica SiO_2 $\geq$ 85% by mass
- loss on ignition $\leq$ 4% by mass
- specific surface (untreated) $\geq$ 15 m^2/g (BET)

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Fig. 10 Grain Size Distribution

KORNGROESSENVERTEILUNG

"HOLDERBANK"
VERFAHRENSTECHNISCHE ABTEILUNG

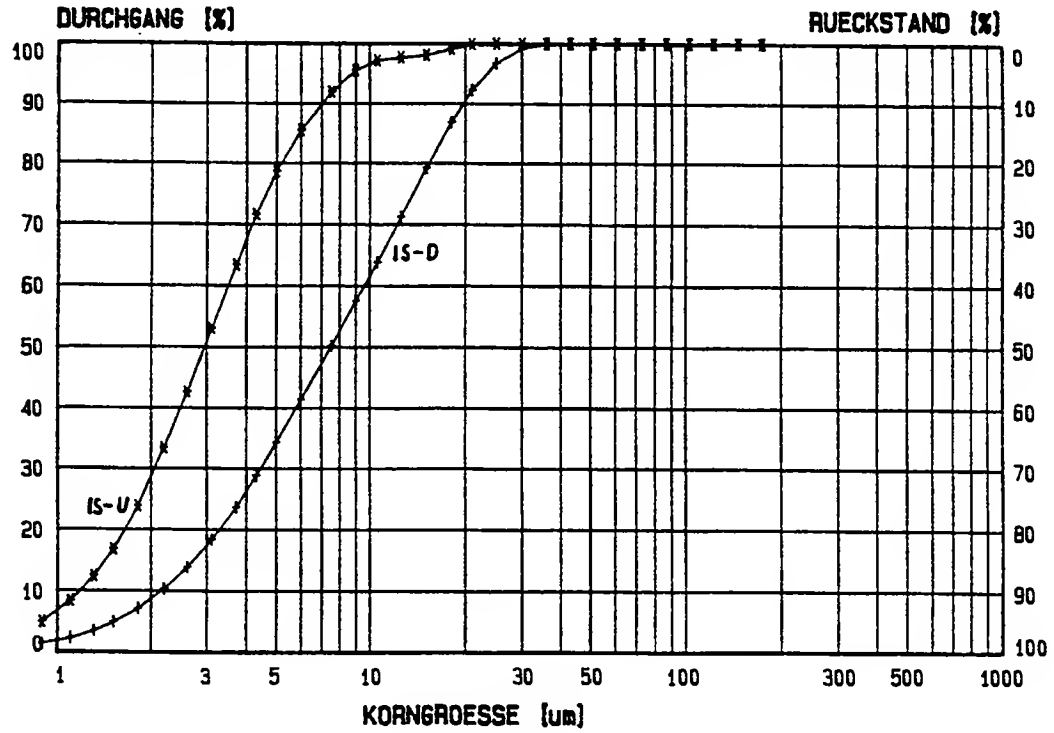
WERK : HOLDERCHEM
ANLAGE :
DATUM/VISUM : 31.03.92 Zg
ANALYSENMETHODE : SYMPATEC (100 mm)

TEST NR. : HCO-0416
AUFTRAG NR. : 33006/109

MATERIALBEZEICHNUNG			SILICA 1	SILICA 2		
SYMBOLE			✱——————✱	+——————+		
TEST NR.			1284 D	1268 W		
MATERIAL NR.			71507 - IS-U	71508 - IS-D		
PROBENMENGE		[g]	0.00	0.00		
FEUCHTE		[%H2O]	0.00	0.00		
DICHTE		[g/cm3]	0.00	0.00		
R1	%R	5.00 [um]	21.40	65.70		
R2	%R	10.00 [um]	3.33	38.30		
R3	%R	15.00 [um]	1.90	20.80		
DURCHMESSER	d (50%)	[um]	2.96	7.47		
SSB-OBERFL.	BLAINE	[cm2/g]	0.00	0.00		
SSC-OBERFL.	BERECHNETE	[cm2/g]	0.00	0.00		

BEMERKUNGEN : IS-U = Undensified

IS-D = Densified



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4.4.3 Initial Experience

The high pozzolanic activity made it seem obvious to add the silica fume directly to cement or concrete. Tests performed in a great number of laboratories proved that there are a number of problems connected with the handling properties of the silica dust, however, they can be overcome when suitable modifications to the process technology are made.

The level of replacement cement by silica fume is limited and lies generally in the range of 8 ± 2%. The following hardening characteristics (Table 14) have been observed for an ISO standard mortar with a pure Portland cement compared to a 5% silica fume substitution.

Table 14 Characteristics of an ISO Standard Mortar

Strength after ... Days	Pure Portland Cement	95% PC + 5% Silica Fume
2	20.3 N/mm^2	21.4 N/mm^2
28	42.8	54.1
90	53.3	61.2

The amount of water required by the binder is higher and the setting time is generally somewhat shorter compared to pure PC. The increase of strength is remarkable and conduct naturally to the use of silica fume for the production of high performance concrete. In order to reduce the higher water requirement, it is recommended to add superplasticizer in the concrete mix. In spite of its remarkably high price, this material is chosen in most cases where production of a high performance concrete is required.

4.5 Burnt Shale - Calcinated Marls and 'KALSIN'

Burnt shale, in special cases burnt oil shale, calcinated marls and Kalsin (a patented product of the 'Holderbank' Group) are produced in conventional rotary kilns as well as fluidized beds, at a temperature of approximately 800°C. Owing the composition of the natural materials and the manufacturing process, these products contain hydraulic mineralogical phases identical to those produced during the clinkering operation. Those phases are mainly dicalcium silicate and monoaluminate as well as reactive gehlenite and calco-spurrite. Due to the specific burning process (generally under reducing atmosphere) only small amounts of free lime is produced whereas calcium sulfate proportion depends essentially on the sulfur present in the raw material. Beside those hydraulic minerals, large proportions of pozzolanicly reacting oxides, especially silica and alumina are present.

In a very finely ground state, burnt shale, calcinated marls and Kalsin show like Portland cement, pronounced hydraulic and in addition pozzolanic properties.

Those products must develop, when burnt only - when tested in accordance with EN 196-1 - a compressive strength of at least 25.0 N/mm^2 after 28 days. If not they are classified in the category of artificial pozzolan.

The expansion shall be less than 10 mm (Le Châtelier test) in accordance with EN 196-3 in using a mixture of 30% by mass burnt material plus 70% by mass reference cement.

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4.6 Burnt clays

After calcination at a relatively low temperature (usually < 800 °C), some clays can be converted to pozzolanic materials, after production of reactive phases by alteration of the original crystalline structure of the clay. The required heat treatment is generally variable, since it depends on the nature and fineness of clays. Both the calcination temperature and time have to be adjusted to get best enhancement of pozzolanic activity.

5. ASPECTS OF MANUFACTURE OF BLENDED CEMENTS

5.1 Grinding of blended cements

For optimum production of blended cements, some important aspects related to the grinding process must be carefully evaluated

- pretreatment of additions
  - drying
  - preblending
- properties of the addition
  - grindability
  - strength activity
- type of grinding
  - compound
  - separate
- dosing of components.

5.1.1 Pretreatment of additions

The additive may have a considerable moisture content, and drying has to be done prior to or during the grinding process. The dosing facilities should be adequate to allow precise and trouble free operation even in case of very moist and sticky materials.

Depending on the nature and the number of the additions used, preblending of the materials would be necessary to achieve sufficient homogeneity of the component and of the cement itself.

5.1.2 Properties of the addition and type of grinding

5.1.2.1 Grindability

The grindability of the components is generally different, even in materials of the same type. Big differences might therefore be justified between different plants, and generally a specific solution has to be found for each plant.

Table 15 summarizes some data, taken out of the annual report, giving information on physical properties and the grinding energy needed for the production of blended cements containing slag, pozzolan or fly ash compared to Portland cement.

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Table 15 Average Physical Properties and Grinding Energy for Blended Cements

Material	Density [g/cm^3]	Blaine [cm^2/g]	Grinding Energy [kWh/t]
Clinker	3.18	3135	32.43
Slag	2.90	2980	34.17
Nat. Pozzolan	2.55	6150	20.92
Fly ash	2.45	4405	11.92

This table shows that slag is usually harder to grind than Portland cement clinker, while pozzolanic materials are mostly softer. The advantage of fly ash is that it can be added directly to the separator feed, since most of it is already of the required cement fineness.

This also implies that during grinding of a slag cement, clinker will be ground finer, thus contributing to early stregth development, while the slag remains coarser and will react lately. Thanks to its glassy nature, slag does not affect water demand and workability and can be used at very high addition rates.

In the case of pozzolanic cements, the very soft pozzolana will concentrate in the fine fraction of cement, thus affecting water demand, workability and early strength development. Proper selection of the grindng equipment (central discharge mills, vertical roller mills) will prevent overgrinding of Pozzolanas and improve cement properties.

When designing a new grinding plant for blended cement manufacturing, the process can be optimized to the requirements. Unfortunately, in most plants blended cements have to be produced in old grinding equipment designed for Portland cement production.

High fineness of silica fume and/or important variability of the fineness of fly ash may be responsible for severe problems of handling, transporting and proportioning. This type of material is generally introduced after the mill, in the output bucklett or directly in the separator, in order to provide a good mixing, avoid quality fluctuations and improve mill operation.

5.1.2.2 Strength activity

A mineral component is clearly expected to contribute to strength development of cement. Its strength activity will reflect on cement design and on the proportioning system. Typical dosage of a mineral component ranges between 25 and 40%.

5.1.3 Type of grinding

As already mentioned, grindability of the various components can influence the choice of the grinding system, especially in new plants.

Intergrinding of cement is easier from the point of view of plant design and operation, but requires accurate dosing and might be considerably disturbed by quality fluctuations of the different constituents, also in terms of grindability.

The advantage of separate grinding is that each component can be ground at the optimum fineness, thus enhancing its peculiarities and reducing drawbacks usually associated with unproper grinding.

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5.1.4 Use of chemical admixtures

Some of the drawbacks of blended cements can be compensated by the use of selected chemical admixtures.

Besides the simple addition of grinding aids especially in the case of high replacement rates with hard to grind materials, more sophisticated quality improvers can be applied

- water reducers/plasticizers
- early strength improvers
- multipurpose admixtures.

5.2 Proportioning

The most important quality requirement of blended cements are usually related to their performance in the different applications.

Some of these properties can be summarized as follows:

- Satisfactory rheological properties for good workability in concrete and mortar
- Acceptable setting behavior
- Adequate strength development and sufficiently high 28-day strength
- Satisfactory expectations for durability of the resulting concrete, as regards
  - residual porosity and permeability
  - resistance to carbonation
  - freeze-thaw resistance
  - resistance to chemical attack (sulfates, pure water, sea water, weak acids)
- Acceptance of resulting concrete color
- Reduction of Alkali Aggregate Reaction (AAR) risks
- Lower heat of hydration

and all specific properties providing better service or higher performances compared to ordinary Portland cement.

The design of composite cements requires first of all the establishment of clear objectives with respect to product characteristics and performances. Both characteristics and performances must comply to cement standard specifications and test methods as well as for concrete production.

The optimum proportion of mineral additions in blended cements is then to be found as a consequence of the available materials and the related cement standards.

In general terms, slags are usually characterised by a good hydraulic potential and a low water requirement in cement, derived from their glassy nature; these features allow for the production of slag cements with high contents of BFS (up to 95%!).

On the other hand, the amount of pozzolana is a compromise between the minimum amount necessary to combine the hydration lime and produce additional silicate hydrates and the maximum acceptable content due to the softness and high porosity of pozzolanas that yield particles with high water demand, thus affecting cement workability. In this case, a normal range for natural pozzolanas or activated clays in cement is 20-45%.

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As a rule of thumb, one shall consider that pure replacement of OPC with a mineral component will cause a loss in early strength of 1.5 times the amount of the addition. That is to say that 20% replacement will drop early strength by approximately 30%. This strength loss can be compensated to a great extent by finer grinding, either the cement as such in case of intergrinding, or the clinker fraction in case of separate grinding and following blending.

Increase of grinding fineness will anyway reflect in higher energy consumption of the mill and could impair some of the cement characteristics, especially water demand. In this respect, the technology of chemical admixtures is capable to provide a selection of suitable products for the various needs. Through the proper use of additions, one can improve mill performance, increase early strength, reduce water demand, improve workability.

6. PROPERTIES AND APPLICATION OF BLENDED CEMENTS

Composite cements permit both replacement of normal pure Portland cement and production of special cements whose manufacture usually entails high production costs. In concrete constructions, composite cements can completely replace pure PC with only few exceptions. In fact, there are countries where no pure PC is produced and all the needs of the construction industry are met by composite cements.

According to the experience of many countries, typically 25 to 40% of clinker may be replaced by fly ash or pozzolan, even much higher proportions of blast furnace slag and around 5% limestone filler or kiln dust, without significantly affecting the characteristics of the cement produced, if compared to an ordinary Portland cement. The production of composite cements provide a good solution for a partial or global elimination of the problem kiln-dust. Such a production requires, however, that additions of reasonable quality are used and that the blending/grinding process is optimally adjusted.

6.1 Reactivity of Composite Cements

The main disadvantage of composite cements is mostly their reduced reactivity, i.e. their slower setting and lower early strength compared to pure PC. For this reason, the fast-setting high grade Portland cement (corresponding to ASTM type III) cannot be replaced completely by composite cements, for example in prefabrication. Special care has to be taken when using composite cements for thin concrete elements, the slow reactivity of the composite cement requires special attention in curing to avoid premature drying out, which would prevent a sufficient degree of hydration, particularly near the surface.

6.2 Influence on Setting Time and Water Requirement

Fig. 11 shows the influence of slag, fly ash and natural Pozzolanas on initial set for laboratory cements, on paste of constant consistency. Slag (SL) give generally a longer initial setting time, whereas addition of pozzolan contribute to a small diminution of initial set. The case of fly ash is more problematic and most of this additive cause a significant and detrimental lengthening of setting time. However, this particularity can be beneficial for the production of road cements where a long initial set is desired.

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Fig. 11 Influence of Additions on Initial Set (Laboratory Cements)

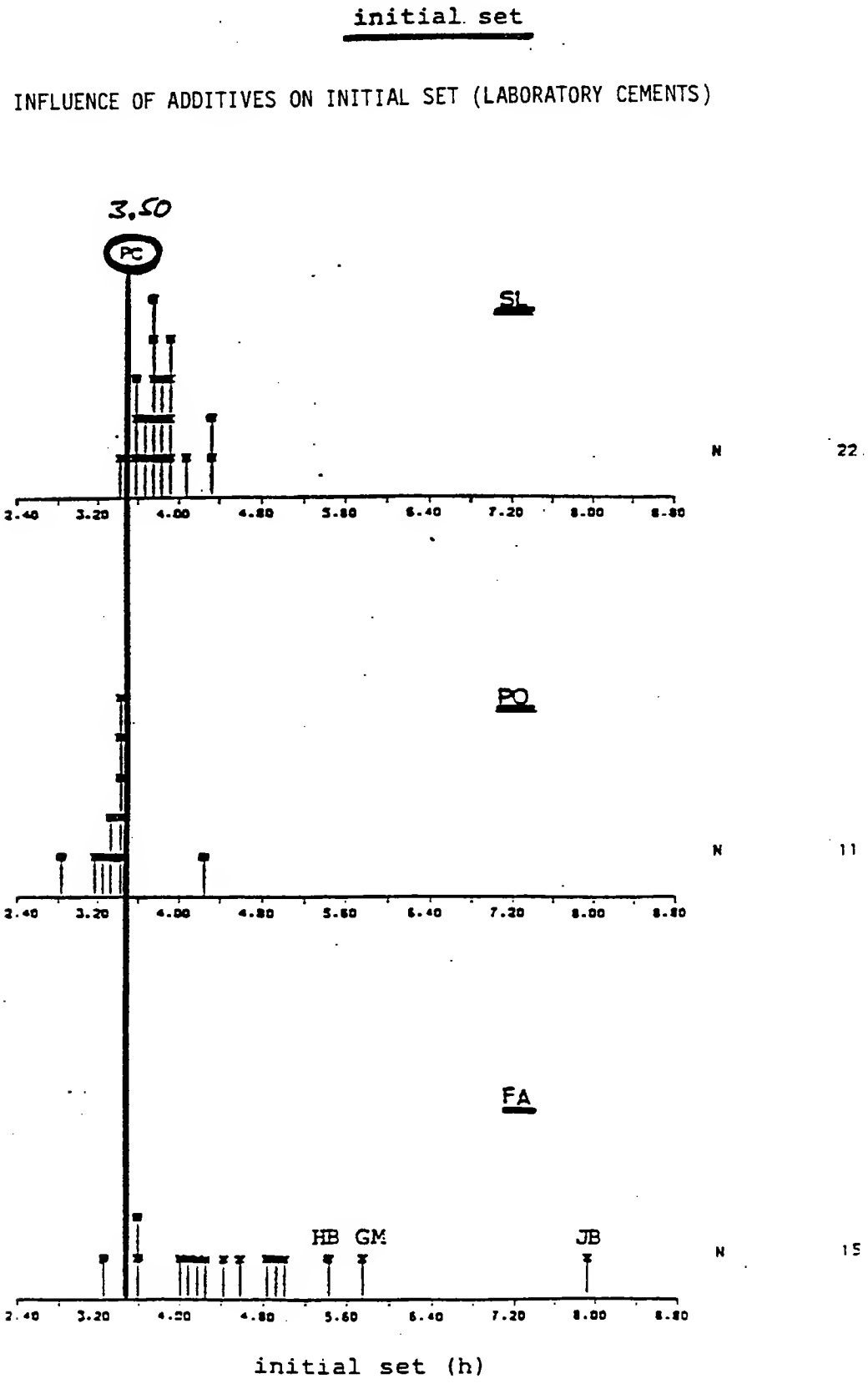

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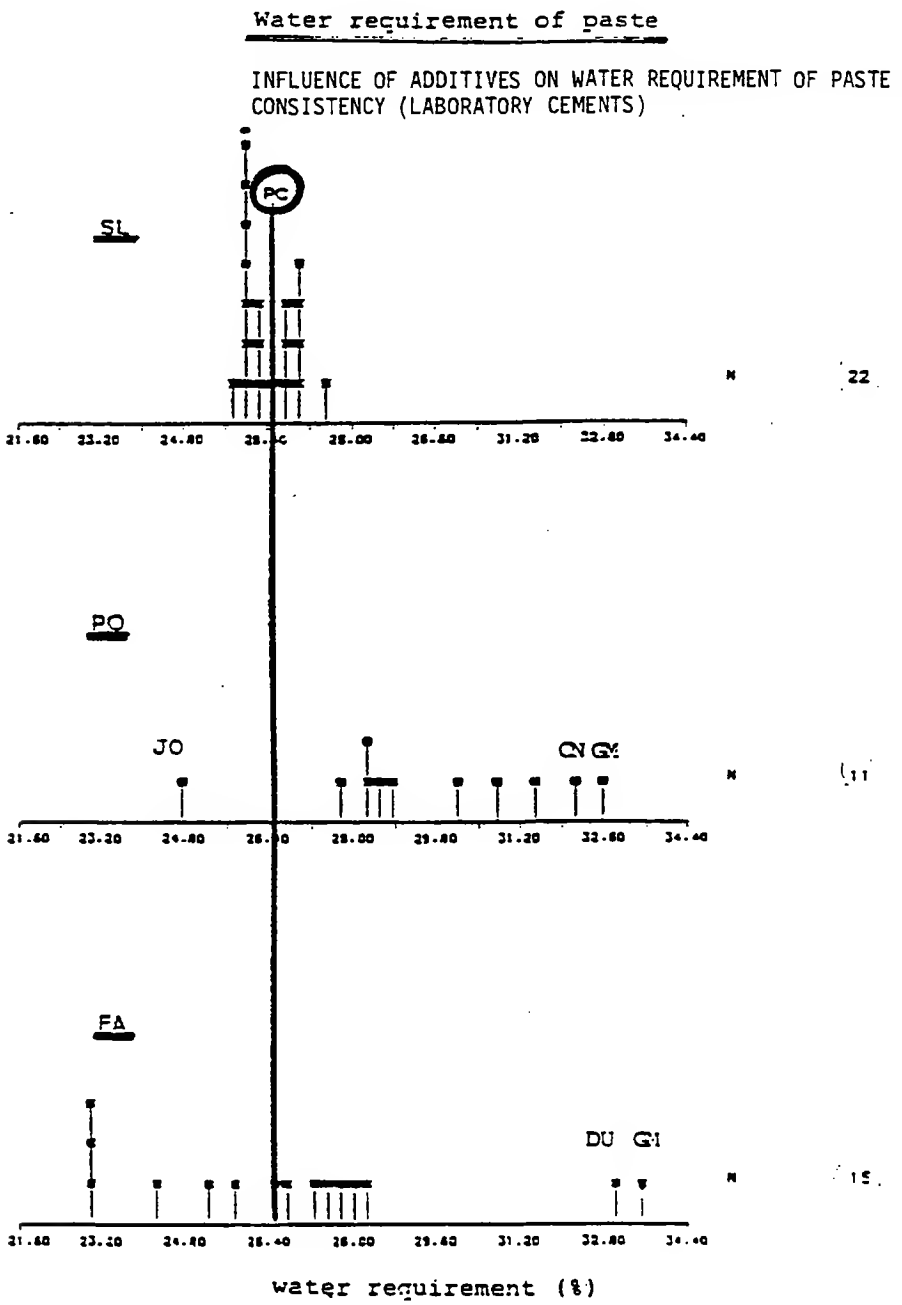
Fig. 12 shows the influence of slag, fly ash and natural Pozzolanas on water requirement for laboratory cements, on paste of constant consistency. If slag has no influence on this parameter, Pozzolanas give generally an important increase of water requirement depending of the nature and grindability of the pozzolanic material. Siliceous fly ash gives generally a decrease of water requirement, whereas calcareous (or lignitic) fly ash show an increase of this parameter which increases proportionally to the calcium content of the ash. The increase of the quantity of spherical particles conduct in most cases to a decrease of water requirement compared to pure PC.

Fig. 12 Influence of Additions on Water Requirement of Paste of Standard Consistency (Laboratory Cements)

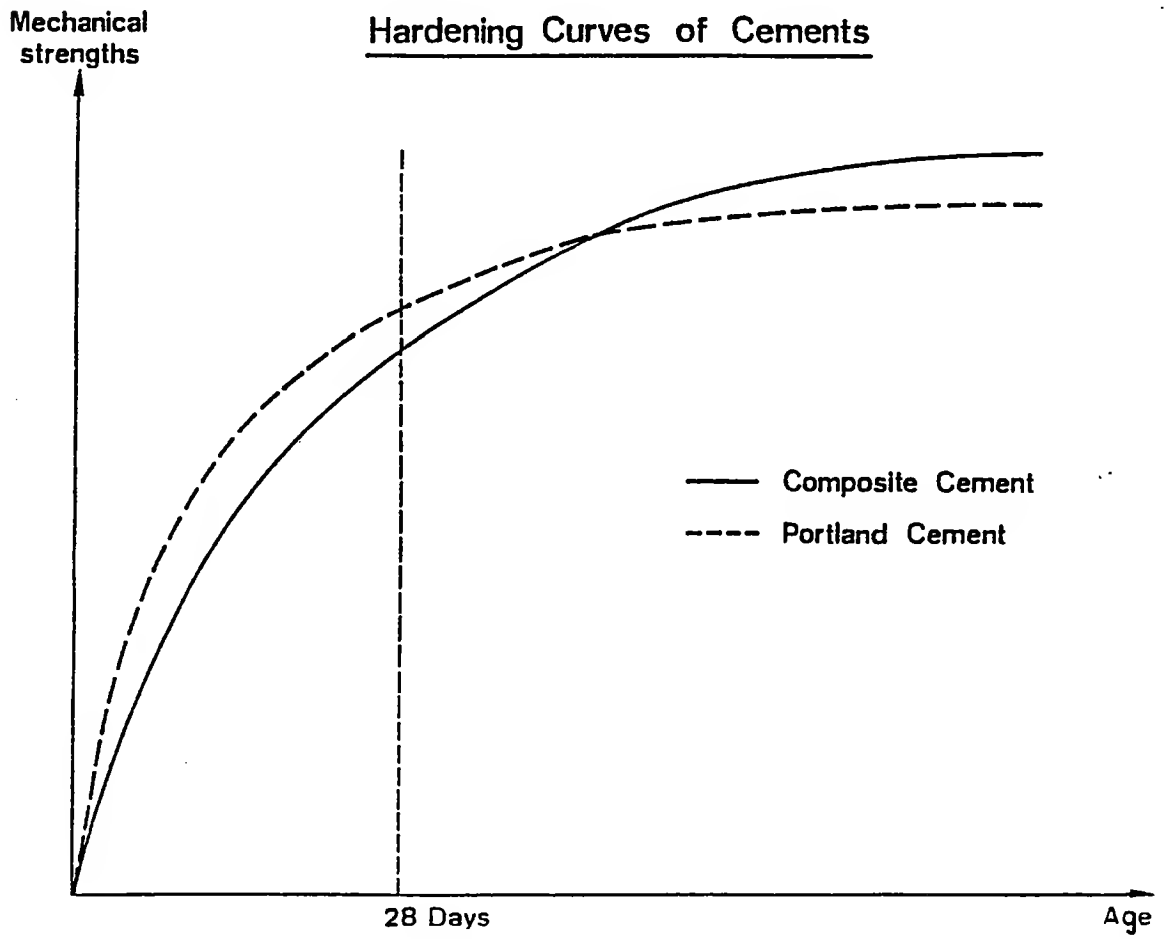


6.3 Other Important Properties

It is well accepted that composite cements are especially suited for mass concrete due to their lower heat development and hardening.

The disadvantage of initial slower strength development, later generally turns out to be an advantage - when correctly cured - for the durability of concrete construction because composite cements usually have a higher strength reserve than pure PC (see Fig. 13), due to the formation of a denser cement matrix which at least offsets the risk of enhanced carbonation due to the lower pH of the matrix.

Fig. 13 Hardening Curves of Cements



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Last but not least, composite cements are also recommended for specific applications such as high sulfate resistance, sea water resistance, prevention of alkali-aggregate reaction. In brief, blended cements can be used instead of special, or pure Portland cements where the following concrete characteristics are required:

- Improved workability (especially fly ash cements)
- Lower heat development of bulk concrete
- High to very high final strength
- Increased water tightness
- Improved resistance to sulfate and other chemical attacks
- Lower sensitivity for alkali-aggregate reaction
- Reduced tendency to unsoundness due to free lime and MgO

Moreover, since slag, fly ash and pozzolan combine with the calcium hydroxide which is formed during the hydration of clinker, the tendency to form efflorescence is diminished.

7. LIMESTONE AND LIMESTONE FILLERIZED CEMENT

Difficulties in the supply of blast furnace slag and fly ash have led cement manufacturers to study the possibility of replacing them by other additions permitted under current standard specifications. Since 1979, French standards have included fillers among those additions that may be used in CPJ cement in a proportion up to 35%.

The recently issued European standard ENV 197/1 allows for the production of two main types of limestone cement, with limestone content of resp. 6 to 20% and 21 to 35%.

The high availability and the low cost of limestone for a cement plant made this type of cement very attractive for most producers, and it is considered opportune to deal with it in a dedicated chapter.

7.1 Properties and advantages of limestone

Limestone filler is an inert material. It produces no chemical effect on the cement in presence of water. If influences the physical properties of cement, and mainly certain features such as:

- Improved workability
- Reduced permeability and capillarity
- Higher resistance to cracking
- Better finishing, etc.

Prior to their introduction on the market, cements containing limestone filler have been subjected to laboratory and industrial trials in order to determine the main characteristics:

- Mechanical strength
- Setting behavior
- Shrinkage
- Resistance to freeze/thaw
- Effect on rheology and concrete properties, etc.

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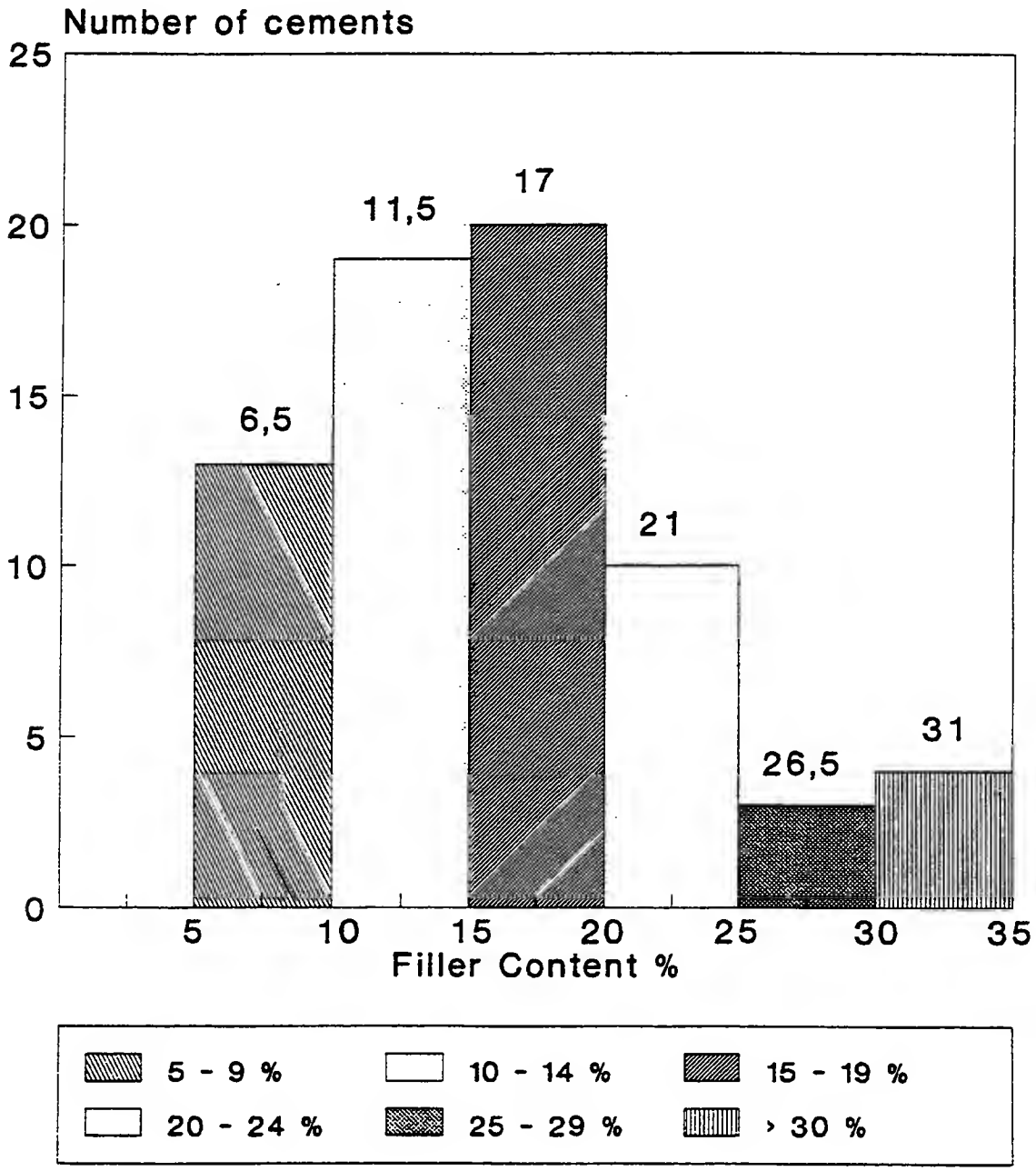
Fig. 14 illustrates the distribution of the standardized limestone containing cements in France in October 1992, according to their individual filler content. The data are collected from 68 cements containing between 5 and 32% filler, the mean filler content being 15%.

Fig. 14 Distribution of 68 Limestone Filler Cements: Individual Filler Content



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The main reasons for using calcareous additions in the production of blended cements are the following:

- Cost reduction
- Scarce availability of mineral components
- Improvement of cement workability
- Substitution/complementation of gypsum
- Differentiation of product
- Environmental issues

7.2 Standard Specifications - Classification

According to the European standard ENV 197/1, limestone used as a main constituent in cement shall meet the following requirements:

- limestone $CaCO_3 \geq 75\%$ mass
- clay content methylene blue absorption ≤ 1.2 g/100 g
- organic material content TOC $\leq 0.20\%$ by mass

Clay content has to be limited in order to keep the water requirement of the filler low and avoid a degradation of strength.

The low value of organic material content will eliminate limestone exhibiting cracking risks and micro cracks detrimental to freeze-thaw stability.

Limestone fillerized cements can be subdivided in three main types:

- Cements (pure Portland or composite) with minor proportions of limestone (< 5%)
- Portland-limestone cements (5 to 35%, according to EN standards)
- Masonry cements (up to 75% limestone).

7.3 Cement Properties

7.3.1 Grain Size Distribution

The difference in grain size distribution of the particles when comparing clinker and limestone filler by the intergrinding process is given in Fig.15 for a cement containing 25% limestone. The Blaine fineness of a limestone fillerized cement is greatly affected by the higher quantity of limestone smaller to 10 microns compared to the quantity of clinker. However, the increase of specific surface does not affect (or only very rarely) the required grinding energy.

To explain this, it is worthwile mentioning that a Blaine measurement is based on a physical property, that is air permeability of a compacted layer of cement at a given porosity. When the composition of cement is changed, by replacing clinker with softer materials, the same particle size distribution of different cements will not necessarily yield the same Blaine fineness, due to the different air flowability characteristics of the cements.

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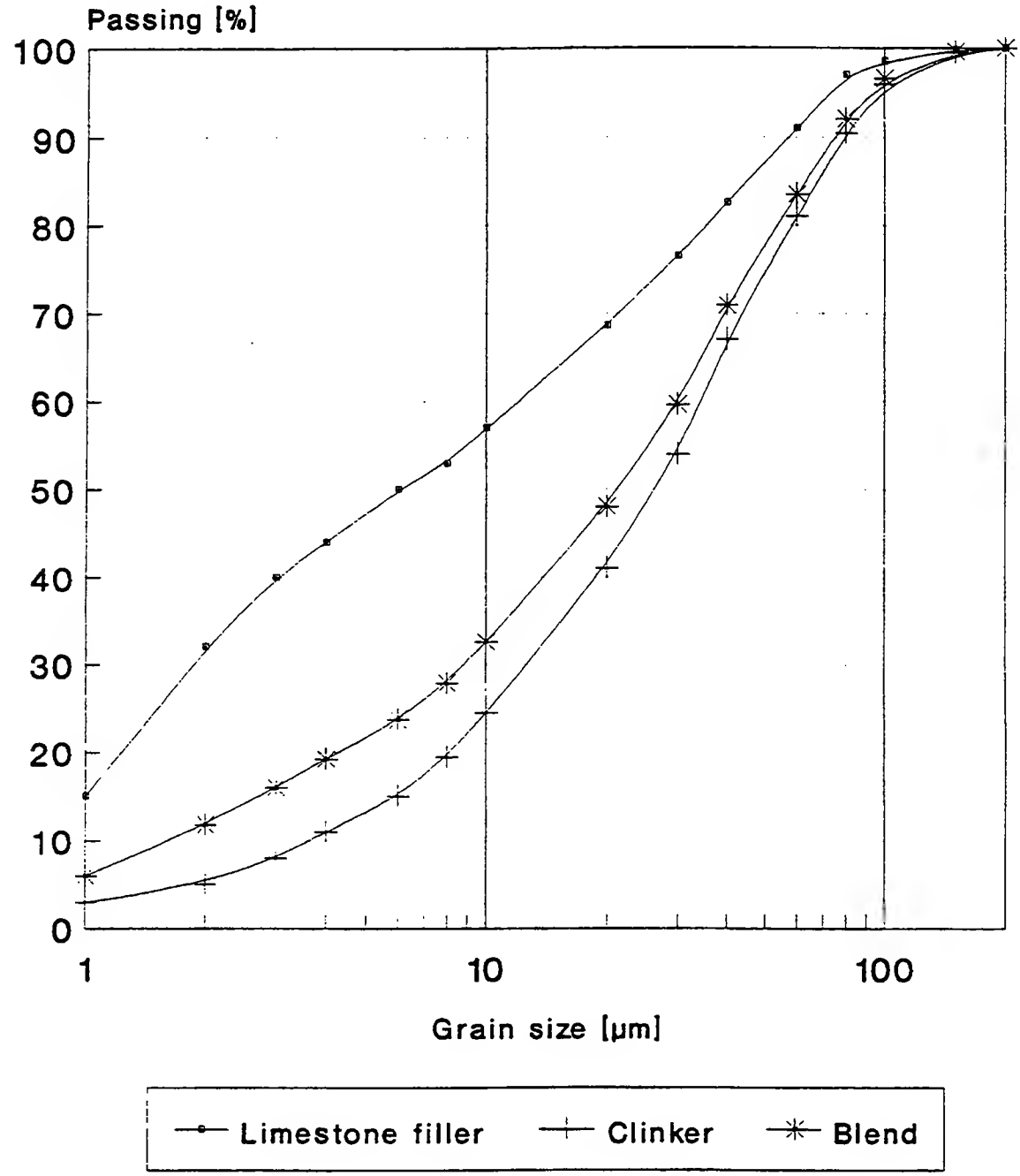
Fig. 15 Grain Size Distribution in Limestone Fillerized Cement



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7.3.2 Influence of Limestone on Strength Development

Although limestone filler is an inert material, it was always shown that it has a favorable effect on strength development due to:

- the better workability
- the accelerated crystallization of hydrated minerals, more especially aluminates
- the aggregate bonding properties of the paste.

7.3.3 Consistency and Setting Time

The water requirement for paste and mortar of constant consistency is generally not affected by the incorporation of limestone filler in any major way. Moreover, it was shown that limestone filler can partially replace gypsum without shortening the setting time of the blended cement.

7.3.4 Other Properties

- The influence of temperature on initial properties remains within the same order of magnitude as that observed with other additions.
- Shrinkage is similar, or somewhat lower compared to pure Portland cement.
- The heat of hydration of limestone fillerized cement decreases proportionally with the filler content, the heat of hydration being only affected by the clinker fraction in the blend.

The following Table 16 summarizes the main characteristics of two Portland limestone cements industrially produced in France and in Germany.

Table 16 Quality Characteristics of Industrially Produced Portland Limestone Cements

Origin		France	Germany
Limestone content	[%]	27.5	19.0
Fineness	[cm^2/g]	5150	4960
water requirement	[%]	27.7	25.7
Init. setting	[min.]	95	95
ISO mortar	[N/mm^2]		
	1 d	16.5	13.0
	2 d	26.1	22.9
	7 d	38.5	41.0
	28 d	48.0	49.7

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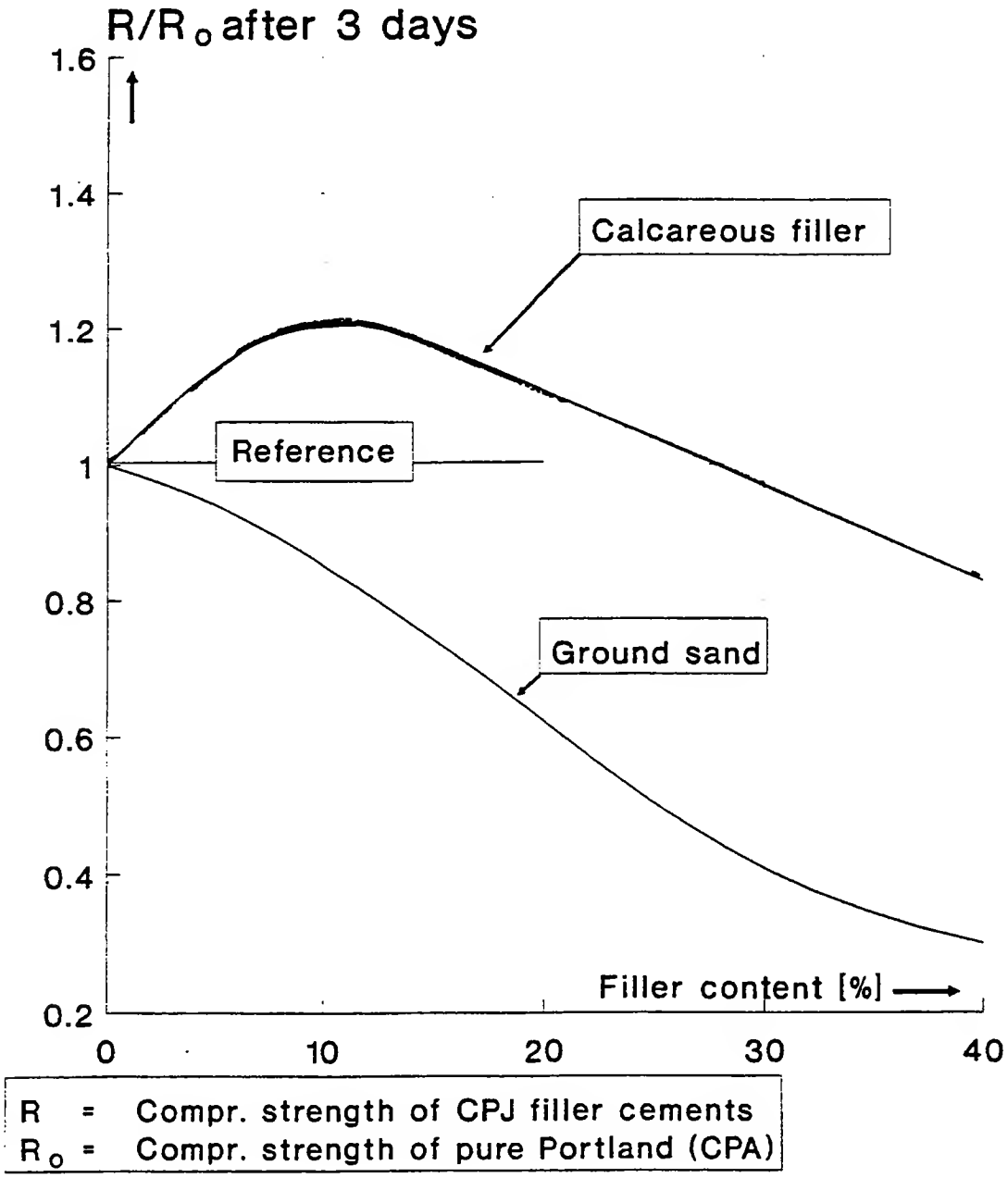
7.4 Concrete Properties

- Water requirement of standard concrete is influenced by both the nature of the limestone and the filler content in a non-linear way. However, the workability is often improved by addition of limestone filler as is the response to addition of superplasticizers/water reducers in concrete.
- Early strength development is usually not affected by the limestone content and the somewhat lower w/c ratio compensates for lower late strength.
- As long as the limestone meets the requirements of the standards, the susceptibility of mortar and concrete to frost and frost/thaw is only directly connected to the quality of concrete (the use of air-entraining agent is required).

Fig. 16 Influence of Calcareous Filler in Limestone Cement on Early Strength



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7.5 Masonry Cements

Masonry cements can be considered as a special type of blended limestone cements; they actually contain 40 to 75% limestone and suitable chemical admixtures such as air-entraining agent and/or plasticizer.

Characteristics of masonry cements are reported in the chapter on Special Cements, to summarize it here, the choice of limestone (and other minor constituents like kiln dust, fly ash, slag) must be focussed on the obtainment of the following properties

- good adhesion, bond strength
- good cohesion and plasticity
- high water retention
- no segregation
- air-entrainement (15 to 20% vol) without foam formation
- improved impermeability.

7.6 Conclusion

The experience with manufacturing and utilization of limestone blended cements has shown that they exhibit favorable characteristics compared to pure Portland cements or other types of blended cements.

Limestone fillerized cements enable the cement industry to continue offering a standard high quality product at a favorable price, despite potential shortage of conventional additions such as slag and pozzolan.

8. TRENDS OF PRODUCTION AND MARKETING COMPOSITE CEMENTS

Rather than talking about cements, it will be necessary in the next years to consider binding agents and concrete mixes, in which Portland cement clinker will still be the principal active ingredient, but whose quantity should have to be drastically reduced for evident problems of environment awareness and reduction of emissions (CO, SO, NO, etc.).

Already today, concretes are being produced in which the clinker represents only 40% (and in some cases less than 20%) of the binding agent, the rest of the constituents such as fly ash, silica fume, ground slag, limestone filler, etc. being added directly at the concrete batching plant.

If we want to maintain the same quality of concrete and assure the future of our industry, we must increase drastically the proportion of additions in cement and master the production of binders with a maximum content of clinker substitutes. The possibilities for substitution are numerous and may be dived into three principal categories:

- Hydraulic materials for which the level of the average substitution of clinker will be in the order of 50% and may achieve 90% in the case of slag.
- Natural pozzolanic materials which will be able to substitute up to 30 to 40% of the clinker, but which require a definite proportion of CaO as activator.
- Artificial hydraulic and/or pozzolanic materials for which the level of substitution will be of the same order as of the two former categories, depending on their hydraulic and/or pozzolanic properties.

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The real degree of replacement for these types of substitute materials will vary as a function of their individual reactivity. They can be mixed with other types of possible additions (triple blends or multi-blends or 'cocktail' cements), the clinker (or the activator) proportion depending on the quantity of CaO needed for the activation of the additions.

The reduction of the quantity of slag available for cement production on the market as well as the necessity of reducing the emissions, will require the production of thermally activated substitutes for clinker.

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Chapter 4

Optimised Cement Design

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1. INTRODUCTION

The process of cement design consists of the following interrelated steps:

1) selection of the most convenient set of components
2) determination of the relative proportions of the components
3) definition of the fineness and grain size distribution of cement (compound grinding) or the cement components (separate grinding)

The objective of the cement design is to achieve the specified or desired performance of the cement at the minimum possible cost.

The procedures applied in practice for cement design are usually based on analytical tools (models to predict cement performance), experiments (trials on laboratory and industrial scale) and on experience. The better the knowledge of the relationships between cement components, proportioning and processing and the cement properties, the easier it is to arrive at the optimum solution.

The optimisation of the cement design requires nowadays more attention than in earlier times. The main reasons for this development are:

- increase in number of cement components (use of mineral components and chemical admixtures in the cement)
- use of new grinding technologies (roller press, vertical mill, Horomill) having an effect on the resulting grain size distribution and grinding temperature
- extension of product and application range

The object of the present paper is to describe the basic considerations influencing the cement design and to give an overview on the influence of the cement components, their proportioning and cement grinding on the properties of cement.

2. BASIC CONSIDERATIONS FOR CEMENT DESIGN

2.1 Product requirements

The cement design will strongly depend on the performance requirements to be fulfilled by the cement. These requirements, which are determined by the respective standards and by the market, may comprise specifications on.

- proportioning of cement components and chemical composition
- workability (water demand, setting), volume stability and strength
- special properties:
  - heat of hydration
  - sulphate resistance
  - alkali-aggregate reactivity
  - shrinkage, etc.

Besides the above specifications, there may be further requirements with regard to the handling of the cement (i.e. temperature, flowability and storage stability).

In the future, also certain requirements with respect to energy consumption and emissions (in particular CO2) during the cement production may be imposed.

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2.2 Available cement components

The flexibility in cement design will be obviously controlled to a large extent by the available cement components. The most important aspects of the cement components (clinker, mineral components and gypsum) in this respect are:

- available quantities
- quality / uniformity and
- costs

The cement plants usually count with one "normal" type of clinker, whose characteristics are pre-determined by the raw material situation and by the burning and cooling conditions in the kiln. Occasionally, also special clinkers are produced for certain cement types, but with the increased use of mineral components in the cement, which allow to obtain special properties with "normal" clinker, less and less of such clinkers will be applied in the future.

The availability of the mineral components varies from country to country. The main industrial by-products used for cement production - blast furnace slag and fly ashes - are principally available world-wide in great quantities (see Table 1); however, only part of it complies with the necessary quality requirements for an application in the cement. Among the natural mineral components, limestone of suitable quality should be available at all cement plants, whereas the natural pozzolans are less wide-spread.

Table 1: Estimated production of fly ashes and blast furnace slags (Mio t/a)

	Blast furnace slag (1994)	**Fly ash (1992)**
Western Europe	36	61
Eastern Europe (+ former USSR)	28	95
North America	20	51
Latin America	11	3
Africa	3	24
Asia	78	125
Australia	3	7
World	178	366

Natural gypsum deposits are scarce in some countries. In such cases, alternative materials like natural anhydrite and limestone or by-product gypsum from other industries have to be considered.

The different type of chemical admixtures, which can be added at the cement grinding stage, can in principle be purchased anywhere in the world.

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2.3 Production facilities

The available production facilities (in particular the cement grinding installations) put certain constraints on the cement design. Such constraints may lead to limitations with respect to:

- clinker factor
- fineness range of cements
- feasible number of products
- type of cement grinding (compound or separate)

Moreover, the cement design will be influenced decisively by the type of cement mill used for grinding. For instance, with the new grinding technologies, adjustments have to be made to account for the differences in grain size distribution and grinding temperature compared to the traditional systems.

2.4 Economy

The production costs of cement basically consist of the costs of the materials entering the cement mill and the grinding costs. From the two factors, the material costs generate by far the greatest part of the production costs.

The most expensive material in the cement is usually the clinker. The minimisation of the clinker content in the cement is therefore the single most important factor in reducing the production cost, provided that mineral components of suitable quality are available at convenient prices.

The principal ways to reduce the clinker factor in the cement are:

- adjustment of the fineness and grain size distribution of the cement and its components
- use of high quality clinker
- use of chemical admixtures already in the cement

The possible clinker reduction is of course limited by the factors discussed in the previous chapters.

3. INFLUENCE OF CEMENT COMPONENTS ON CEMENT PROPERTIES

3.1 General

Due to the great variety of factors involved, it is difficult to describe precisely the relationship between the cement components and the cement properties. The available models for the prediction of cement performance usually only reflect the general trends.

The effects on the cement properties are best understood for the clinker and gypsum. Least knowledge is available in case of the mineral components and the chemical admixtures, so that virtually the only way to assess their influence is to carry out performance tests.

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3.2 Clinker

The composition of clinker gives some indications on the properties of cement to be expected, as it influences the rate of hydration reaction and thus the setting and hardening rate of cement The composition of clinkers control the quantity and rate of heat evolved during hydration and the resistance of cement to sulphate attack; therefore, limiting values are specified.

In this section, the influence of composition of clinker on the following properties of cement shall be discussed:

- water requirement of standard paste and consistency of concrete
- stiffening rate and setting time of standard paste and slump loss of concrete
- heat of hydration
- strength of mortar and concrete
- sulphate resistance
- other properties of concrete

A summary on the relationship between clinker composition and the principal cement properties workability (water demand, setting) and strength is given in Table 2.

Table 2: Effect of clinker composition on water requirement and setting time of standard paste and compressive strength of ISO mortar (general trends)

Clinker	Water req.	Setting time	Strength	
			early	final
C3S	--	--	↗	↗
C2S	--	--	↘	↗
C3A	↗	↘	↗	↘
C4AF	--	--	↘	↗
K2O	↗	--	↗	↘
Na2O	↗	--	↗	↘
SO3	--	↗	↗	↘
P2O5	--	↗	↘	--

↗ increasing

↘ decreasing

-- no effect

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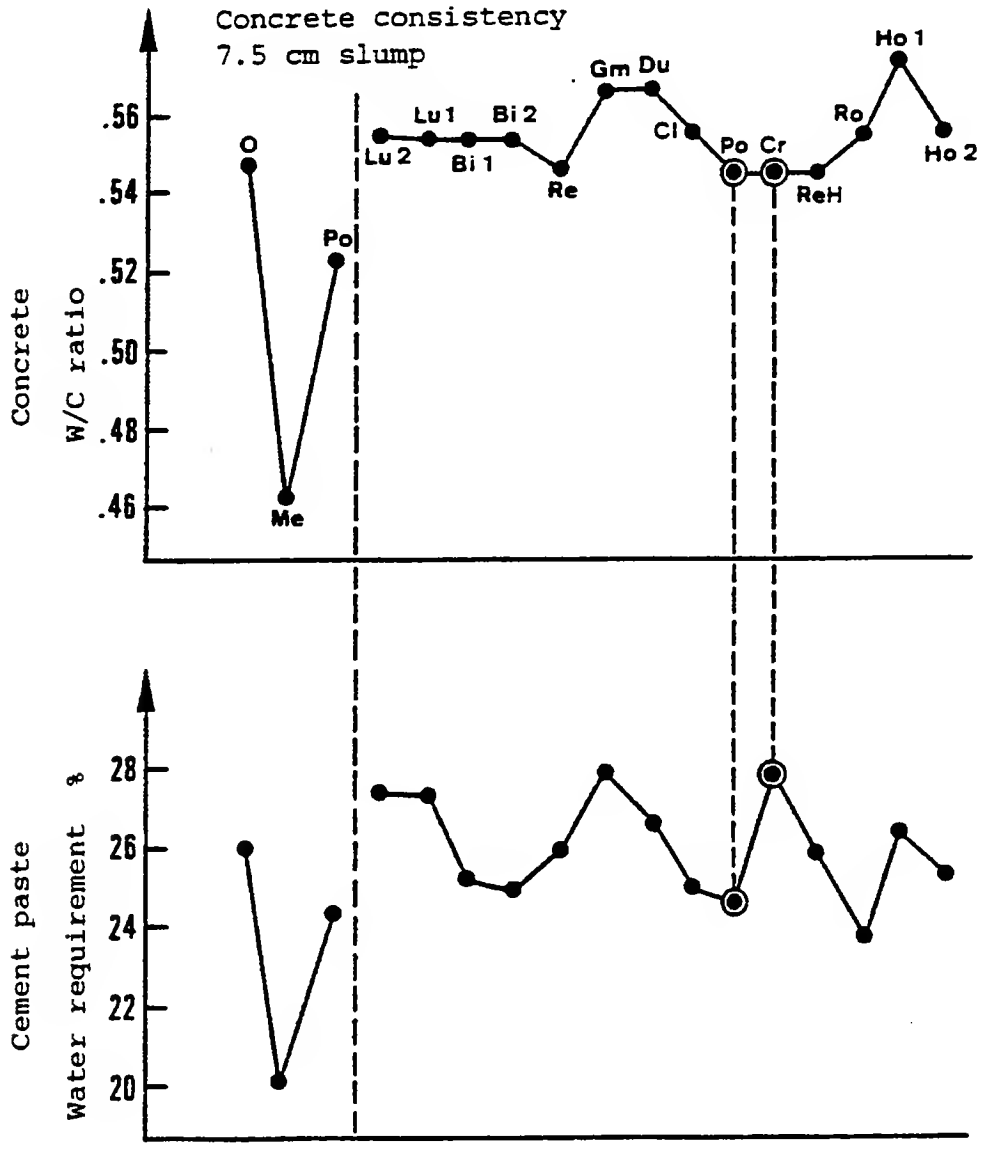
3.2.1 Water requirement of standard paste and consistency of concrete

The water requirement of the standard paste of normal consistency depends primarily on the aluminate and alkali content of clinker and on the fineness of cement. From a multiple regression analysis carried out at HMC on 48 different ordinary Portland cements, the following relation between the water requirement and cement composition was derived:

W.r. %	=	17.4 + 0.15 a + 0.26 b + 0.12 c
a	=	particle size fraction 10 to 30 μμ in wt %
b	=	C3A content in wt % (Bogue's formula)
c	=	total alkali content in wt %

The relation between the water requirement of standard paste and the composition of cement cannot be applied to concrete, as there is a rather weak relationship between the water requirement of paste and water/cement ratio of concrete (see Figure 1)

Figure 1: Water requirement of cement and w/c-ratio of concrete



Po	-	Water reducing admixture (Pozzolith)
Me	-	Superplasticizer (Melment)
Lu 2 ... Ho 2	-	various OPC
Gm, Du, etc.	-	various Group plants

The effect of cement on the consistency or water requirement of concrete is rather small compared to other factors, such as sand, admixtures and temperature. An exception is concrete with a very short mixing time, where cement with false set may seriously impair the consistency of concrete.

3.2.2 Rate of stiffening and setting time of standard paste and slump loss of concrete

The stiffening rate or the „Vicat" setting time of the standard paste is significantly influenced by the composition of clinker. The sulphates and phosphates of clinker usually delay, whereas aluminate shorten the setting time of cement.

The relation between the stiffening rate or setting time of standard paste and the stiffening rate - expressed as slump loss - of concrete is, just as for the water requirement, rather poor. Therefore, it is difficult to estimate the stiffening rate of concrete on the basis of composition or fineness of cement.

3.2.3 Heat of hydration

The effect of the clinker composition on heat of hydration has already been discussed in detail in the paper on cement hydration. The principal way to control the heat evolution of the clinker is the adjustment of the C3S and C3A content.

3.2.4 Strength of mortar and concrete

The rate of strength development of mortar or concrete depends on the type (or composition) of cement. The general tendency of cements with a slow rate of hardening is to have a slightly higher ultimate strength.

The ASTM type IV cement, with low content of C3S, has the lowest early strength, but develops the highest ultimate strength (see Figure 2). This agrees with the influence of individual clinker components on the rate of strength development measured on pure clinker minerals (see Figure 3).

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Figure 2: Strength development of concrete made with different cement types

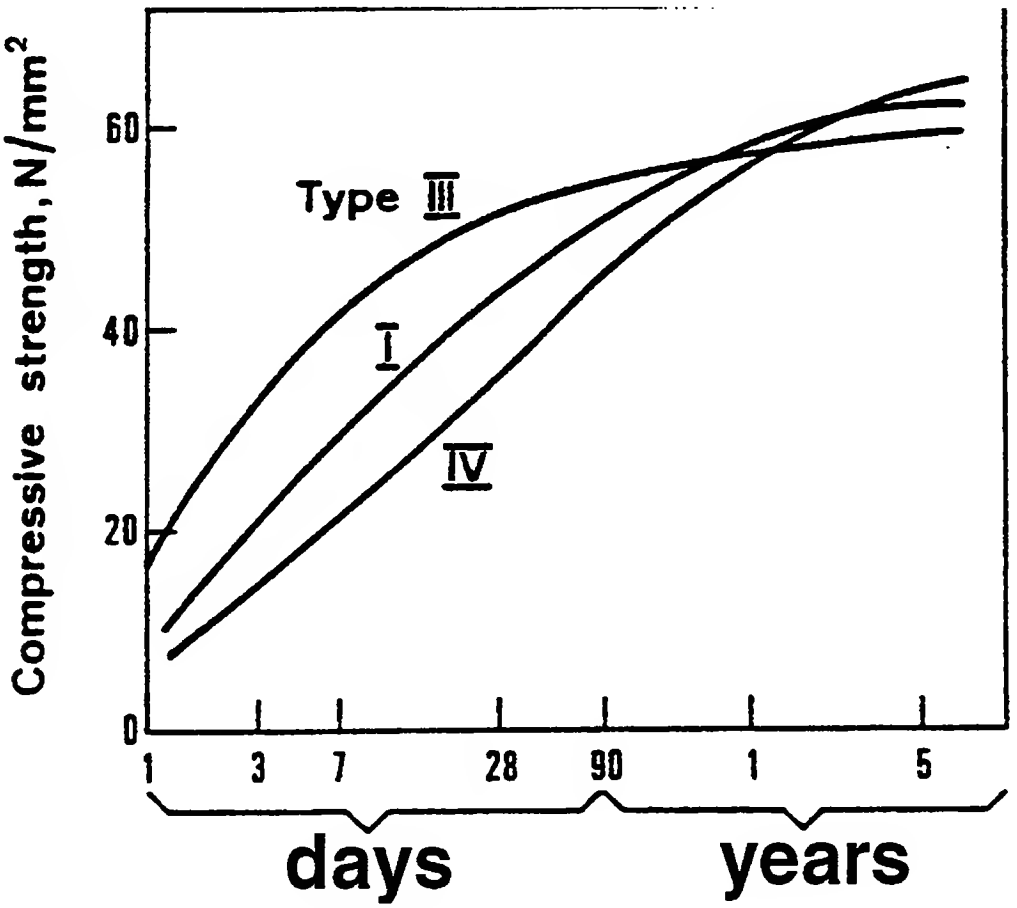
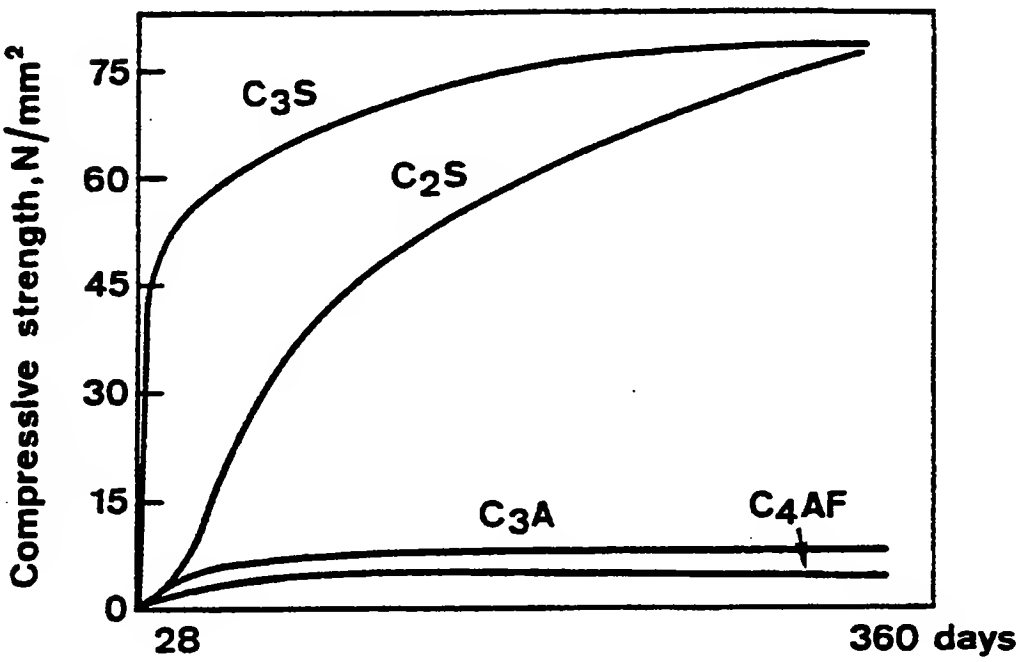


Figure 3: Compressive strength of cement compounds



The two calcium silicates develop the highest strength, but at different rates. The aluminate develops little strength, despite a high rate of hydration.

The rate of strength development of mortar or concrete depends on the clinker composition as follows:

a) **Calcium silicates.** The different rates of hydration of C3S and C2S affect the rate of hardening in a significant manner: A convenient rough rule assumes that C3S contributes the most to the strength development during the first four weeks and C2S afterwards. In general, somewhat higher ultimate strengths are reached by cements with lower calcium content, i.e. rich in C2S. This observation corresponds with the assumption that the strength of cement depends on the specific surface of its hydration products. C2S produces more colloidal CSH gel and less of the crystalline $Ca(OH)_2$ than the C3S.

b) **Aluminates and ferrites.** The influence of the other two major components on the strength development is still controversial. Presumably, the C3A contributes to the strength of the cement paste during a period of one to three days. In general, both aluminate and ferrite contribute to the strength of cement to a minor extent, but significantly influence the hydration process of the silicates and thus have an indirect effect on the rate of hardening.

c) Of the **minor components,** the alkali sulphates exert the greatest influence on the rate of hardening. The alkali sulphate - mostly present as easily soluble potassium sulphate or calcium-potassium sulphate with a molar ratio of 2:1 to 1:2 - accelerates the rate of hardening, improving the early strength and decreasing the 28 day and ultimate strength (see Figure 4). Of the other minor components, fluorine accelerates, whereas the phosphorous compound delays the rate of hardening.

d) **Clinker characteristics other than chemical composition.** Particularly the burning and cooling conditions influence the rate of hardening of a particular clinker composition. Frequently, clinkers of the same chemical composition have different strengths and clinkers of different chemical composition have the same strength. A simple experiment proves that the very same clinker composition may have rates of hardening which vary considerably. Reburning of a clinker in a laboratory furnace changes the rate of hardening, but does not affect the chemical composition of clinker (see Figure 5).

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Figure 4: Effect of soluble K2O on the compressive strength of ISO mortar

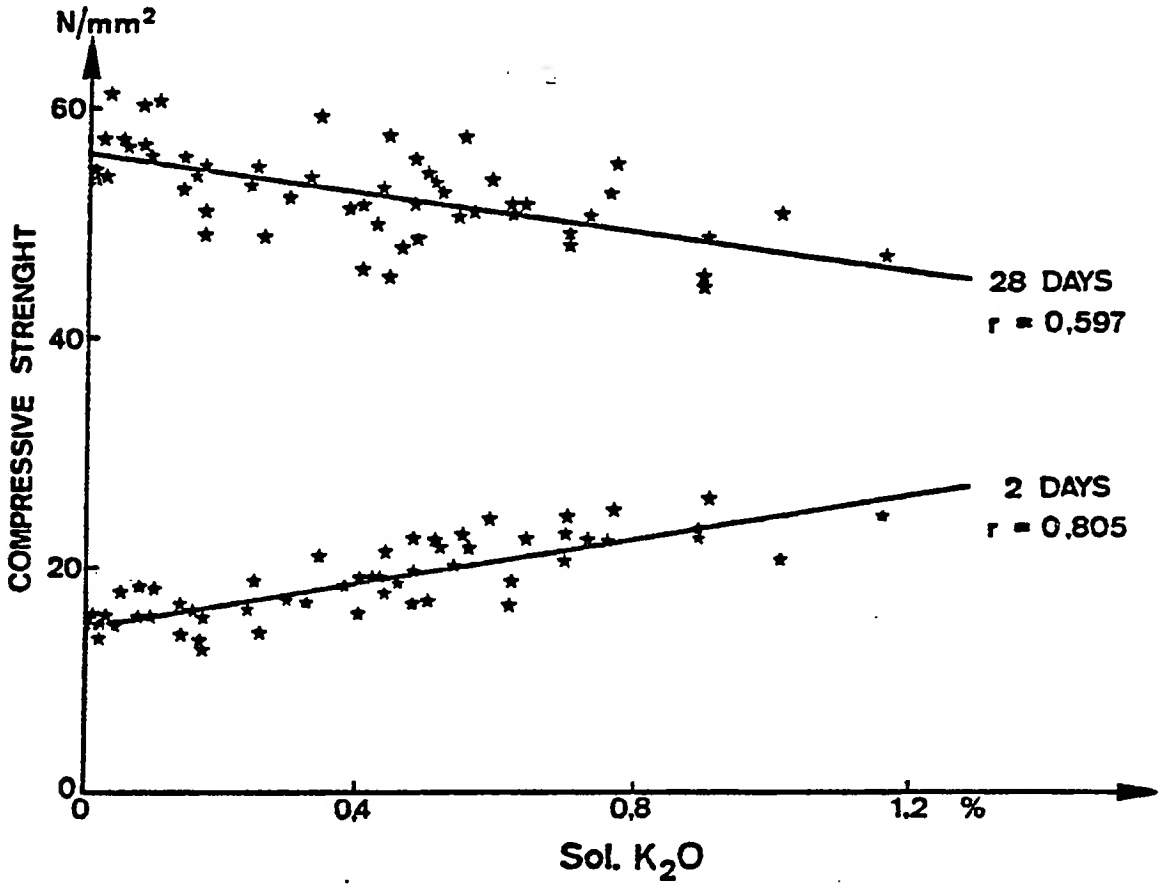

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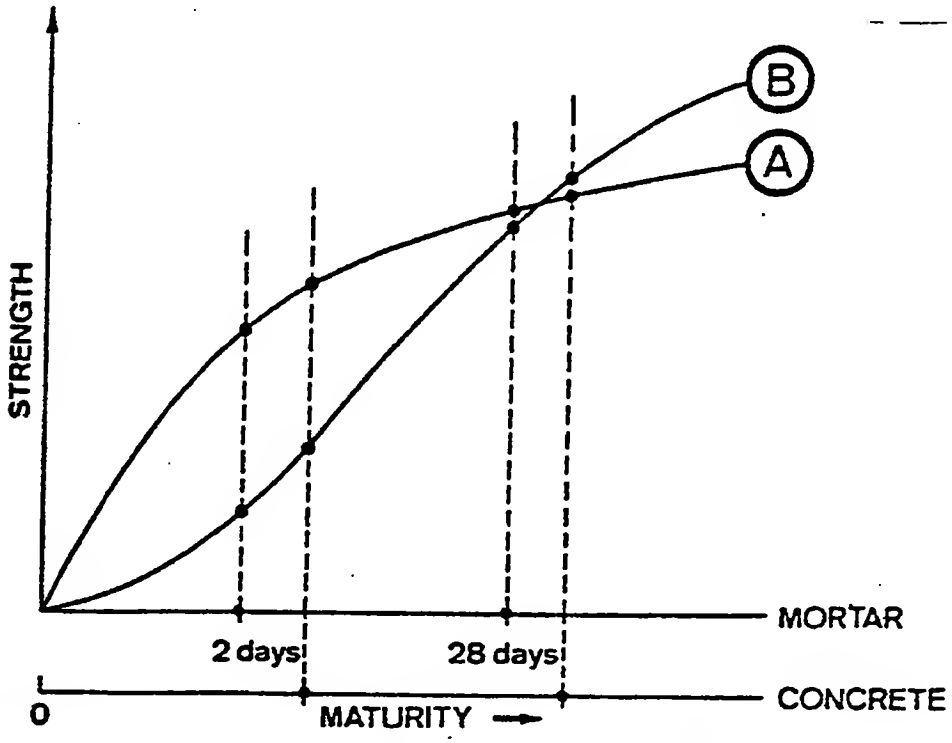
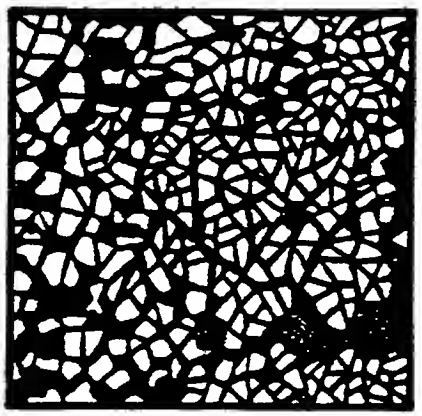
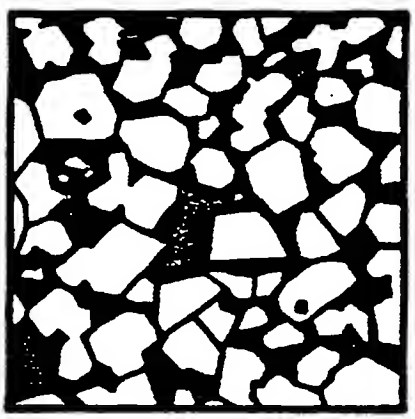
Figure 5: Model of strength development of mortar and concrete made with two clinkers of same chemical composition and different activity







A

B

A	=	Clinker of high activity
B	=	Clinker of low activity
Maturity	≅	Degree of hydration

A general guide on the necessary amount of the main clinker phases to achieve optimum strength development is given in Table 3. The most essential point is to have a high C3S content (in the order of 60%) and to adjust the C3A content.

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Table 3: "Ideal" composition of the clinker for optimum strength development

Clinker phase	"Ideal" content (%)
C3S	55 - 65
C2S	15 - 25
C3A	7 - 10
C4AF	7 - 10

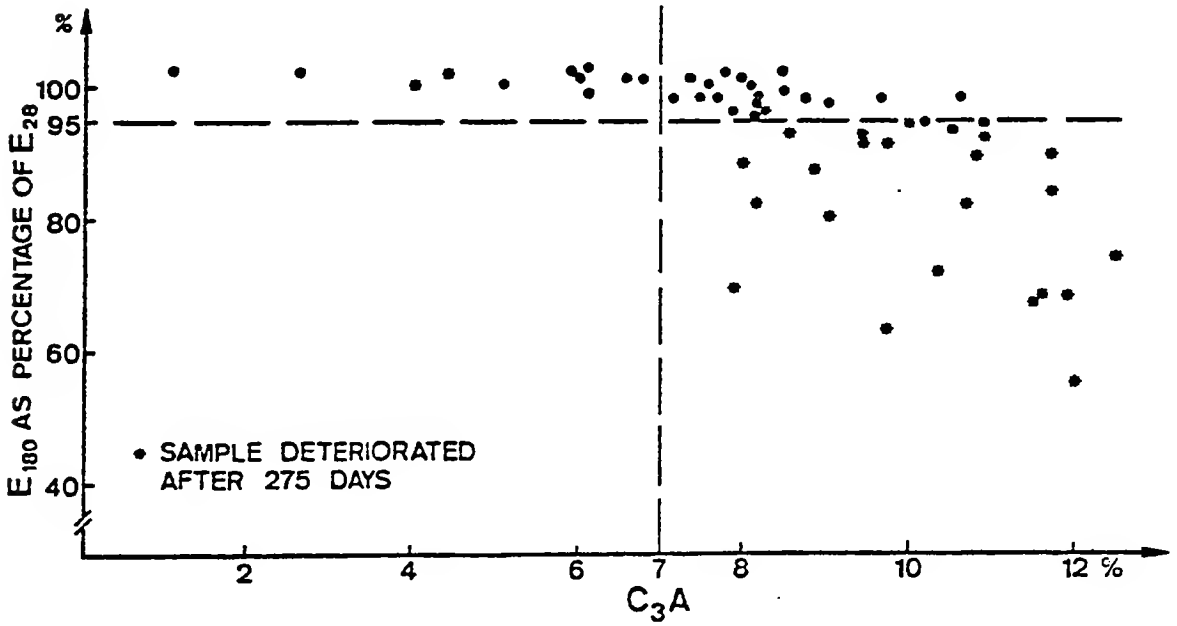
The influence of the clinker composition on the standard mortar strength is noticeably reduced in concrete. Depending on the quality of cement, sand and aggregate, the proportioning of concrete or mortar components, curing temperature, specimen dimension, the rate of hardening in mortar and in various concrete compositions is quite different.

Moreover, the use of admixtures in concrete - which is common practice today - makes the relation even more complicated. Due to different hardening rates of mortar and concrete, the relation between concrete and mortar strength at various ages varies and depends on the above mentioned factors. Concluding, the cement properties, as demonstrated through standardised testing methods, do not show their effect in the same way in concrete.

3.2.5 Sulphate resistance

The sulphate resistance of concrete depends primarily on the C3A content of clinker. The ferrite phases (C4AF) affect the sulphate resistance to a much lesser degree. The higher the C3A content of clinker, the more susceptible the concrete is to sulphate corrosion (see Figure 6).

Figure 6: Sulphate resistance of cement measured on ISO mortar specimen (55 OPC)
Influence of C3A content on the loss of Young's modulus of elasticity E (determined from ultrasonic pulse velocity measurements)



E28 after 28 days of regular curing = 100%
E180 after 28 days of regular curing and 180 days of exposure to 10% sodium sulphate solution

The rate of sulphate corrosion depends - apart from the C3A content of clinker - on factors other than cement:

- composition of concrete, particularly the water/cement ratio
- age of concrete at the time of the first exposure to sulphates
- type and concentration of sulphate solution
- duration and mode of sulphate exposure

3.2.6 Other properties

The other properties of concrete, such as

- freeze - thaw - resistance
- permeability
- cracking
- shrinkage and creep

are only slightly influenced by the composition of clinker and quality of cement. Other influencing factors, such as air content, w/c-ratio, curing conditions, are decisive.

The cement exerts only an indirect influence on these properties by its effect on the water requirement and rate of hardening.

3.3 Mineral components

(see also paper on blended cements)

The effect of the mineral components on cement performance can be related mainly to their activity. The three main classes of materials in this respect are latent hydraulic (e.g. blast furnace slag), pozzolanic (e.g. fly ash and natural pozzolans) and inert (e.g. limestone).

In case of the active mineral components (latent hydraulic and pozzolanic), the general effects with respect to cement properties are as follows:

- lower water requirement (except for natural pozzolans)
- delay in setting times
- lower heat of hydration
- lower early strength
- higher long term strength
- lower permeability
- improved resistance to sulphate and other chemical attacks
- lower sensitivity for alkali-aggregate reaction

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The actual influence on the cement properties will of course still depend on the individual nature of each material. A more detailed comparison on the effects of the main active mineral components blast furnace slag, fly ash and natural pozzolan (at same dosage) is made in Table 4.

Table 4: Effect of main active mineral components on cement properties (general trends)

	Blast furnace slag	Fly ash (class F)	Natural pozzolan
Water requirement	0	↘	↗
Setting time	↗	↗↗	↗↗
Heat of hydration	↘	↘↘	↘↘
Early strength	↘	↘↘	↘↘
Final strength	↗	↗	↗
Sulphate resistance	↗	↗↗	↗↗
Permeability (chloride)	↘	↘↘	↘↘
Alkali-aggregate reactivity	↘	↘↘	↘↘
Shrinkage	0	0	↗

↗ increase
↘ decrease
O neutral effect

The inert mineral components like limestone do exert similar influences as the active materials in terms of water requirement, setting and heat of hydration, but they will not improve the final strength and durability characteristics of the cement.

Other cement properties than the above mentioned are generally not affected to a great extent by the addition of mineral components.

3.4 Gypsum

(see also paper on cement hydration)

The main function of the gypsum in cement is to regulate the cement setting, but the gypsum also influences other cement properties such as grindability, flowability and storage stability, volume stability and strength.

The use of anhydrite instead of gypsum helps to reduce the risk of false setting and to improve the storage stability and flowability of the cement at high grinding temperature (see also chapter 5.6). In case of highly reactive clinkers, proper set retardation may, however, be a problem and blends with gypsum have to be used.

The substitution of natural gypsum by by-product gypsum may sometimes cause problems with setting and strength development due to potential presence of impurities in such type of materials.

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3.5 Chemical admixtures

The chemical admixtures, which can be added at the cement mill, are divided into the two following main groups:

- grinding aids having mainly a positive effect on the grinding energy
- performance modifiers influencing significantly the cement quality, in particular water requirement and strength development

The first group of admixtures (typically organic compounds based on alcohol and amines) do as mentioned not really change the engineering properties of cement. The action of the grinding aids is based on the reduction of the adhesive forces between the cement particles. They may, however, facilitate the handling of the cement due to the resulting improvement in flowability.

The performance modifiers are of similar nature as the products used in the concrete mix. Such admixtures are generally based on accelerators and water reducers and thus improve the workability and strength development of the cement.

Needless to say that the use of chemical admixtures is only worthwile if there is a real benefit with regard to the economy or performance of the cement to be produced.

4. OPTIMUM PROPORTIONING OF THE CEMENT COMPONENTS

4.1 General

The proportioning will be discussed here mainly from the point of view of cement performance. The economic aspects, which are of course of primary importance for the proportioning (see chapter 2.5), will not be dealt with, as they greatly depend on the specific circumstances.

For a given cement performance, the proportioning is basically controlled by the quality of the available cement components and the selected fineness of the cement and its components. Further limitations are set by the standards, which specify the permitted contents for the different cement components.

4.2 Clinker and mineral components

Portland cements

In case of the Portland cements, the flexibility in proportioning of clinker and mineral components is obviously limited. According to the European Norm, up to 5% of mineral component can be added to the cement, whereas ASTM does virtually not allow the addition of mineral components besides clinker and gypsum. The focus in the optimisation of the cement properties lies therefore in the determination of the proper gypsum dosage (see chapter 4.3).

Blended cements

The most critical point of the blended cements in terms of cement performance is the decrease in early strength. The dosage of mineral components in general purpose applications, where a similar strength development as for the Portland cement has to be achieved, is therefore limited.

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The possible dosages in such applications are the highest for the latent hydraulic mineral components and go gradually down for the pozzolanic and inert materials. Typical proportioning limits for the main mineral components in the cement are:

- blast furnace slag: 30 - 40%
- fly ash (class F): 15 - 30%
- natural pozzolan: 15 - 30%
- limestone. 10 - 20%

For blended cements used in special applications related to low heat evolution and durability, the early strength development is not of primary importance and the dosages of the mineral components can be higher. Some guide values on the proportioning in these applications are given in Table 5. It has, however, to be mentioned that always specific tests should be carried to verify the compliance with the application requirements.

Table 5: Guide values for proportioning of mineral components in cements for special applications

	Blast furnace slag	Fly ash (class F)	Natural pozzolan
Low heat of hydration	> 50%	> 30%	> 30%
Suphate resistance	> 70%	>30%	> 30%
Low chloride permeability*	> 60%	> 40%	> 40%
Avoidance alkali-aggregate reaction	> 40%	> 25%	> 15%

*provided the w/c-ration in concrete is sufficiently low

The actual proportioning of the mineral components in all applications will obviously also depend on the selected cement fineness and on the permitted contents specified in the respective standards.

4.3 Gypsum

(see also paper on cement hydration)

Portland cements

In Portland cements, the gypsum dosage has to be adjusted to the reactivity of the clinker (i.e. C3A and alkali content) and the cement fineness to ensure proper set retardation. Further adjustments may be necessary depending on the obtained grain size distribution and the grinding temperature in the cement mill.

It is usually assumed that the gypsum dosage for proper set retardation is more or less equivalent to the one required for best strength development and volume stability of the cement. A practical method to find the optimum gypsum content is described in the ASTM standard C 563 ("Standard test method for optimum SO3 in Portland cement"). It is well possible that the SO3 content at the optimum gypsum content would even be above the maximum value given by the standards.

In case that the Portland cement shows false setting tendency and problems with flowability and storage stability, the gypsum content should be lowered or part of the gypsum should be replaced by natural anhydrite. Replacement levels of up to 60% are possible for all type of clinkers without having any problems with set retardation.

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Blended cements

In case of blended cements, the situation gets much more complex and there exists no clear procedure on how to adjust the gypsum dosage. Studies at HMC have indeed shown that the optimum gypsum content has to be evaluated for each individual cement type. Nevertheless, the findings indicated that the optimisation of the gypsum content in blended cement can be a very effective means for the improvement of the cement quality.

Special attention in the determination of the optimum gypsum content has to be given to cements, which contain limestone filler. In such cements, it may be possible to reduce the gypsum content, since limestone acts also as a set retarder.

4.4 Chemical admixtures

Grinding aids are added at the cement mill at dosages, which are generally below 500 g/t. The determination of the optimum dosage for a specific grinding aid depends mainly on the cement fineness and the characteristics of the cement mill.

The dosage of the performance modifiers will essentially be determined by the objective for their use. The main purpose of such admixtures is generally to achieve a desired cement performance or to maintain the cement quality at a lower clinker content.

5. CEMENT GRINDING

5.1 General

The cement components have to be ground to fine particles, in order to attain the required cementitious properties. The fineness after grinding is usually characterised by the specific surface area or by the particle size distribution (PSD). The type of cement mill used can have a considerable effect on the PSD.

During the grinding process with the traditional systems (ball mills), only a small portion of the introduced energy is consumed for the comminution of the cement particles. A large quantity of heat is set free and the temperature of ground cement increases appreciably. In the modern grinding systems, less heat is produced, resulting in lower cement temperature during grinding.

Both, the fineness and the temperature of grinding are principal factors in determining the cement properties.

5.2 Description of fineness

5.2.1 Specific surface area

The specific surface area of cement is usually determined by the Blaine method. The Blaine value is calculated from the air permeability of a cement sample compacted under defined conditions. The resistance to air flow of a bed of compacted cement depends on its specific surface. The Blaine specific surface is not identical with the true specific surface of the cement, but it gives a relative value which suffices for practical purposes.

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An absolute measurement of the specific surface can be obtained by the nitrogen (or water vapour) absorption method - BET. In this method, the "internal" area is also accessible to the nitrogen molecules and the measured value of the specific surface is therefore considerably higher than that determined by the air permeability method:

Method	Blaine	Nitrogen Absorption (BET)
Cement A	2'600 cm2/g	7'900 cm2/g
Cement B	4'150 cm2/g	10'000 cm2/g

The Blaine value can sometimes be misleading, especially in the case of outdoor stored clinker, blended cements - consisting of a more easily grindable component - and clinkers containing underburnt material which is easier to grind. The properties of such cements can often be poorer compared to other ground to the same specific surface.

5.2.2 Particle size distribution

Cements of the same specific surface may have different PSD and different properties. Thus, the specific surface is not the only fineness criterion determining the properties of a particular cement composition.

The determination of the PSD can be carried out by the following methods:

- mechanical sieving (residues on sieves of a definite size (e.g. 32, 45 and 60 μμ))
- laser and sedigraph (residues over the whole range of particles sizes)

The mechanical sieving is usually applied in the cement plants. Due to the limitations in sieve sizes, this method does not allow to measure the whole range of particle sizes.

The overall particle size distribution of cement is commonly analysed by means of the theoretical distribution according to Rosin-Rammler-Sperling (RRS), which is described be the following formula:

$$\ln [\ln (100/Rd)] = n [\ln (d) - \ln (d')]$$

being:

Rd = % of particles with diameter greater than d (residue)

d = particle size in μm

d' = characteristic diameter in μm (36.8% of the particles greater than d')

n = slope of RRS straight line

The data obtained in the particle size analysis is accordingly plotted in a so-called RRS-diagram (see Figure 7), having a double logarithmic ordinate (y-axis) and a logarithmic abscissa (x-axis). After linear regression of the particle size distribution, the slope n of the straight line and the characteristic diameter d' (at 36.8% residue) can be calculated.

The slope n and diameter d' are the significant values for the particle size distribution. The first characterises the degree of distribution (wide-narrow), whereas the second one states its location and is an indicator for the overall fineness. High n values results from a narrow PSD and low d' values from a high overall fineness.

Differences in the PSD of cement can also be seen in the relation between the traditionally measured Blaine values and sieve residues. At same sieve residue, the Blaine tends to be lower with higher n values (see also Figure 8).

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Figure 7: Particle size distribution of cement in RRS-diagram

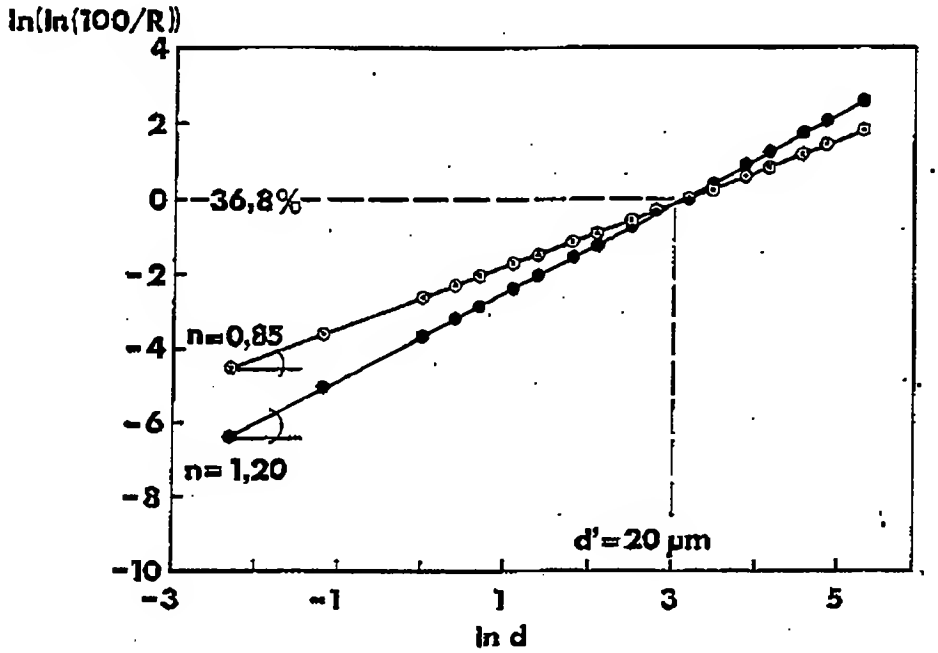
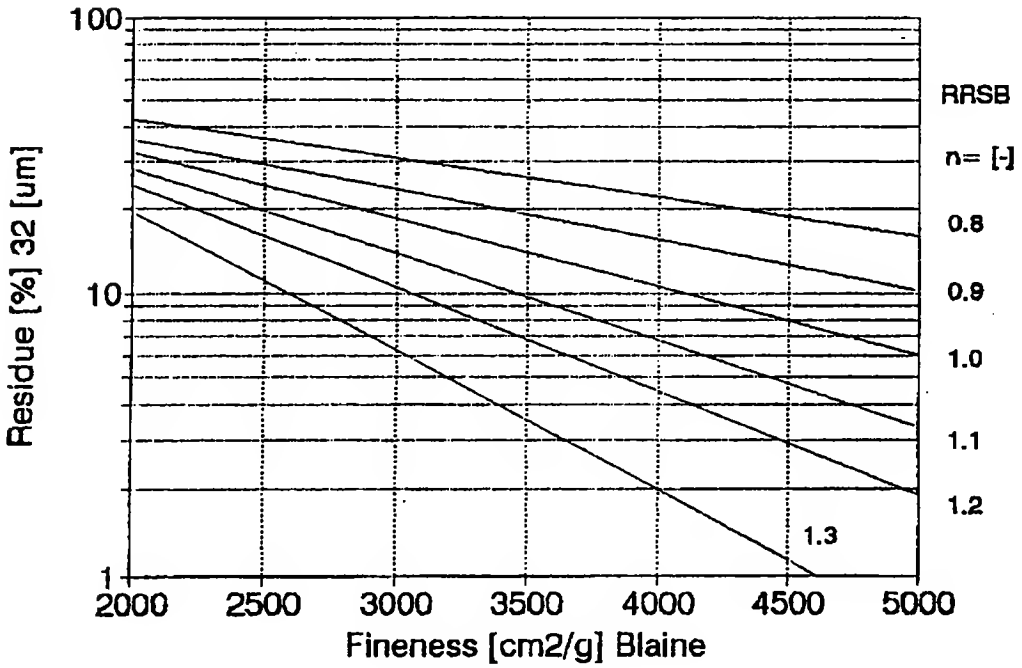


Figure 8: Correlation between Blaine and residue 32 µm for different n values (data FLS)



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5.3 Particle size distribution in the different grinding systems

The grinding in the modern cement mills goes together with a narrower PSD of the produced cements. Due to the more efficient grinding process, less under- and over-size particles are produced, which obviously leads to a shorter particle range and to higher steepness of the PSD.

The steepness n as expressed by the RRS-distribution ranges from 0.8 for an open circuit ball mill up to 1.2 for the newest grinding systems like vertical mill. Typical n values of cements ground in various industrial mill systems are indicated in Table 6.

The characteristic diameter d' of commercial cements produced in the different grinding systems varies typically between 10 and 30 µm. For identical specific surface, the d' values are lower in the systems which give a narrower PSD, that means that the overall fineness of the cement at same Blaine will be higher. At same d', the Blaine will be lower when the PSD gets narrower.

Table 6: Typical n values for PSD of cements ground in various industrial mill systems

Mill type	Steepness n of RRS-distribution
Ball mill (open circuit)	0.8 - 0.9
Ball mill (closed circuit)	0.9 - 1.0
Ball mill (high efficiency separator)	1.0 - 1.1
Vertical mill, roller press, Horomill	1.1 - 1.2

As mentioned before, the different PSD of the various grinding systems are also reflected in the relationship between Blaine and sieve residues (e.g. on 45 µm), which are the usual fineness measures applied in practice. The corresponding trends observed in the modern grinding systems compared to the traditional ball mills are as follows:

- lower Blaine at same sieve residue or
- lower sieve residue at same Blaine

It is important to mention that, in view of certain quality problems experienced with a too narrow PSD, the actual tendency for the new grinding systems is to adjust the PSD to a somewhat wider distribution.

5.4 Grinding of Portland cements

5.4.1 Influence of Blaine fineness

Since the hydration starts on the surface of the cement particles, it is the specific surface area of Portland cement that largely determines the rate of hydration and thus the setting and hardening rate. To achieve a faster hydration and strength development, rapid-hardening cements are ground finer than ordinary Portland cement. It is common practice to produce cement of various strength classes from one clinker by altering the fineness to which it is ground. The Blaine value of cement varies between 2'500 cm2/g for ordinary Portland cement (type I, ASTM) and 5'000 cm2/g for high early strength cement (type III, ASTM).

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The rate of hydration is slowed down by the presence of cement gel and if a large quantity of gel is formed rapidly, because of a large cement surface, the inhibiting action of the gel soon takes place. For this reason, extra fine grinding is efficient only for the early strength up to 7 days. Moreover, the rate at which the strength of concrete increases is substantially lower than that of mortar (see Figure 9).

Considering the energy consumption for grinding, the fine grinding is often not economically feasible. In those cases, where high early strength is not required, fine grinding is of little value (see Figure 10). A large number of concrete applications are unable to exploit the effects of fine grinding.

The relations between the Blaine fineness of cement and concrete properties can be summarised as follows:

1) Increasing the fineness of cement, reduces the amount of bleeding in concrete (see Figure 11).
2) Increasing the fineness of cement above 3000, increases somewhat the water requirement of concrete. Compared to the influences other than cement on the water requirement of concrete, the influence of cement fineness is considerably smaller.
3) The strength of concrete is influenced by the fineness of cement. The early compressive strength increases with an increase in cement fineness. The difference in compressive strength due to the difference in fineness of cement, is considerably smaller at 28 days and at later age (see Figure 9).
4) The fineness of cement influences the drying shrinkage of concrete. When the water content is increased because of fineness, the drying shrinkage is increased.

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Figure 9: Effect of cement fineness on strength of mortar and concrete

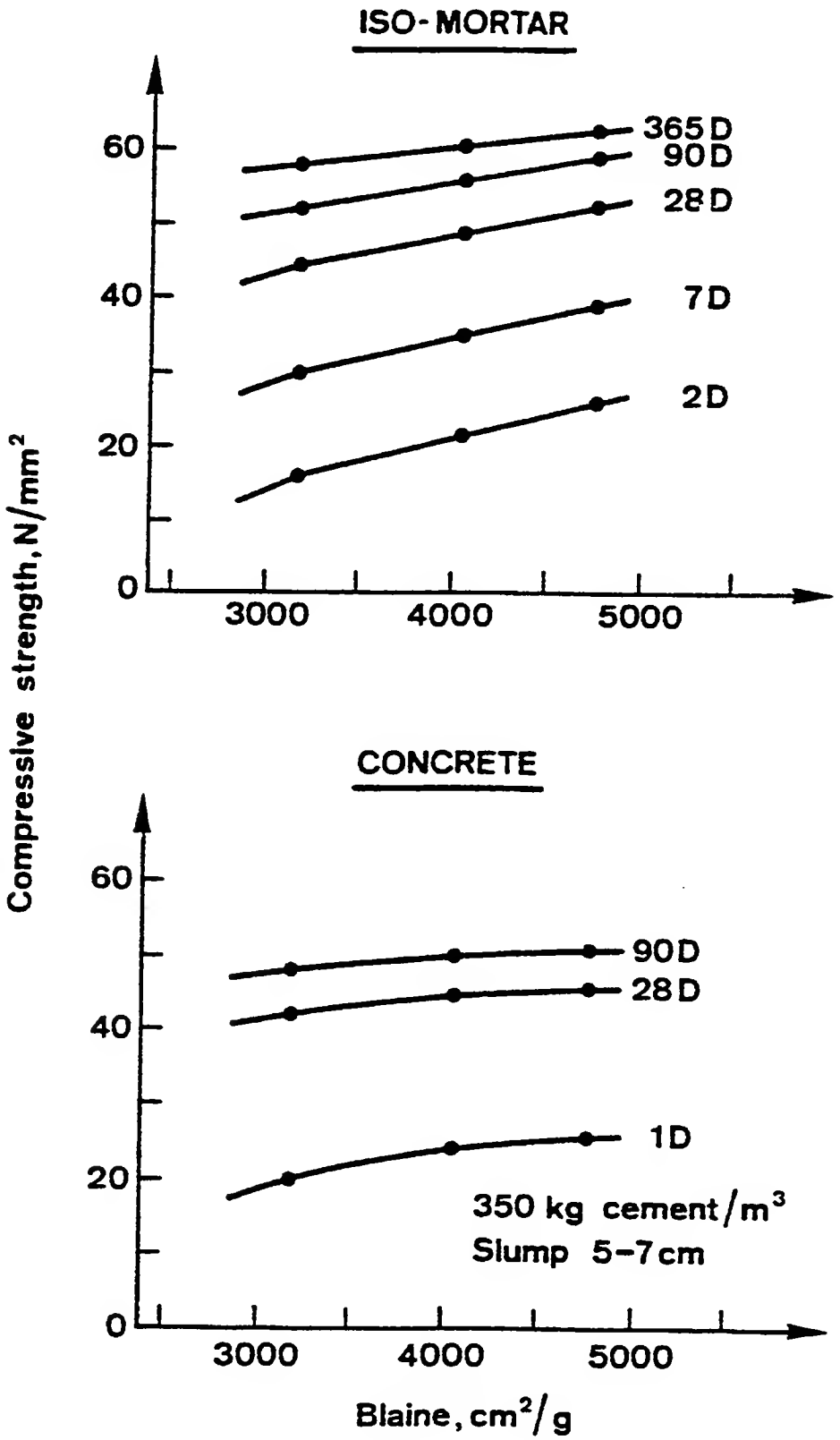

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Figure 10: **Relative specific energy consumption and compressive strength development**

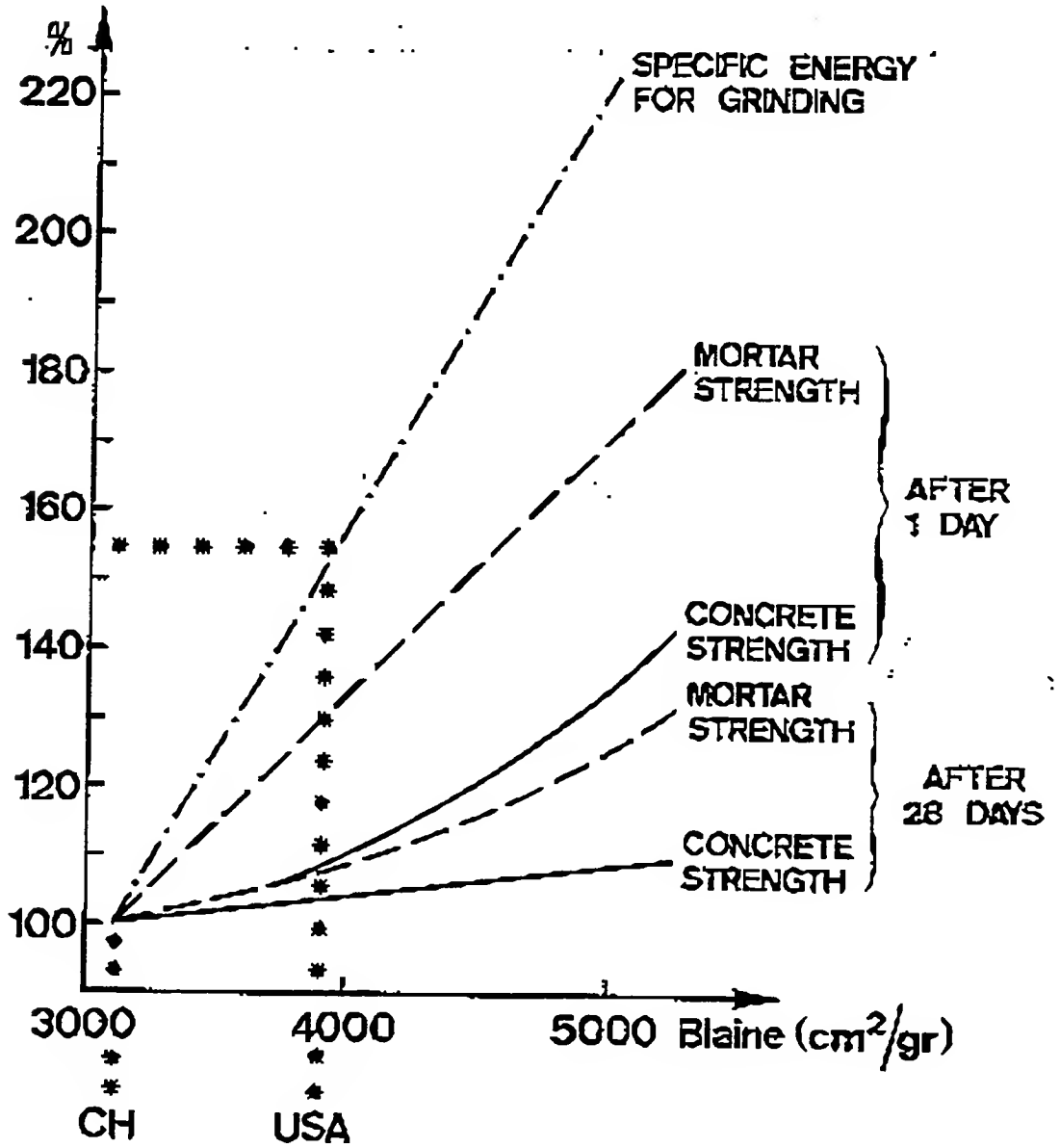

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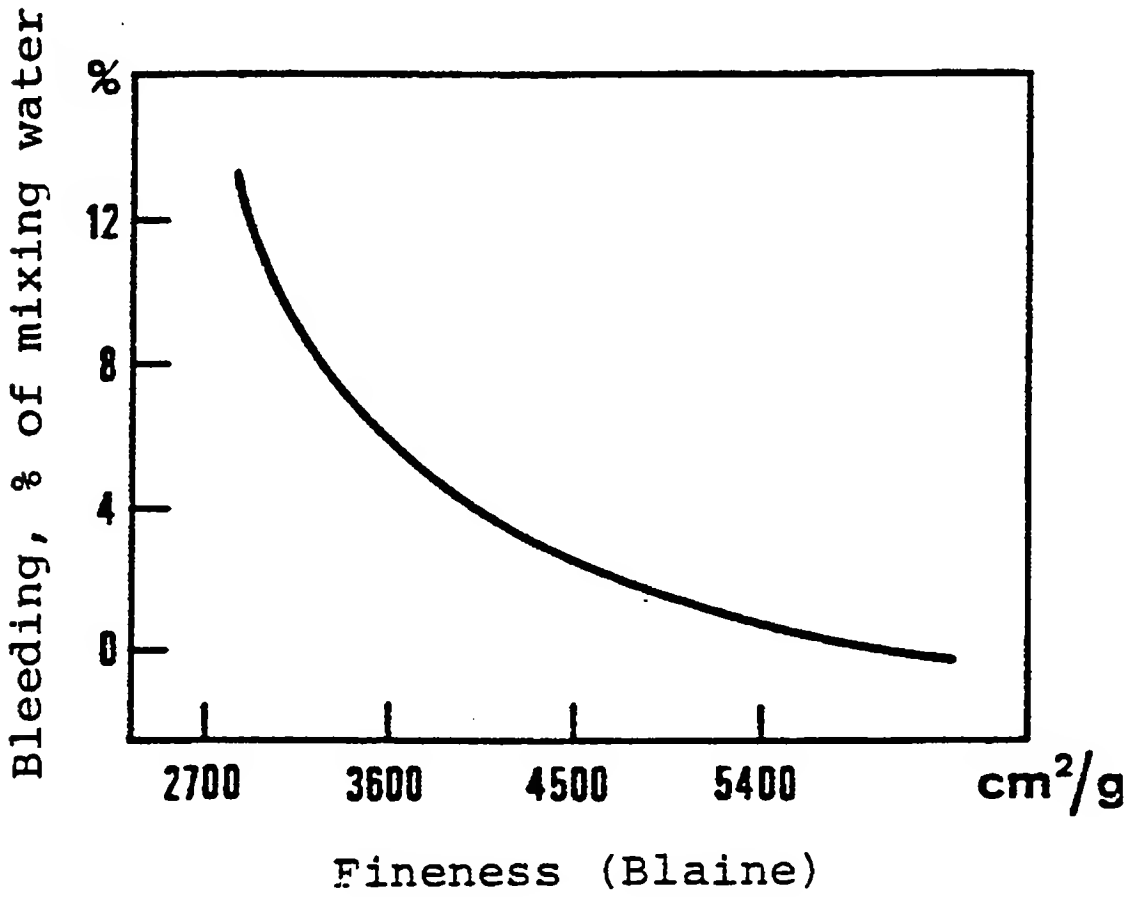
Figure 11: Effect of cement fineness on bleeding of concrete (non air-entrained concrete, w/c-ratio = 0.57)



5.4.2 Influence of particle size distribution

The influence of the fineness on the cement properties can be described more precisely, when taking into account the PSD of the Portland cement. The PSD is of particular importance with respect to workability and strength development.

The workability of Portland cement and concrete may impair when the PSD becomes narrower (at constant specific surface). On one hand, the water requirement for a certain consistency tends to increase and, on the other hand, the faster conversion of aluminate at narrow PSD may lead to early stiffening problems.

The mentioned stiffening problems may especially occur if clinkers of high reactivity (high C3A and alkali content) are ground together with the gypsum at low temperatures (little formation of easily soluble sulphates), as it is the case in the modern grinding systems. With such clinkers, the proper adjustment of the wideness of the PSD and/or of the calcium sulphate carrier is therefore important.

The effect of the PSD of Portland cement on strength development is not always clear. The general trends can be summarised in the following way:

- The most valuable particles for early strength are the ones between 0 - 8 µm. The Blaine value is thus a good indicator in this respect, as it is proportional to the portion in this fraction.

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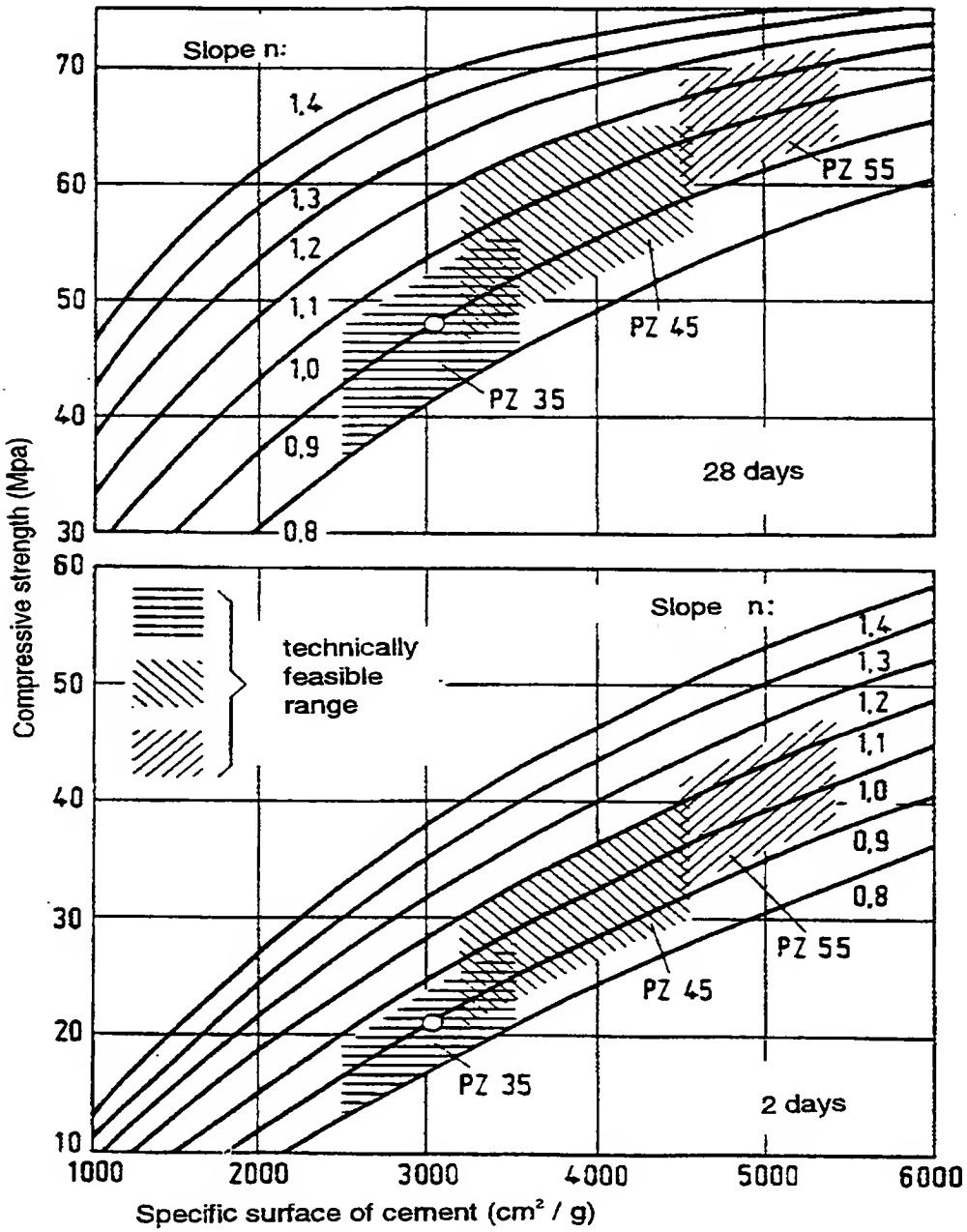
- The 28 day strength is mainly controlled by the amount of particles in the range between 2 - 24 µm, which is proportional to steepness n of the PSD.

The increase in the steepness n at a given Blaine is accordingly an effective means in improving the strength potential at 28 days as illustrated in Figure 12. The positive effects of higher n values are, however, less pronounced on concrete.

Figure 12: 2 day and 28 day compressive strength of Portland cement, as a function of the specific surface area and the slope of the RRS-distribution of the cement



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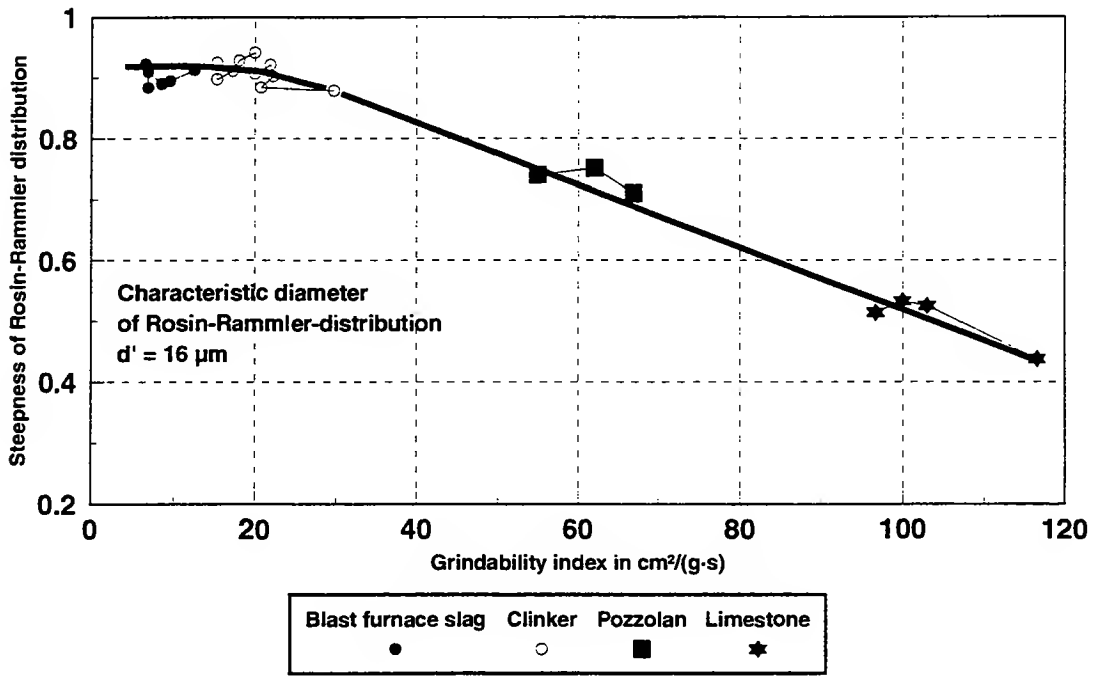
5.5 Grinding of blended cements

5.5.1 General

The properties of blended cements are decisively influenced by the fineness of the cement and its components respectively. Blended cements must generally be ground to a higher overall fineness than Portland cements to maintain a similar strength development.

The grinding behaviour of the different components in blended cements may vary quite significantly as illustrated in Figure 13. At constant fineness, the softer materials like limestone and natural pozzolan yield a wider PSD than the clinker and blast furnace slag. Despite its worse grindability, the grain size distribution of blast furnace slag does, however, not differ too much from the one of clinker. The mentioned differences in grindability are of great significance in the grinding of blended cements.

Figure 13: Steepness of the RRS-distribution of ground blast furnace slag, clinker, pozzolan and limestone at same characteristic diameter d' in function of the grindability index



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The question to which fineness the components of blended cements shall be ground to obtain optimum cement properties is often debated. The general concept of HMC is that the hydraulic potential of the clinker should be used as much as possible by grinding it to a sufficiently high fineness. The mineral components may be ground coarser, but latent hydraulic and pozzolanic materials must still have a sufficient fineness to be suitably activated to provide good final strength.

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5.5.2 Compound grinding

The compound grinding of clinker, gypsum and mineral component(s) is still the most common practice for the production of blended cements. The combined grinding with mineral components softer than the clinker like limestone will widen the grain size distribution of the resulting blended cement, whereas the mixture of clinker with a harder material like blast furnace slag will give a somewhat steeper PSD than the ground clinker alone. The different PSD can be explained by the fact that for compound grinding, the harder material is enriched in the coarser fraction and the softer material in the finer fractions of the cement.

In compound grinding, the different cement components can accordingly not be ground individually or independently from each other. For a given grinding system, the fineness of the components is pre-determined by their respective grindabilities; it is thus not possible to adjust freely their fineness. The inevitable enrichments of certain components in the fine or coarse fraction of the blended cement are, however, reduced with the modern grinding technologies.

This lack of flexibility in compound grinding may limit the optimisation of the properties of blended cements. With soft mineral components, the clinker will always remain rather coarse, in particular at higher replacement levels, so that its hydraulic potential can not be fully exploited. On the other hand, there might be an overgrinding of the mineral component like in the case of the natural pozzolans leading to an increase in water demand.

In case of slag cement, the clinker will get indeed finer and contribute as desired to the strength development. The slag may, however, not be adequately refined and activated.

5.5.3 Separate grinding

Separate grinding of blended cements gives more flexibility in the design and optimisation of the cement quality than compound grinding, since it permits free choice of the fineness of the cement components. Nevertheless, the opinions on the real benefits of separate grinding are still controversial.

In the following, the experience with separate grinding for the most relevant blended cements containing blast furnace slag, fly ash, natural pozzolan and limestone is discussed.

5.5.3.1 Slag cements

The studies on separate grinding of slag cement revealed that the fineness of the clinker and slag influence the cement quality in the following way:

- the clinker fineness is mainly related to early strength. In cements with low slag content (<30%), the clinker fineness will also have an impact on final strength
- the slag fineness determines mainly the final strength (at very high fineness, also significant contribution to early strength possible). For good workability of the cements, the PSD of the slag should not be too narrow.

In separate grinding, it is thus in principle possible to fine tune the strength curve according to the above relationships. When the grinding energy is kept constant, the observed improvements compared to intergrinding do, however, not seem to be too significant.

An interesting advantage for separate grinding may be in some cases the activation of the slags through very fine grinding. Recent studies have shown that at Blaine finesses higher than 4000, there is a considerable potential to improve the strength development at all ages (see also Figure 14).

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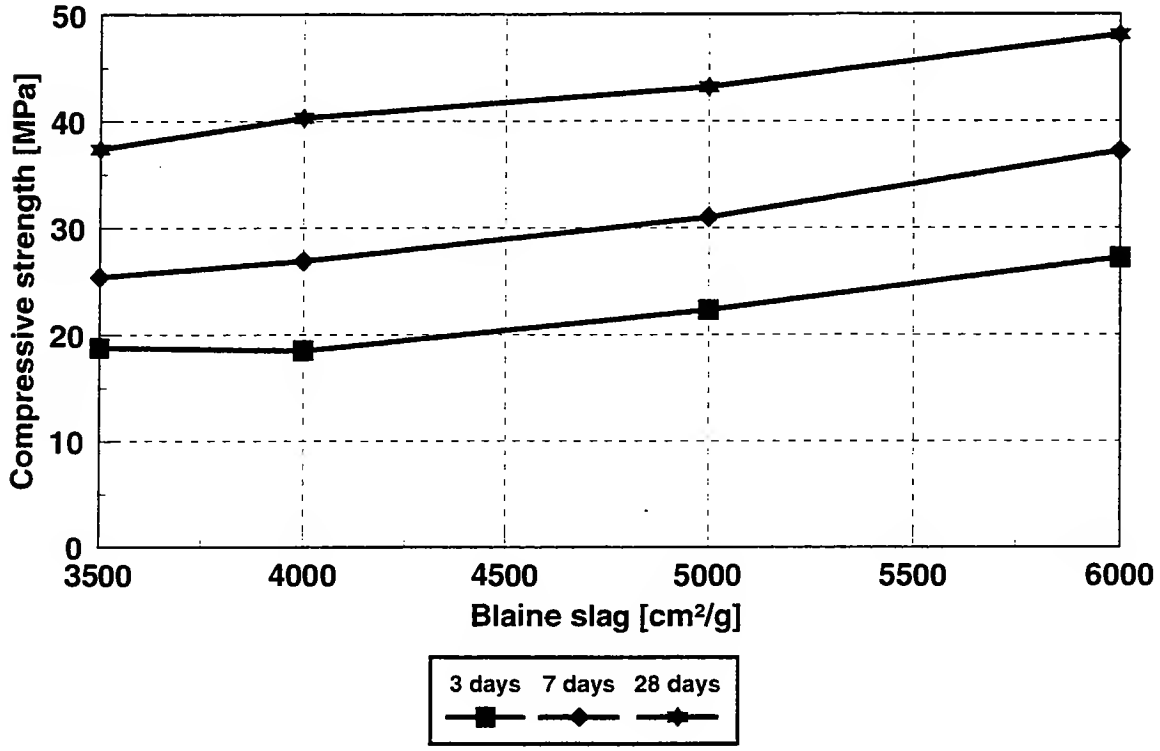
Figure 14: Influence of Blaine fineness of slag on compressive strength of ASTM mortar for cement with 40% slag

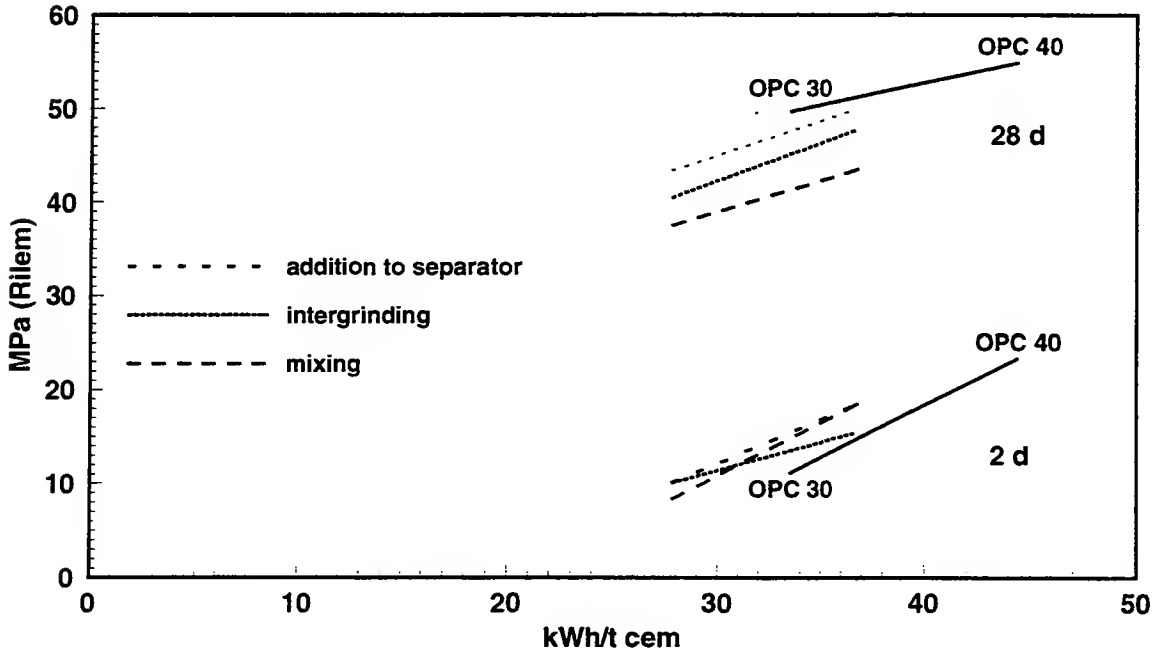


5.5.3.2 Fly ash cements

The separate grinding of fly ash cement as such has hardly been investigated. It appears that at constant grinding energy separate grinding gives certain possibilities to fine tune the final strength development of the fly ash cement.

Presently the most appropriate solution to produce fly ash cements is to add the fly ash to the separator of the grinding system. The example in Figure 15 of a cement containing 16% fly ash demonstrates that in terms of quality and consumption of grinding energy, this seems to be the best solution, also compared to intergrinding.

Figure 15: Influence of treatment undergone by the fly ash on the strength of cement with 16% fly ash



5.5.3.3 Pozzolanic cements

Separate grinding of cements with natural pozzolan gives a somewhat higher early strength and lower final strength than intergrinding at constant energy input. This relationship seems quite logical as in intergrinding the clinker responsible for the early strength remains rather coarse and the pozzolan contributing to the final strength is refined.

The studies on separate grinding of pozzolanic cements carried out at "Holderbank" showed a certain potential to increase the clinker factor at same cement quality, though at the expense of a higher grinding energy. A typical increase might be in the order of 5% as it is illustrated by the comparison of intergrinding and separate grinding for a pozzolanic cement from Mexico in Table 7.

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Table 7: Comparison of intergrinding and separate grinding for pozzolanic cement from Mexico

Physical and mechanical properties	Compound grinding (ball mill)	Separate grinding (ball mill and vertical mill)
Cement		
Pozzolan (%)	20	25
Blaine (cm2/g)	4170	4120
n (-)	1.0	x)
R 45 µm (%)	3.3	7.5
Paste ASTM		
Water demand (%)	28.3	27.0
Setting time (min.)		
- initial	160	145
- final	225	175
Mortar ASTM		
w/c-ratio	0.52	0.53
Compressive strength (MPa)		
- at 1 day	11.0	11.6
- at 3 days	20.9	20.7
- at 7 days	27.8	26.0
- at 28 days	35.0	34.8

x) Pozzolan: n = 0.95 R 45 µm = 20.9% Blaine = 2170 cm2/g
Clinker/gypsum: n = 0.95 R 45 µm = 3.1% Blaine = 4300 cm2/g

5.5.3.4 Limestone cements

Studies on separately ground limestone showed that the limestone fineness as such has virtually no influence on the strength development. The PSD of the limestone powder can, however, play a role with regard to the workability characteristics of the limestone cement: a wide distribution is in this respect more favourable than a narrow one.

In combined grinding with clinker, the limestone is automatically ground to the favourable wide PSD. This quite advantageous behaviour in intergrinding, at least at lower limestone dosages (up to 20%) may also explain the fact that separate grinding of limestone cement to improve cement quality has usually not been applied in practice.

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5.6 Temperature and moisture conditions

The grinding of cement influences the properties of cement not only through an increase in fineness, but also through the reactions taking place in the cement mill. Depending on the temperature and moisture conditions prevailing in the mill, dehydration and hydration occur which influence the grinding process, flowability, lump formation in silos, setting and hardening of cement.

Due to the heat liberated during the grinding process, the temperature in the traditional ball mills rises to temperatures above 100°C. In the modern grinding systems, the grinding temperatures are significantly lower (down to 50 - 60°C). In a particular mill, the exit temperature of cement can vary in a wide range, in function of the inlet temperature of clinker, cooling conditions and fineness of grinding.

The effect of the grinding temperature on the cement properties is mainly related to the dehydration of gypsum. With increasing temperature, gypsum ($CaSO_4 2H_2O$) gets unstable and transforms to hemihydrate and anyhdrite III under the release of water. The dehydration of the gypsum will not only depend on the temperature, but also on the time the gypsum is exposed to this temperature (see Figure 16). Another factor of less importance for gypsum dehydration is the humidity in the mill atmosphere (see Figure 17).

Figure 16: Influence of temperature on the dehydration of gypsum

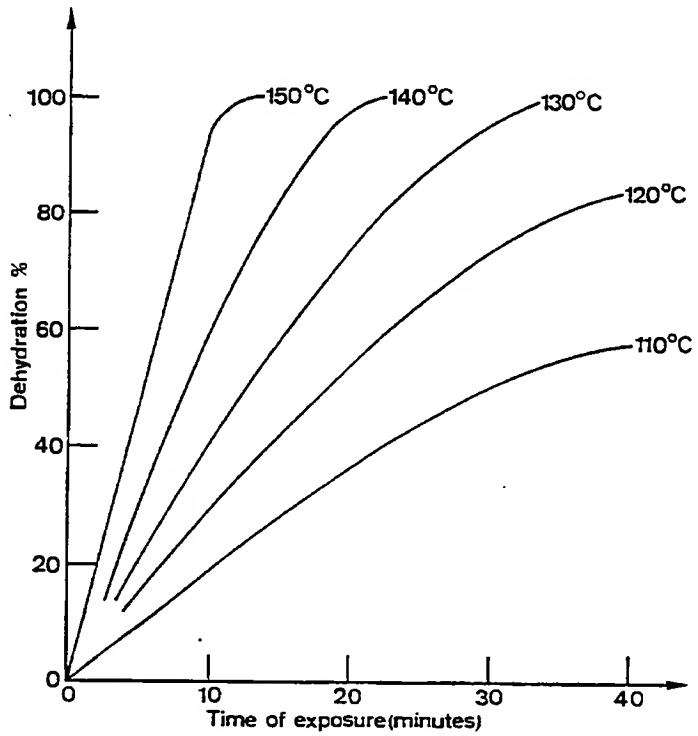

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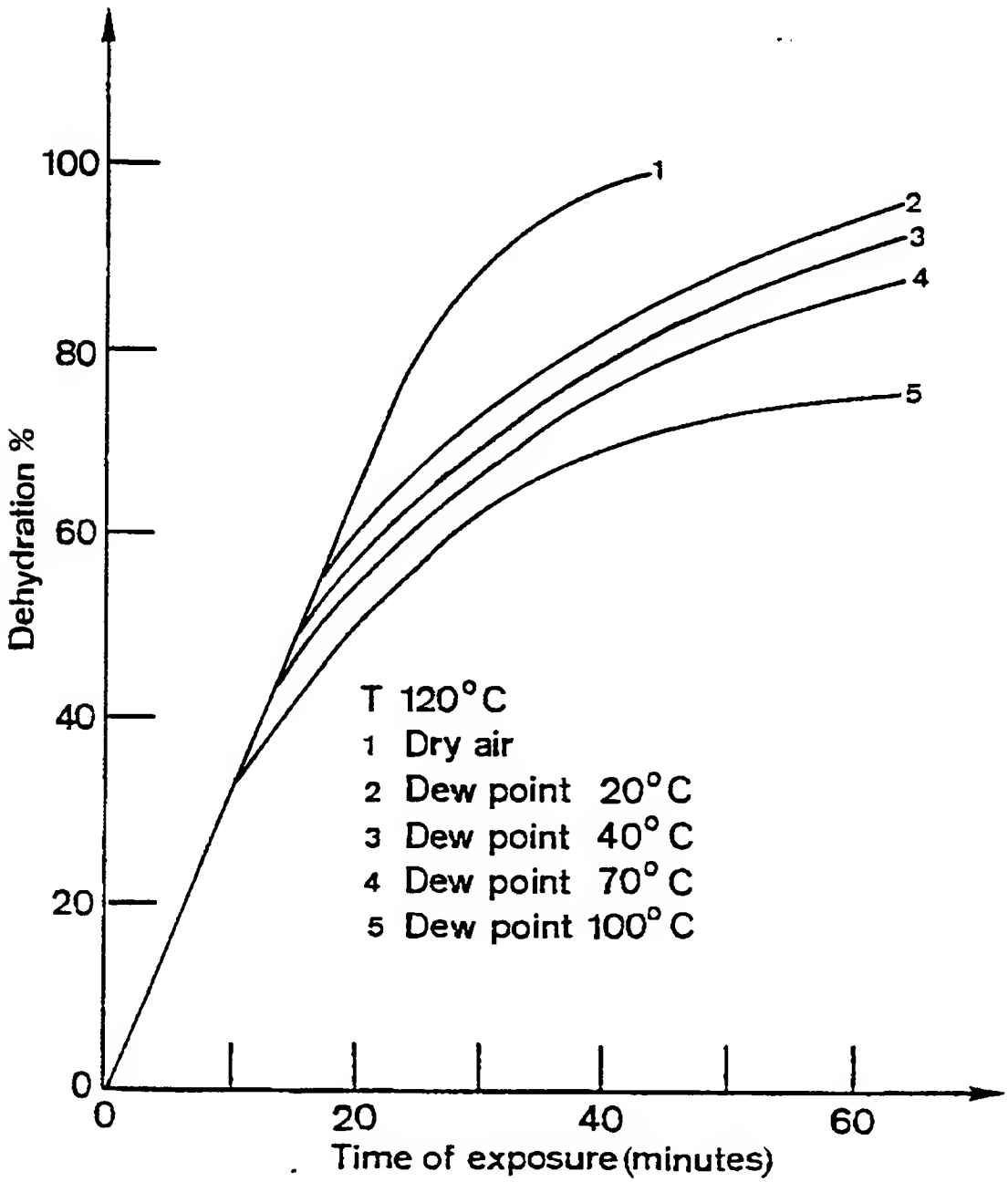
Figure 17: Influence of humidity on the gypsum dehydration.



According to the degree of dehydration, the gypsum will exert a different influence on the cement properties (see also paper on cement hydration). If great part of the gypsum is converted to the more easily soluble hemihydrate and anhydrite III, there may be some

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problems with false setting in case a clinker of low reactivity is used. On the other hand, a too low degree of dehydration may lead to flash setting tendency with reactive clinkers.

The dehydration of the gypsum may also have an impact on the storage stability of the cement. If the cement enters the silo with a high temperature (80 - 90°C), further water can be released form the still not dehydrated gypsum and lead to the hydration of the cement. These hydration reactions can cause lump formation and affect the strength of the cement (see Figures 18 and 19). Clinkers with high C3A and alkali content are particularly subject to hydration reactions during storage.

Figure 18: Lump formation in a storage sensitive cement after one week storage at various temperatures

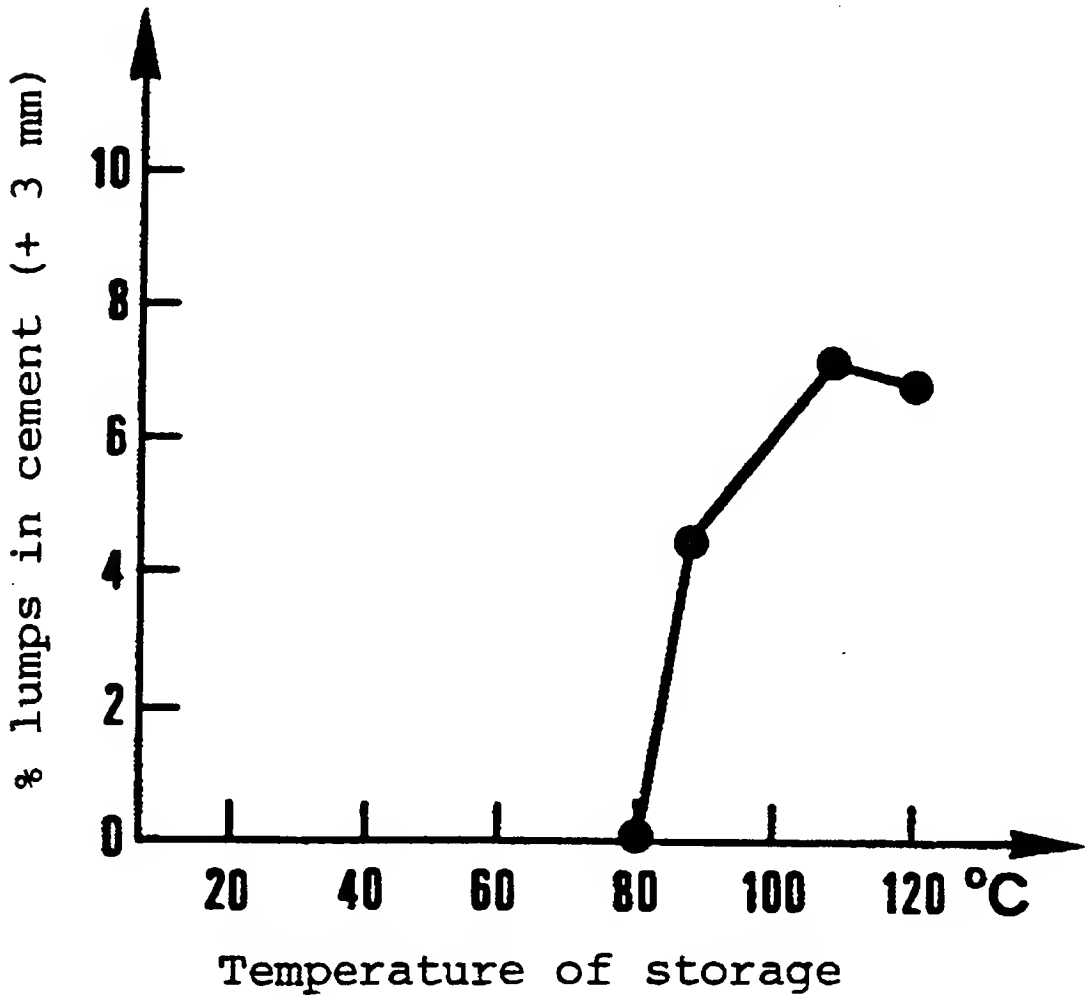

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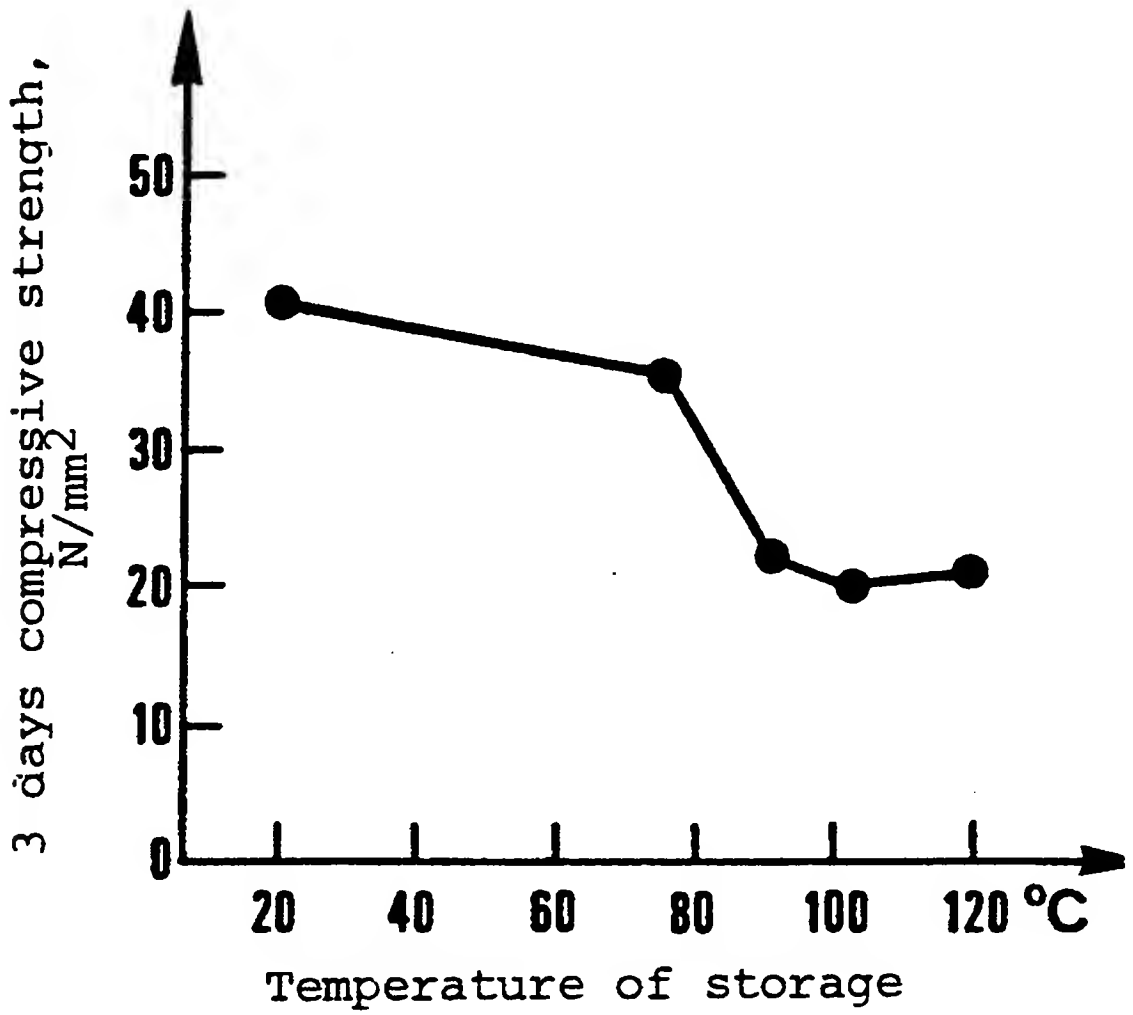
Figure 19: Compressive strength of a storage sensitive cement after one week Storage at various temperatures



Some measures to ensure the storage stability of the cement are:

- low cement temperature in the silo
- short storage time
- reduction of gypsum content in the cement
- substitution of gypsum by natural anhydrite

The storage stability is usually less problematic with the new grinding technologies,where the temperatures of the cement coming from the miss is generaly low. The cement temperatures in ball mills can be lowered by cooling the cement during the grinding process (e.g. water injection) or by installing a cement cooler after the mill. For the cooling by means of water injection, the temperature in the mill should always be kept above 100°C. Otherwise, prehydration of the cement and strength losses may occur.

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Chapter 5

Cement Standards

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CEMENT STANDARDS

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1. INTRODUCTION

1.1 Importance of Standards

In many of our daily activities we are unconsciously confronted with standards.

When we go to see a football game at 3.30 p.m. we seldom realize that if it really begins at the scheduled time it is simply because we have standardized our time according to fixed rules. Then, it is up to us to decide buying an expensive Swiss watch in case we look for a reliable product, proven to keep its conformity to the time standard long enough.

Many other less trivial examples could be reported to witness to what extent our life is defined and conditioned by standards of any type (mainly moral...).

We need standards because they are the way we have to fix generally valid and accepted rules to measure and compare things with one another and so define the relationship between "producer" and "consumer".

Producing according to an official standard means then putting on the market a product of well defined, measurable and recognizable characteristics.

This will positively affect the producer's position with respect to own customers in terms of reliability, confidence and commercial relationship and will protect him against unfair competition and undue complaints.

The consumer is then assured that by using a standardized product he can rely on a market-conforming, well defined and reasonably constant quality product, that will reflect in a consistent and cost-effective production.

The designer and contractor will be provided with confidence about the essential material characteristics they need to design and execute the desired structures.

It is therefore of vital importance that cement and concrete producers be aware of the expected developments of own standards and, if needed, take the necessary actions to influence development in the desired direction, be it through either direct or own association's involvement.

1.2 Development of Standards

Developing a standard is not an easy deal. From previous comments, it should be clear that a product standard involves both the producer' and the consumer's interests and this is not an ideal starting point to find rapidly an overall agreement on the standard objectives.

On the other hand, standards can meet easy and widespread acceptance only when all parts involved participated in its development.

Standards shall then be a common elaborate of

- the producer
- the seller
- the consumer
- the designer
- the official testing institute
- the universities or research laboratories
- the government.

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They should also be conceived (and periodically revised) taking into account

- the present technical situation of the industry
- the need of progress for the country
- an effective environmental compatibility (energy saving and by-products recycling)
- all technical improvements and economic changes that took place from the last edition
- modified market exigences and demands
- last but not least, the effort to pull down trade barriers in large State federations.

2. CEMENT STANDARDS

2.1 Product Standards

Product standards, which we are mostly interested in, are usually meant to define

- the quality of a product, in terms of composition, performance and application (i.e. the European ENV 197-1 standard on common cements)
- the common rules for uniformly testing and assessing the prescribed product quality (i.e. the associated EN 196 series of testing methods)
- the conditions for the use of the product in particular applications (i.e. recommendations for durability of concrete structures, a.s.o.).

They include classification, composition, specification and compliance criteria. As regards specification, that is a statement of one or a set of requirements the material shall comply with, they can be prescriptive- or performance oriented.

The prescriptive specification defines a product according to its composition, the performance specification defines a product according to its function in application, thus products of different composition can comply.

It is rather difficult to say which is better. On one hand a performance standard is supposed to be tightly related to the product's field performance and should better represent its ability to fulfill the customer's expectations and needs. On the other hand, assessing particular properties can involve use of complicated and time consuming testing methods; sometimes even the correlation between lab testing and field performance is still to be demonstrated. Then, in many cases, an easy prescriptive standard (i.e. maximum C_3A for sulfate resistance) is preferred.

We should not forget anyway, that some of the specifications for cement rely on performance testing, such as setting time and strength.

In the case of cement, product specifications take into account

- production aspects, as regards raw materials availability, plant design, environmental needs
- application aspects, as regards experience of use, available technology, needs of differentiation, customer expectations
- marketing aspects, as regards cost effectiveness of products, principles for fair competition, appropriate price/quality differentiation
- quality aspects, by setting minimum and/or maximum acceptable values for most of the standardized properties, as a protection against dangerous behaviour of non-conforming products.

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Some of the most recent international standards (e.g. ENV 197-1 on common cements) now include also a chapter on conformity evaluation.

This is a very innovative and clever approach to product standardization, since it introduces statistical tools to evaluate quality, not simply in terms of fulfilling absolute limit values, that are still considered as threshold values for conformity, but mainly taking into account the characteristic values, that include calculation of mean value and standard deviation of the measured property. An example will be given in the following pages.

After highlighting the merits of standards, we cannot help mentioning their main drawback. Product standards are capable to depict the present situation and technology only, without being able to forecast, and open paths for, technical development. This sometimes reflects in a serious hurdle to innovation, since a real new product will hardly find a corresponding standard to be referred to.

This is a strong argument for those people who are in favour of performance standards, that hinder technical progress much less than prescriptive ones.

Standardization bodies should therefore find the right balance and make their mind more progress-oriented, so to walk aside innovation and not run after it.

2.2 Significance of Specifications

We have seen that a specification is a requirement the product shall comply with. But why are these requirements specified?

The main concern in a cement standard is that the binder has to be used in constructions where safety, comfort and durability must be safeguarded. Additionally, special applications need special cement properties that have to be adequately described in specifications.

In the following, a series of examples indicate properties that are specified for cements and also the inconveniences they must prevent; as mentioned, requirements can be related either to cement composition or to performance.

- minimum setting time - to prevent early or flash setting of cement, that would excessively shorten the life time of the fresh and workable blend;
- soundness (Le Chatelier or autoclave test) - maximum expansion is specified to prevent the risk of cracking in hardened structures;
- maximum SO_3 content - same as above;
- maximum C_3A content - to limit reactivity to sulfate attack and heat development of cement;
- maximum C_3S content - to limit reactivity to pure water attack and heat development of cement;
- maximum heat of hydration - to prevent excessive heat development;
- compressive strength - to guarantee a minimum strength development in properly designed concretes and cementitious blends;

and so on.

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2.3 Testing Methods

Product standards usually deal with the characteristics of a product, making reference to the proper testing methods to be used for the assessment of these characteristics.

Standards about testing methods aim at defining uniform conditions under which the product is tested.

They fix the product's testing conditions, in the case of cement if the measurement of the property will be carried out on

- powder
- paste
- mortar
- concrete;

they define

- scope of testing
- referenced documents
- testing apparatus
- reactants
- calibration of apparatus
- testing procedure
- calculation of results
- reporting
- precision and bias of the method.

As for product specifications, the purpose is to have a common tool to evaluate the product and reproduce the results obtained by different testers with reasonably narrow deviation. So, a product with a wide (even transnational) market distribution can be checked in different places with sufficient confidence on the accuracy and reliability of results.

This is clearly assuming a growing importance in the view of free trade activities, but the most practical advantage for a plant technologist is to use the same language and the same criteria to identify products and properties.

Drafting testing methods is usually easier than for product specifications, because it does not involve commercial and political issues, nevertheless some main principles should be observed. A testing method should be

- ♦ reliable (repeatable and reproducible) and meaningful
- ♦ able to allow easy correlations between results and product characteristics and/or properties
- ♦ easy to be implemented in any official as well as plant laboratory
- ♦ as far as possible not hazardous to workers or environment.

Most of the main testing methods for cement are well established standards. If a new property needs to be measured, then a working group is usually formed and a draft test is issued; the following round-robin testing will prove the effectiveness of the method to transform it into an official standard.

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3. INTERNATIONAL CEMENT STANDARDS

Standards are usually issued on a National basis by the local standardization institute.

Such institutes are ASTM for USA, BSI for UK, DIN for Germany, AFNOR for France, and so on.

Recent formation of large State federations (i.e. the European Community) have brought to a centralized management of standardization, to make it compatible with the free trade of products among countries. The European Committee for standards (CEN) has then rapidly become the main technical reference for all member and affiliated countries in Europe and also in some other areas of the world.

In the context of international standardization of testing methods, even ISO, the International Standards Organization, has simply and directly adopted some of the new documents developed by CEN technical Committees.

As a consequence, the trend all over the world is towards a reduction of the number of standards to a selected minimum, even maintaining some national peculiarities. Therefore we are going to mention here the cement standards that gained worldwide recognition and application, the ASTM and the EN specifications, as well as the Australian standard, an example of a very simple and effective way to draft a technical specification for cement.

People that are interested in a complete compilation of the world cement standards can consult the publication "Cement Standards of the World", issued by Cembureau, Brussels.

3.1 The ASTM Standards

ASTM standards are widely adopted not only in the United States of America, but also in many other countries of Latin America, as well as Asia and Africa.

In the following pages, the specifications contained in the relevant ASTM cement standards are reported

- C 150 - "Standard Specification for Portland Cement"
- C 595M - "Standard Specification for Blended Hydraulic Cements"
- C 1157M - "Standard Performance Specification for Blended Hydraulic Cements".

a) Cements defined by **C 150** are divided in five different types.
   - Type **I** For use in general concrete construction
   - Type **II** For use when exposed to moderate sulfate action, or where moderate heat of hydration is required
   - Type **III** For use when high early strength is required
   - Type **IV** For use when low heat of hydration is required
   - Type **V** For use when high sulfate resistance is required

(Types I to III can also be produced as air-entrained cements, when air entrainment is desired; in this case they bear the suffix "-A").

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Table 1 ASTM C150 - Standard Specification for Portland Cement

Cement type →	I	I A	II	II A	III	III A	IV	V
SiO_2, min %	-		20.0		-		-	-
Al_2O_3, max %	-		6.0		-		-	-
Fe_2O_3, max %	-		6.0		-		6.5	-
MgO, max %	6.0		6.0		6.0		6.0	6.0
SO_3, max %								
when C_3A<8%	3.0		3.0		3.5		2.3	2.3
when C_3A>8%	3.5		not applicable		4.5		n.ap.	n.ap.
Loss on ignition	3.0		3.0		3.0		2.5	3.0
Insoluble resid.	0.75		0.75		0.75		0.75	0.75
C_3S, max %	-		-		-		35	-
C_2S, min %	-		-		-		40	-
C_3A, max %	-		8		-		7	5
C_4AF+C_3A, max %	-		-		-		-	25
% air in mortar								
min	-	16	-	16	-	16	-	-
max	12	22	12	22	12	22	12	12
Fineness, m^2/g								
turbidimeter	160	160	160	160	-	-	160	160
air permeability	280	280	280	280	-	-	280	280
Autoclave exp.	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
Compr. strength								
MPa 1 day	-	-	-	-	12.0	10.0	-	-
3 days	12.0	10.0	10.0	8.0	24.0	19.0	-	8.0
7 days	19.0	16.0	17.0	14.0	-	-	7.0	15.0
28 days	-	-	-	-	-	-	17.0	21.0
Gillmore setting								
minutes init.set	60	60	60	60	60	60	60	60
final set	600	600	600	600	600	600	600	600
Vicat setting								
minutes init.set	45	45	45	45	45	45	45	45
final set	375	375	375	375	375	375	375	375

For special applications, additional chemical and physical requirements are set, e.g.

- Limits on C_3A content for cement types III and III A for moderate (<8%) and high (<5%) sulfate resistance
- limits on the total silicate content (<58%) for types II and II A and heat of hydration for types II, II A, IV and V for low heat cement
- equivalent Na_2O content (<0.6%) for prevention of alkali-aggregate reaction
- 28-day strength when not specified in the main table.

b) Provisions for blended cements in **C 595** are as follows

- Type **IS** - Portland Blast-Furnace Slag Cement (slag content 25÷70%)
- Type **IP** - Portland-Pozzolan Cement (pozzolan content 15÷40%)
- Type **P** - Portland-Pozzolan Cement with low early strength
- Type **S** - Slag Cement (slag content > 70%)
- Type **I(PM)** - Pozzolan-modified Portland Cement (pozzolan content < 15%)
- Type **I(SM)** - Slag-modified Portland Cement (slag content < 25%)

these types can be **MS** (moderate sulfate resistent), **A** (air entrained) or **MH** (moderate heat of hydration).

Table 2a ASTM C595M - Specification for Blended Hydraulic Cement - Composition

Cement type	Clinker + gypsum	BF slag	Pozzolan / fly ash
IS (ev. MS, A, MH)	30 - 75	25 - 70	
S (ev. A)	< 30	> 70	
IP (ev. MS, A, MH)	60 - 85 *		15 - 40
P (ev. MS, A, MH)	60 - 85 *		15 - 40
I (PM) (ev. MS, A, MH)	> 85 *		< 15
I (SM) (ev. MS, A, MH)	> 75	< 25	

* These cements can be produced by blending pozzolan either with ordinary Portland cement or with an IS type cement.

Table 2b ASTM C595M - Specification for Blended Hydraulic Cement - Chemical Requirements

Cement type →	I(SM), I(SM)-A, IS, IS-A	S. S-A	I(PM), I(PM)-A, P, PA, IP, IP-A
MgO, max %	-	-	6.0
SO_3, max %	3.0	4.0	4.0
Sulfide sulfur, max %	2.0	2.0	-
Insoluble resid., max %	1.0	1.0	-
Loss on ignit., max %	3.0	4.0	5.0
Water sol. alkali, max %	-	0.03	-

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Table 2c ASTM C595M - Specification for Blended Hydraulic Cement - Physical Requirements

Cement type →	I(SM) I(PM) IS, IP	same with air en.	IS(MS) IP(MS)	same with air en.	S	SA	P	PA
Autoclave exp.	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
A. contraction	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Vicat setting								
minutes initial	45	45	45	45	45	45	45	45
hours final	7	7	7	7	7	7	7	7
% air of mortar	12	19±3	12	19±3	12	19±3	12	19±3
Compr.strength								
min. MPA 3 d	13.0	10.0	11.0	9.0	-	-	-	-
7 d	20.0	16.0	18.0	14.0	5.0	4.0	11.0	9.0
28 d	25.0	20.0	25.0	20.0	11.0	9.0	21.0	18.0
Heat of hydrat.								
kJ/kg max 7d	290	290	290	290	-	-	250	250
28 d	330	330	330	330	-	-	290	290
Water req. %	-	-	-	-	-	-	64	56
Drying shrink.	-	-	-	-	-	-	0.15	0.15
Mortar expans.								
max % 14 d	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02
8 weeks	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Sulfate resist.								
max exp.180 d	-	-	0.10	0.10	-	-	-	-

c) An interesting example of a performance oriented standard is ASTM **C 1157M**. Here blended cement are classified according to their performance, with no restrictions on the composition of the cement or its constituents.

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The specified cement types are based on specific requirements for general use (**GU**), high early strength (**HE**), resistance to sulfate attack (**MS** and **HS**), heat of hydration (**MH** and **LH**), low reactivity with alkali-reactive aggregates (**R**).

Table 3 ASTM C1157M - Performance Spec. for Blended Cement

Cement type →	**GU**	**HE**	**MS**	**HS**	**MH**	**LH**
Autoclave length change, max %	0.80	0.80	0.80	0.80	0.80	0.80
Vicat setting time, minutes						
initial	45	45	45	45	45	45
final	420	420	420	420	420	420
% air of mortar	to be specified on documents to customer					
Compressive strength, min MPa						
1 day	-	10.0	-	-	-	-
3 days	10.0	17.0	10.0	5.0	5.0	-
7 days	17.0	-	17.0	10.0	10.0	5.0
28 days	-	-	-	17.0	-	17.0
Heat of hydration, max kJ/kg						
7 days	-	-	-	-	290	250
28 days	-	-	-	-	-	290
Mortar bar exp. 14 days, max %	0.020	0.020	0.020	0.020	0.020	0.020
Sulfate exp., max % 6 months	-	-	0.10	0.05	-	-
1 year	-	-	-	0.10	-	-
Option R, low ASR reactivity						
expansion 14 days, max %	0.020	0.020	0.020	0.020	0.020	0.020
expansion 56 days, max %	0.060	0.060	0.060	0.060	0.060	0.060
Optional physical requirements						
Early stiffening, final penetration						
min %	50	50	50	50	50	50
Compressive strength, min. MPa						
28 days	28.0	-	28.0	-	22.0	-

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3.2 THE EUROPEAN STANDARD ENV 197-1

This standard was issued in the early 90's after some 25 years of drafting, with the aim at including in one and only specification all the cements produced all over Europe. It is clear that the extremely variable climatic, industrial and economic conditions of the continent were the main reason of the huge diversity of binders produced and consequently of the very tough job for the members of CEN to reach a reasonable compromise among member countries.

The ENV 197-1 standard is divided in nine chapters, the most important are

- chapter 4 "Constituents", dealing with cement components; here some main characteristics of the materials to be used in cement production are set
- chapter 5 "Cement types, composition and designation", where the official names and abbreviations are described for every type of cement, according to its composition
- chapter 6, 7, 8 "Mechanical-, Phisical-, Chemical requirements" setting limits for the main cement properties
- chapter 9 "Conformity Criteria", dealing with the procedures to apply for the autocontrol testing of cement and the subsequent evaluation of its conformity to the specifications.

Description of cement components (constituents) in Chapter 4 is very accurate, complete and rigorous, including additional specifications (Table 5) that define the main component characteristics in order to assure their performance in cement.

The ENV 197-1 standard recognizes and legitimates the use of some constituents that have a long tradition of use in many countries, but at the same time are not available in other areas (i.e. natural pozzolan, fly ash, burnt shale). There is absolute freedom, with the exception of Portland cement clinker as the only obligatory constituent, in the choice of materials to produce cements according to ENV 197-1, provided they comply with the requirements of Chapter 4.

This freedom is reflected in Chapter 5 (Table 4), where cement compositions are reported. As many as 25 different types of cements can be produced, using one or more of the indicated constituents.

It is worthwile noting that the relative amounts are in mass-% on the cement nucleus, that is without considering gypsum (since its amount in cement can vary according to purity and optimum gypsum content). Then, for each type of cement the industrial composition shall be calculated *with* the actual gypsum content.

Mechanical, physical and chemical requirements are listed in Table 6 and 7. A significant difference in the strength classification is the subdivision of every main strength class into two subclasses, to differentiate rapid hardening cement from those with normal strength development. This makes the theoretical number of producible cements as high as 150 (25 types x 6 classes)!

Additionally, the upper limit for 28 day strength is also set. This is somewhat a marketing besides technical requirement, it has the advantage to force all producers to keep cement strengths (and quality) within a certain interval and at the same time protects against unfair competition. The drawback is, for some blended cements (high slag content, mainly), the serious difficulty to comply with both the minimum 2-day and the maximum 28-day strength limit, that is often exceeded.

The last chapter introduces a new criterion to evaluate conformity of cement, based on a continuous statistical control (autocontrol).

Cements must fulfill

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- conformity criteria on single absolute values
- statistical conformity criteria (according to variables or attributes) on characteristic limits.

This means that for the main parameters, cement must conform to a minimum (or maximum) limit value, as it was previously provided by the standards, and also to a characteristic limit. All results over a six- or twelve month period are evaluated according to the mean value and standard deviation and calculated to yield a consumer's risk lower than 5% (that is the risk that a defective lot is accepted as conforming the standard) for lower strength limits and 10% for the upper one.

To make it simple, let's consider the conformity for 28 day strength. According to ENV 197-1, the following equation shall apply

$x - k_A s \geq L$ (lower limit) as well as $x + k_A s \geq U$ (upper limit)

where L is the lower (resp. upper) strength limit at 28 days, s is the calculated standard deviation, k_A is the acceptability constant (depending on the number of tested samples).

Now if a cement of mean 28 day strength = 38.0 MPa is produced with s = 2.0 and tested over 110 samples/year (k_A = 1.93 for L and 1.53 for U), the conformity equation for a 32.5 MPa characteristic value will yield a result of

$x - k_A s = 38.0 - 2.0 \times 1.93 = 34.1$ MPa, which is higher than 32.5 as well as

$x + k_A s = 41.1$ MPa which is lower than 52.5 then the cement is conforming the standard. But a value of s = 3.2 will yield

$x - k_A s = 38.0 - 3.2 \times 1.93 = 31.8$ which is lower than 32.5; in this case the cement does not fulfill the standard specification set for minimum late strength.

The following graphs will help to visualize the situation. The first one shows that a "good" cement, with low strength variability (s = 2) fulfill the requirements with an average 28-day strength of 36.4 MPa, while the other refers to a "bad" cement (s = 4) and the necessary average strength to fulfill specs is 40.2 MPa (3.8 MPa higher!).

This approach is quite interesting and innovative, because it comes clear from the proposed example, that a consistent production, with low deviation, can allow for the production of a complying cement with lower mean strength values, that cannot be sufficient if the deviation is higher, thus forcing to raise the average strength to higher and more costly levels.

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Figure 1 **Conformity of a "good" cement (s = 2)**

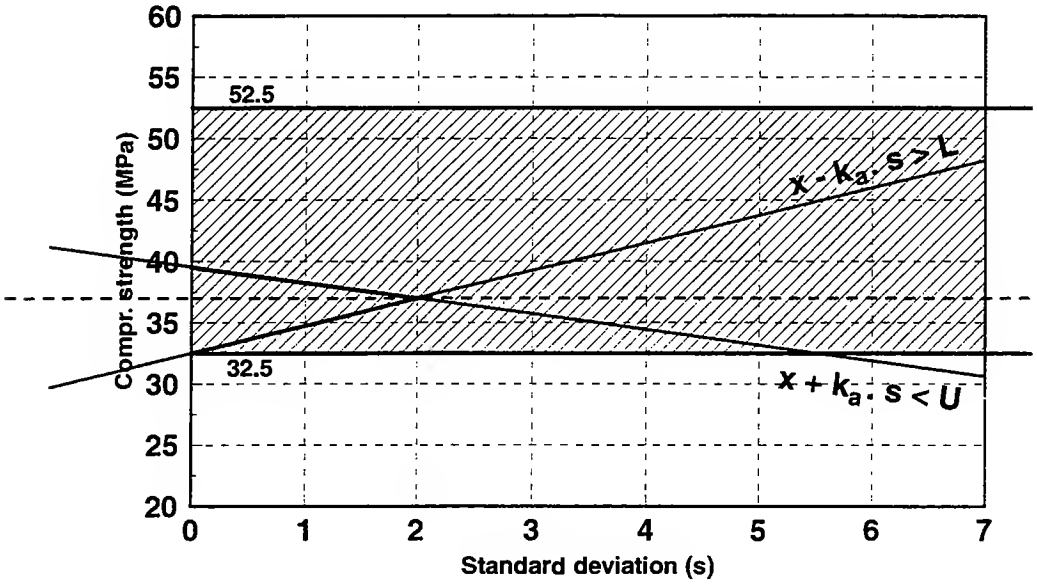

Figure 2 **Conformity of a "bad" cement (s = 4)**

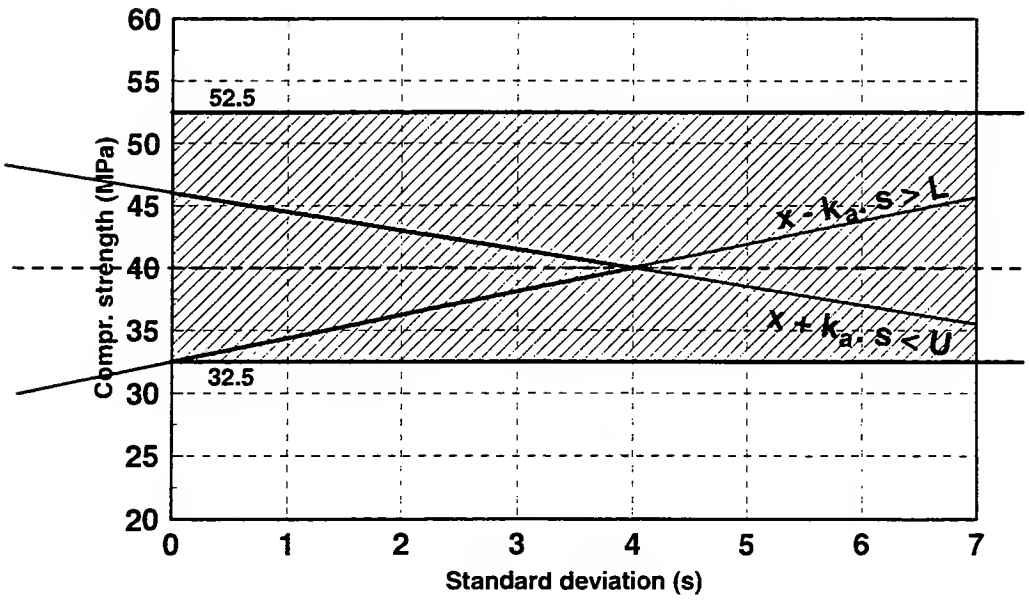

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Table 4 The ENV 197-1 specification on common cements - Cement types and composition

Cement type	Designation	Abbrev.	Clinker	GGBF slag	Micro silica	Natural pozzol.	Industr. pozzol.	Silic. fly ash	Calcar. fly ash	Burnt shale	Limest.	Other constit.
I	Portland Cement	I	95-100	-	-	-	-	-	-	-	-	0-5
II	Portland Slag Cement	II/A-S	80-94	6-20	-	-	-	-	-	-	-	0-5
		II/B-S	65-79	21-35	-	-	-	-	-	-	-	0-5
	Portland MS Cem	II/A-D	90-94	-	6-10	-	-	-	-	-	-	0-5
	Portland Pozzol. Cement	II/A-P	80-94	-	-	6-20	-	-	-	-	-	0-5
		II/B-P	65-79	-	-	21-35	-	-	-	-	-	0-5
		II/A-Q	80-94	-	-	-	6-20	-	-	-	-	0-5
		II/B-Q	65-79	-	-	-	21-35	-	-	-	-	0-5
	Portland Fly Ash Cement	II/A-V	80-94	-	-	-	-	6-20	-	-	-	0-5
		II/B-V	65-79	-	-	-	-	21-35	-	-	-	0-5
		II/A-W	80-94	-	-	-	-	-	6-20	-	-	0-5
		II/B-W	65-79	-	-	-	-	-	21-35	-	-	0-5
	Portland Burnt Shale Cement	II/A-T	80-94	-	-	-	-	-	-	6-20	-	0-5
		II/B-T	65-79	-	-	-	-	-	-	21-35	-	0-5
	Portland Limest. Cement	II/A-L	80-94	-	-	-	-	-	-	-	6-20	0-5
		II/B-L	65-79	-	-	-	-	-	-	-	21-35	0-5
	Portland Compos Cement	II/A-M	80-94	<---------- 6-20 ---------->								
		II/B-M	65-79	<---------- 21-35 ---------->								
III	Slag Cement	III/A	35-64	36-65	-	-	-	-	-	-	-	0-5
		III/B	20-34	66-80	-	-	-	-	-	-	-	0-5
		III/C	5-19	81-95	-	-	-	-	-	-	-	0-5
IV	Pozzolanic Cement	IV/A	65-89	-	<---------- 11-35 ---------->				-	-	-	0-5
		IV/B	45-64	-	<---------- 36-55 ---------->				-	-	-	0-5
V	Composite Cement	V/A	40-64	18-30	-	<--------- 18-30 --------->			-	-	-	0-5
		V/B	20-39	31-50	-	<--------- 31-50 --------->			-	-	-	0-5

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Table 5 ENV 197-1 - Requirements for cement constituents

Type of constituent	Property	Conformity Limit
Clinker	sum of silicates $C_3S + C_2S$	> 2/3 of total
	ratio CaO / SiO_2	> 2.0 %
	MgO content	< 5.0 %
Blast Furnace Slag	glass content	> 2/3 of total
	$CaO + MgO + SiO_2$	> 2/3 of total
	$CaO + MgO / SiO_2$	> 1.0
Fly ash	loss on ignition	< 5.0 %
Siliceous fly ash	reactive CaO	< 5.0 %
	reactive SiO_2	> 25 %
Calcareous fly ash	reactive SiO_2	> 5 %
	28 d strength of ground material	> 10 MPa
	expansion on a 30% blend	< 10 mm
Pozzolans	reactive SiO_2	> 25 %
Burnt shale	28 d strength of ground material	> 25 MPa
	expansion on a 30% blend	< 10 mm
Limestone	$CaCO_3$ content	> 75 %
	clay content (methilene blue test)	< 1.2 %
	total organic carbon (TOC)	< 0.2 %

Table 6 ENV 197-1 - Mechanical and Physical Requirements

Strength class	Compressive strength (N/mm^2 - MPa)				Initial setting time(min)	Expans. (mm)
	Early strength		Standard strength			
	2 days	7 days	28 days			
32,5	-	> 16	> 32,5	< 52,5	> 60	< 10
32,5 R	> 10	-				
42,5	> 10	-	> 42,5	< 62,5		
42,5 R	> 20	-				
52,5	> 20	-	> 52,5	-	> 45	
52,5 R	> 30	-				

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Table 7 ENV 197-1 - Chemical Requirements

Property	Test refer.	Cement type	Strength class	Requirement
Loss on ignit.	EN 196-2	CEM I CEM III	All	< 5.0 %
Insol. residue	EN 196-2	CEM I CEM III	All	< 5.0 %
Sulfates SO_3	EN 196-2	CEM I CEM II	32,5 - 32,5 R 42,5	< 3.5 %
		CEM IV CEM V	42,5 R 52,5 - 52,5 R	< 4.0 %
		CEM III	All	
Chlorides	EN 196-21	All	All	< 0.10 %
Pozzolanicity	EN 196-5	CEM IV	All	positive test

After drafting for common cements, CEN is also preparing new specifications for "special cements", such as sulfate resisting and low heat cements. The drafts are in an advanced stage and are likely to be distributed for approval by member States in 1998.

3.3 THE AUSTRALIAN STANDARD AS 3972-1991

This is a good example of how a standard can be drafted in a simple, clear and effective way.

It consists of two pages only, where reference documents, materials and cements are listed and defined, as well as requirements, properties and dispatching conditions.

All one need to know is contained in this astonishingly essential document.

Cement types are defined as follows

- type **GP** - general purpose Portland cement
- type **GB** - general purpose blended cement

while special purpose cements are

- type **HE** - high early strength cement
- type **LH** - low heat cement
- type **SR** - sulfate resisting cement.

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A blended cement should contain more than 5 % of mineral addition (fly ash, BF slag, or both).

Specified properties are compiled in Table 8.

Table 8 AS 3972-1991 - Portland and blended cements

Type	Property									
	Setting time		Exp.	SO_3	Compr. strength minimum MPa			Heat hydrat. max. J/g		C_3A
	min.	max.	max	max						max
	min	h	mm	%	3 d	7 d	28 d	7 d	28 d	%
GP	45	10	5	3.5	-	25	40	-	-	-
GB	45	10	5	3.5	-	15	30	-	-	-
HE	45	10	5	3.5	20	30	-	-	-	-
LH	45	10	5	3.5	-	10	30	280	320	-
SR	45	10	5	3.5	-	20	30	-	-	5.0

Additional properties such as

- loss on ignition
- fineness or fineness index
- nature and proportion of materials in the cement
- major oxide composition of the cement
- chloride content, if exceeding 0.05 %

can be provided by the manufacturer upon specific request from the purchaser.

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3.4 Conclusion

The importance of standards in our technical life and the present situation for cement have been highlighted. The expected trend for cement standards is represented in table 9.

Table 9 World trends in cement standards

Object	Past/present situation	Trend
Number of standardized cements	Few	Increasing number of common and special cements
Mineral components or additions in cement	Not allowed in OPC, limited use of blended cements	Allowed in OPC, increased number of blended cements
Strength	Compressive and flexural	Only compressive str.
Compr. strength limit	Minimum strength requir.	Min. and max. stength
Age of tests	1, 2, 3, 7, 28 days	2 and 28 days
Standard mortar	constant consistency, constant W/C ratio	constant W/C ratio
W/C ratio	different in each country	0.5
Setting time	initial and final settiong t.	initial setting time

4. STANDARDS FOR MINERAL COMPONENTS

Mineral components or additions have always been used in the production of building materials.

In ancient Greece and Rome blends of burnt lime and pozzolan were used for brickworking and building purposes.

Nowadays the use of mineral additions has improved in quantity and quality.

On one hand the available mineral components are quite extensively evaluated and utilized in cement manufacture, for production, cost-effectiveness and environmental purposes, on the other hand the advancement in technology of high performance concrete, in terms of ultra-high early strengths but also high durability, have benefited by the availability of these materials.

Then the more sophisticated uses of mineral components require adequate standards to describe their properties and performance in cement and concrete.

As a matter of fact, mineral components (and other mineral additions) can be mainly used as

- cement constituents, in the production of blended or Portland modified cements
- raw materials in concrete production, as value added products which impart additional characteristics to the cementitious conglomerate.

Recent standardization has covered both of these aspects, and more is expected in the future.

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Requirements set forth by standards dealing with cement production (see as an example Table 7 in previous chapter) are usually enclosed in the standard itself; they consider aspects of the chemical composition of mineral components and put sometimes also limits with respect to the hydraulic activity, the main responsibility on the effectiveness of the addition being left on the cement producer's shoulders since he has to comply with the final product requirements.

Standards on materials to be used as main concrete constituents are separate documents. They deal about the same parameters as for cement production, but they are mainly focussed on the impact that these characteristics will have on concrete.

Some examples related to mineral components are reported

- **ASTM C 311** "Sampling and Testing Fly Ash or Natural Pozzolans for Use as a Mineral Admixture in Portland-Cement Concrete"
- **EN 450** "Fly Ash for Concrete"
- **ASTM C 989** "Specification for Ground Granulated Blast Furnace Slag for Use in Concrete and Mortar
- **ASTM C 1240** "Specification for Silica Fume for Use in Hydraulic Cement Concrete and Mortar.

5. CONCRETE STANDARDS

5.1 Purpose

The purpose of the concrete standards and specifications is to regulate the relations between the different parties involved in the activity of concrete construction.

Besides the interest to regulate the interaction between the different parties, there is a supreme interest of the society to ensure that the constructions are stable, aesthetic and durable. A typical example are the antiseismic codes, which, although they lead to more costly structures, prevent enormous losses of human lives and goods in case of earthquakes compared to structures built in a normal way.

Similarly to cement standards, the responsibility for the emission of the concrete standards varies a lot from country to country. For instance in Germany and Great Britain, this is the responsibility of the national standard entity (DIN and British Standards respectively), whereas in the United States and Switzerland, it is the task of a professional organisation (ACI (American Concrete Institute) and SIA (Society of Engineers and Architects) respectively). In other countries like Italy, it can also be the responsibility of the government through the Ministry of public works.

5.2 Content of the Standards and Specifications

It will be referred here exclusively to the aspects of concrete technology within the wider field of codes, standards and specifications, with a special emphasis on the corresponding prescriptions in use in the United States (ASTM, ACI) and in Europe (ENV 206).

5.2.1 Quality of the Concrete Components

In general, the concrete standards make reference to the respective standards for each of the ingredients (cement, aggregates, water, mineral and chemical admixtures). That is to say it is simply indicated that the components must satisfy the quality requirements established in the mentioned standards. The relevant ASTM and EN standards for the concrete components cement, admixtures and aggregates are listed in the annex.

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5.2.2 Quality of Fresh Concrete

The following properties of the fresh concrete are specified:

- consistency/workability (always)
- air content (when air is intentionally entrained)
- temperature (in extreme climates)
- density (for light weight or heavy concrete)
- stiffening rate (in extreme climates or for slip forming)

In general, the constructor specifies the consistency of the concrete adequate for the structural element to be concreted and for the available means of placing and compacting. The concrete producer has to deliver a material which complies with this consistency within a certain tolerance. For instance the ASTM standard C 94 for ready-mix concrete prescribes:

Specified Slump	Tolerance
< 50 mm	± 15 mm
50-100 mm	± 25 mm
> 100 mm	± 40 mm

With respect to the content of intentionally entrained air, the ACI recommends percentages which depend on the severity of exposure of the concrete and the maximum aggregate size. A higher severity and a lower maximum aggregate size correspond to higher percentages of air. The ASTM standard C 94 establishes a tolerance of ± 1.5% for the specified air content.

5.2.3 Strength of Hardened Concrete

The strength is the most important characteristic to be specified for the hardened concrete. In the past, it was common to specify the mean strength of the concrete, so that, if the average of the results of the specimens was equal or greater than the specified values, the strength requirement was considered as fulfilled. Based on this criterion, the following two series were equivalent, since both presented the same average (30 MPa):

Series "A": 32, 28, 34, 29, 28, 30, 32, 28, 29, 30

Series "B": 40, 27, 48, 30, 15, 30, 36, 20, 28, 26

At simple view, it results that the series "A" is better than "B", because its values are more uniform (s = 2 MPa against s = 9 of the series "B"). That is to say, although both comply with the specified mean strength, a structure built with concrete "A" will be safer from the structural point of view than if it is built with concrete "B", as the latter presents a strength result which is only the half of the specified value.

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As already mentioned for cement, standards and specifications take into account the variability of concrete and apply the criterion of fractiles to specify the concrete strength. The specified strength corresponds to a fractile related to a certain probability, which means that the designer of the structure accepts that a certain percentage of the produced concrete is "defective". The allowed percentage of "defectives" varies according to the country (it has to be mentioned that the percentage has not necessarily to be related to the safety of the structures):

Percentage p	Country
2 %	Switzerland
5 %	Europe
10 %	United States
20 %	United States (massive concrete)

Assuming that the distribution of strength values follows the law of Gauss, the fractile x_P is calculated as

$$x_P = x_M - z_P \cdot s$$

where z_P is a function of the percentage of "defectives" p. We can see that the mean x_M as well as the standard deviation s participate in this criterion, which explains its universal acceptation.

This principle has now to be applied for the design of the concrete mixes. If we have specified a certain strength that we will denote f'_C (which corresponds to the fractile x_P for a certain percentage of "defectives"), we have to design our mix with a certain margin taking into account the expected variability s_E in the production process. That is to say we have to design the mix for a certain mean design strength f'_D which will be equal to:

$$f'_D = f'_C + \text{margin} = f'_C + z_P \cdot s_E$$

It is evident that the producer which elaborates the concrete in a more uniform manner (lower s_E) will be able to design his mix for a lower strength and therefore more economically.

5.2.4 Durability of Concrete

The traditional criterion to guarantee the durability of the structures against attack of the environment has been the establishment of limits with respect to concrete composition. For instance ENV 206 and ACI code 318 for reinforced concrete establish limits regarding the w/c-ratio of the concrete in function of the exposure condition to which the structure will be submitted, trying to control concrete permeability to improve its durability against aggressive agents.

The fundamental problem of such type of specification is that it is often omitted at the moment of ordering the concrete and anyway its interpretation is unclear (are they mean values, absolute maximums, fractiles?). Moreover, its compliance is very difficult to verify in practice (how is the w/c-ratio of a concrete determined?). The same criticism can be made with respect to the project of the European Standards, where, apart from the maximum w/c-ratio, also a minimum cement content is specified.

A more practical criteria is the one established in the Australian Standards, where, instead of fixing limits for the concrete composition, a minimum strength is specified according to the type of exposure to which the structure will be submitted. The advantage of this criterion is that it is more easy to interpret and to verify its compliance in practice.

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5.3 The European Standard ENV 206

Most discussions on concrete standards are concentrated on the issue of concrete durability, and in this context the specifications on cement content, water cement ratio, chloride content and alkali aggregate reaction.

The durability aspects are here discussed in connection with the new ENV 206 draft „Concrete. Performance, production, placing and compliance criteria", which was issued as a prestandard in 1989. The development of this standard has a similar history to the EN cement standard; the CEN Committee TC 94 started work in 1981, and the present draft is far from being the final version. Nevertheless, it is a good review of the present state of knowledge on the topic, comprising a consensus of expert opinions in Europe.

ENV 206 should cover concrete in general; for all types of concrete - site, precast and ready mixed concrete, plain, reinforced and prestressed concrete. For special purposes, however, additional specifications may be needed, and for these reference is made to the individual national standards. The standard also makes reference to other European standards as regards types of cements, aggregates, mineral and chemical admixtures.

5.3.1 Cement Content and Durability

The minimum cement content in steel reinforced concrete is specified according to environmental conditions and type of concrete structure; it varies between 260 to 300 kg/m^3, as shown in tables 10 and 11.

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Table 10 Exposure Classes related to Environmental Conditions

Exposure Class			Examples of Environmental Conditions
1	Dry environment		interior of dwellings or offices [1]
2	Humid environment	a without frost	- interior of buildings where humidity is high (e.g. laundries) - exterior components - components in non-aggressive soil and/or water
		b with frost	- exterior components exposed to frost - components in non-aggressive soil and/or water and exposed to frost - interior components where the humidity is high and exposed to frost
3	Humid environment with frost and de-icing agents		interior and exterior components exposed to frost and de-icing agents
4	Seawater environment	a without frost	- components completely or partially submerged in seawater or in the splash zone - components in saturated salt air (coastal area)
		b with frost	- components partially submerged in seawater or in the splash zone and exposed to frost - components in saturated salt air and exposed to frost
The following classes may occur alone or in combination with the above classes:			
5	Aggressive chemical environment [2]	a	- slightly aggressive chemical environment (gas, liquid or solid) - aggressive industrial atmosphere
		b	moderately aggressive chemical environment (gas liquid or solid)
		c	highly aggressive chemical environment (gas, liquid or solid)

1) This exposure class is valid only as long as during construction the structure or some of its components is not exposed to more severe conditions over a prolonged period of time

2) Chemically aggressive environments are classified in ISO 9690. The following equivalent exposure conditions may be used:
   * Exposure class 5a: ISO classification A1G, A1L, A1S
   * Exposure class 5b: ISO classification A2G, A2L, A2S
   * Exposure class 5c: ISO classification A3G, A3L, A3S

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Table 11 Durability Requirements related to Environmental Exposure

Requirements	Exposure Class according to Table 2								
	1	2a	2b	3	4a	4b	5a	5b	5c[1]
Max. w/c ratio for [2]									
- plain concrete	-	0.70							
- reinforced concrete	0.65	0.60	0.55	0.50	0.55	0.50	0.55	0.50	0.45
- prestressed concrete	0.60	0.60							
Min. cement content [2] in kg/m^3 for									
- plain concrete	150	200	200				200		
- reinforced concrete	260	280	280	300	300	300	280	300	300
- prestressed concrete	300	300	300				300		
Min. air content of fresh concrete in % for nominal max aggregate size of [3]			[4]	[4]		[4]			
- 32 mm	-	-	4	4	-	4	-	-	-
- 16 mm	-	-	5	5	-	5	-	-	-
- 8 mm	-	-	6	6	-	6	-	-	-
Frost resistant aggregates [6]	-	-	yes	yes	-	yes	-	-	-
Impermeable concrete according to clause 7.3.1.5	-	-	yes	yes	yes	yes	yes	yes	yes
Types of cement for plain and reinforced concrete according to EN 197							sulfate resisting cement [5] for sulfate contents > 500 mg/kg in water > 3000 mg/kg in soil		
	These values of w/c ratio and cement content are based on cement where there is long experience in many countries. However at the time of drafting this pre-standard experience with some conditions in some countries. Therefore during the life of this pre-standard, particularly for exposure classes 2b, 3, 4b the choice of the type of cement and its composition should follow the national standard or regulations valid in the place of use of the concrete. Alternatively the suitability for use of the cements may be proved by testing the concrete under the intended conditions of use. Additionally cement CEI may be used generally for prestressed concrete. Other types of cement may be used if experience with these types is available and the application is allowed by the national standards or regulations valid in the place of use of the concrete.								

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1) In addition, the concrete shall be protected against direct contact with the aggressive media by coatings unless for particular cases such protection is considered unnecessary.
2) For minimum cement content and maximum water/cement ratio laid down in this standard only cement listed in clause 4.1 shall be taken into account. When pozzolanic or latent hydraulic additions are added to the mix, national standards or regulations, valid in the place of use of the concrete, may state if and how the minimum or maximum values respectively are allowed to be modified.
3) With a spacing factor of the entrained air void system < 0.20 mm measured on the hardened concrete.
4) In cases where the degree of saturation is high for prolonged periods of time. Other values or measures may apply if the concrete is tested and documented to have adequate frost resistance according to the national standards or regulations valid in the place of use of the concrete.
5) The sulphate resistance of the cement shall be judged on the basis of national standards or regulations valid in the place of use of the concrete.
6) Assessed against the national standards or regulations valid in the place of use of the concrete.

These requirements reflect the generally accepted opinion that the durability of steel reinforced concrete is governed mainly by its porosity and thus strongly affected by the w/c ratio and cement content.

An important feature of the standard is the differentiation of the cement content according to the severity of its exposure. This allows the concrete producer to use their skills and knowledge in order to design concrete mixes more economically. A prerequisite of this differentiation is, of course, that information on the different conditions to which the specific concrete job is subjected is available.

The prescription of cement content and w/c ratio has some problems: It is difficult to determine both of them in fresh and hardened concrete for control purposes. Furthermore, concrete durability depends not only on cement content as a w/c ratio, it is affected as well by the care taken in its transport, placing, compacting and curing. Therefore, it is indispensable that the quality of concrete in the construction after placing and curing is tested. We know that the porosity of a concrete governs nearly all aspects of durability. Therefore, investigations are under way to introduce a test method enabling the measurement of the porosity of the concrete from specimens taken from the hardened concrete structure. It consists of measuring the amount of gas which flows through the pore system of the concrete specimen in a unity of time. By so measuring the porosity, not only is care taken by the concrete producer, but also that of the contractor placing, compacting and curing the concrete is checked.

An alternative way of testing the porosity and additional aspects of the concrete micro-structure is the microscopical examination of very small specimens. It is very useful for a qualitative inspection of concrete quality, however, it seems difficult to apply this method for a quantitative assessment in the framework of a standard.

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5.3.2 Chloride Content

The maximum chloride content in concrete permitted in plain, reinforced and prestressed concrete is shown in table 12.

Table 12 Maximum Chloride Content of Concrete

Concrete	C1- by Mass of Cement
Plain concrete	1%
Reinforced concrete	0.4%
Prestressed concrete	0.2%

The relatively high content of chloride permitted in steel reinforced concrete, reflects the results of recent investigation, which show that no corrosion of steel is expected at a chloride content of 0.4% by mass of cement. Similar soft limits are prescribed in the ACI standards as shown in table 13.

Table 13 Chloride Content in Concrete, ACI 318-83

Maximum water-soluble chloride ion concentrations in hardened concrete at an age of 28 days contributed from the ingredients including water, aggregates, cementitious materials and admixtures	
Type of Member	**Limit (by mass of concrete)**
Prestressed concrete	0.06
Reinforced concrete exposed to chloride in service	0.15
Reinforced concrete that will be dry or protected from moisture in service	1.00
Other reinforced concrete construction	0.30

5.3.3 Alkali Aggregate Reaction

To prevent deterioration of concrete due to alkali aggregate reaction, it is recommended to apply one or more of the following measures:

- Limit the total alkali content of the concrete mix
- Use a cement with a low effective alkali content
- Change the aggregates
- Limit the degree of saturation of the concrete e.g. by impermeable membranes

Unfortunately no mention is made of the effective measures of using blended cements or mineral admixtures.

5.3.4 Maintenance

Finally, the author would like to point out one serious deficiency of concrete standards; it is the lack of specifications on maintenance of concrete. This is becoming recognized as a gap and attempts are being made to bridge it. Of course this means turning away from the opinion the cement producers have frequently promoted in the past, that concrete need no maintenance whatsoever.

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5.4 Referenced documents

5.4.1 Relevant ASTM Standards for Concrete Components

5.4.1.1 Cement

C 150 Specification for Portland Cement

C 595 Specification for Blended Hydraulic Cements

C 1157 Performance Specifications for blended Hydraulic Cements

5.4.1.2 Admixtures

C 260 Specification for Air-Entraining Admixtures for Concrete

C 494 Specification for Chemical Admixtures for Concrete

C 618 Specification for Fly Ash and raw or Calcined Natural Pozzolan for Use as Mineral Admixture in Portland Cement Concrete

5.4.1.3 Aggregate

C 33 Specification for Concrete Aggregates

C 330 Specification for Lightweight Aggregates for Structural Concrete

C 331 Specification for Lightweight Aggregates for Concrete Masonry Units

C 332 Specification for Lightweight Aggregates for Insulating Concrete

C 637 Specification for Aggregates for Radiation-Shielding Concrete

5.4.2 Relevant EN Standards for Concrete Components

5.4.2.1 Cement

ENV 197-1 Common Cement

ENV 197-X Low-heat Cement

ENV 197-Y Sulfate resisting Cement

5.4.2.2 Admixtures

EN 480 Admixtures for Concrete, Mortar and Grout. Test Methods

EN 934 Admixtures for Concrete, Mortar and Grout

5.4.2.3 Aggregates

EN 932 Tests for General Properties of Aggregates

EN 933 Tests for Geometrical Properties of Aggregates

EN 1097 Tests for Mechanical and Physical Properties of Aggregates

EN 1367 Tests for Thermal and Weathering Properties of Aggregates

EN 1744 Tests for Chemical Properties of Aggregates

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Chapter 6

Concrete as a Building Material

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CONCRETE AS A BUILDING MATERIAL

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1. FUNDAMENTALS OF CONCRETE

1.1 Definition of Concrete

Concrete is basically a mixture of two parts: mineral aggregates and cement paste.

The paste (cement + water) binds the aggregates (sand + gravel or crushed rock) forming a rocklike mass after hardening as a result of the chemical reaction of cement and water (called hydration).

In a properly proportioned, well mixed and placed concrete, each aggregate particle is completely coated with paste. In a good concrete the paste not only coats the surface of the particles, but completely fills the space between the aggregate particles.

Sometimes, materials other than cement, water and aggregates are added (in small quantities) to the concrete during mixing in order to modify one or several properties in the fresh or hardened state. These materials are called admixtures.

The characteristics, performance and serviceability of concrete are a function of the physical and chemical properties of the ingredients, the manner in which they are proportioned and how they interact with each other. Properties of each of the mixture ingredients are therefore significant to the quality of the finished product.

1.2 Constituents

1.2.1 Cement

Cement is a „hydraulic" binder because it reacts with water in air or under water to form a hard, strong mass. By this reaction heat is evolved. A number of calcium silicates, calcium aluminates and related compounds are able to react in this way, and are the active components of commercial hydraulic cements.

1.2.2 Aggregates

Aggregates are generally divided into two groups: fine and coarse. Fine aggregates consist of natural or manufactured sand with particle sizes up to 5 mm. Coarse aggregates are those with particles larger than 5 mm. The maximum admissible grain size of the coarse aggregate in concrete depends usually upon the building process and design of the structural parts of the concrete construction. In most countries the maximum grain size of the aggregate for reinforced concrete ranges from 25 to 40 mm.

Since aggregates make up about 60 to 75% of the concrete volume, their selection is important. Aggregates should consist of particles with adequate strength and resistance to exposure conditions and should not contain materials that will cause deterioration of the concrete. A continuous gradation of particle sizes is desired for efficient use of the cement paste. In this text, it is assumed that suitable aggregates are being used.

A concrete that contains only fine aggregates (sand) with a particle size of up to 5 mm is called mortar.

1.2.3 Water

Water has two functions in the concrete mixture: first, it is essential for the hydration process of the cement; second, it converts cement into a paste and thus acts as lubricant making the concrete workable.

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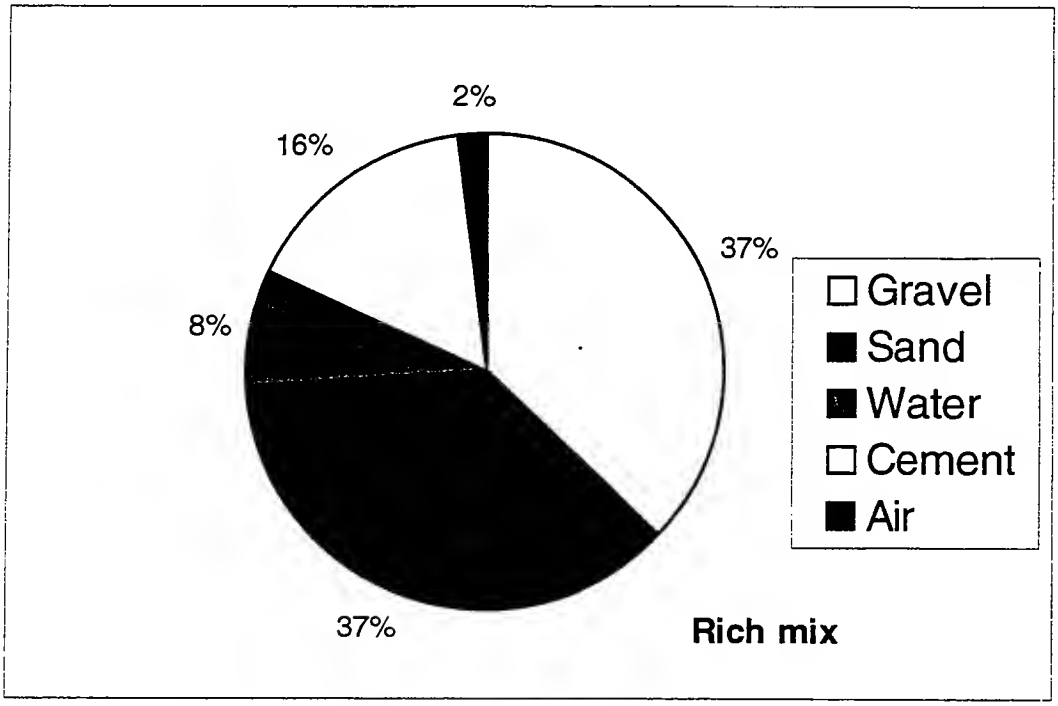
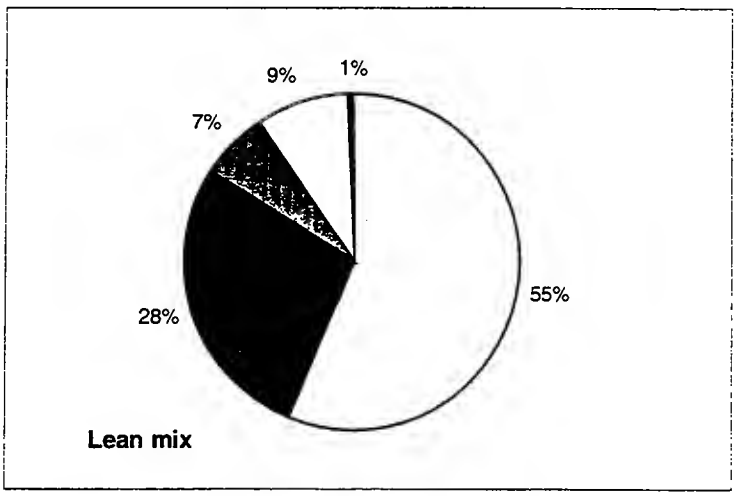
1.2.4 Cement Paste

Cement paste (the mix of cement and water) ordinarily constitutes about 15 to 27% of the total weight of concrete (Fig. 1). The quality of the concrete depends to a great extent upon the quality of the paste.

Fig. 1 Proportions of Concrete Constituent Materials (by weight)





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1.3 Proportions

The proportions of the constituents in concrete can be varied in a certain range. A mix with a cement content below 10% is called lean mix (see Fig. 1 above).

1.3.1 Water/Cement Ratio

Most desirable properties of fresh and hardened concrete are influenced by the water/cement ratio, i.e. the ratio of the amount of water to the amount of cement used in the mix. This relationship is expressed in terms of the ratio of the weight of water (w) to the weight of cement (c):

$$\frac{w\ (kg)}{c\ (kg)} = w/c \text{ ratio}$$

The less water is used (lower w/c ratio), the better the quality of the hardened concrete - provided that it can be mixed and compacted properly. Some of the advantages of reducing the w/c ratio are:

- Increased compressive and flexural strength
- Increased watertightness, lower porosity
- Better bond between concrete and reinforcement
- Less volume change due to wetting and drying (drying shrinkage)
- Increased resistance to weathering (better durability)
- Lower water absorption capacity

2. PROPERTIES OF CONCRETE

The properties of concrete depend very much on the concrete age. Roughly three stages can be distinguished:

fresh concrete
workable mass
↓
intermediate stages (sometimes called „green" and „young" concrete)
↓
hardened concrete
artificial stone which has reached the properties required for a specific construction

The transition from fresh to hardened concrete is a continuous process during which the hydration of cement paste takes place.

Probably the five most important potential properties of concrete are:

- Workability
- Durability
- Resistance to compressive stress (compressive strength)
- Ability to protect reinforced steel against corrosion
- Versatility

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The first property, the workability, allows the fresh concrete to be placed and compacted into forms of any reasonable shape; the durability, ensures a long life for the hardened mass, the third, fourth and fifth property form the basis of modern designs combining the compressive strength of concrete with the tensile strength of steel.

2.1 Properties of Fresh Concrete

2.1.1 Workability and Consistency

The term workability describes the ease of handling, placing and finishing fresh or „plastic" concrete. Although subjective by definition, workability has a profound effect on placement, pumpability, segregation and bleeding of the concrete mixture and, therefore, directly influences economy, strength and durability of the hardened concrete. Workability encompasses a large number of fresh concrete properties. Usually, only one, namely consistency, is measured and the results are used as indication of the workability.

Consistency expresses the ability of the fresh concrete to flow. The mix must be of such a consistency that it will e.g. flow and fill spaces around reinforcing steel or can be pumped. If concrete has the right consistency, it will not crumble or become too stiff to be workable but will flow firmly. For uniform consistency and good workability, the concrete should be as homogeneous as possible.

The use of suitable chemical admixtures called plasticizers or water reducers can help modify the flow properties of concrete and lead to improved plasticity and flowability at the same W/C ratio or to reduced W/C ratio to obtain the same workability.

2.1.2 Consolidation

Vibration sets into motion the particles in the fresh concrete, reducing friction between them, and giving the mixture the mobile qualities of a thick fluid. Mechanical vibration permits use of a stiff mixture with a lower water content.

2.1.3 Unit Weight

Concrete that is normally used in pavements, buildings, and other structures has a unit weight in the range of 2200 to 2400 kg/m^3. The unit weight of concrete varies, depending on the relative density of the aggregate, the amount of air that is entrapped or purposely entrained, and the water and cement contents.

2.2 Properties of Hardened Concrete

2.2.1 Strength

Compressive strength, i.e. resistance to compressive stresses, is certainly a primary characteristic of a good concrete on which the design of most structures (e.g. buildings, bridges, silos, tunnels, etc.) is based. Flexural or beam strength of concrete is essential for highway and airport pavements.

The strength of concrete is most often measured and specified on the basis of compressive strength. Principal factors affecting the compressive (as well as flexural and tensile) strength are type and size of the aggregate as well as the contents of cement and water in the mixture. The strength generally increases as the water/cement ratio is decreased. With time the strength continues to increase as long as unhydrated cement is present, provided that the concrete remains saturated with water.

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2.2.2 Volume Changes

Hardened concrete changes volume slightly due to changes in temperature, moisture content and sustained stress. These volume or length changes may range from about 0.01 to 0.08%. Thermal volume changes of hardened concrete are approximately the same as those of steel.

Concrete kept continually moist will expand slightly. When permitted to dry, concrete will shrink. The shrinkage depends upon several factors such as amounts of mixing water and aggregate, properties of the aggregate, size of the specimen, relative humidity and temperature of the environment, method of curing, degree of hydration, and time.

Concrete under stress will deform elastically. Sustained stress will result in additional deformation called creep.

2.2.3 Watertightness

Where concrete is exposed to weather or in water-retaining structures, it should be watertight. To achieve this impermeability, the paste itself must be watertight and the concrete must be free from cracks and honeycombing. The factors which determine the watertightness of concrete are low w/c ratio, length of curing period, and adequate consolidation.

2.2.4 Durability

The durability of concrete is defined as the ability to resist weathering action, chemical attack, abrasion or process of deterioration caused by nature or man. Durable concrete will retain its original form, quality and serviceability if exposed to the environment.

There are different types of aggression to concrete:

- Freezing and thawing (and deicing chemicals)
- Different types of chemical attack (industrial and natural)
- Abrasion and erosion
- Sea water
- Fire
- Low temperatures
- Nuclear radiation
- Natural and man-caused disasters (earthquakes, hurricanes, explosions, etc.)

From the material point of view, the overall durability of concrete is mostly a function of the cement paste and aggregate properties. Both of these materials must be of adequate quality to ensure acceptable performance, particularly if the concrete will be exposed to severe climatic conditions or different chemical attacks.

As in a kitchen, where the quality of the ingredients used in a recipe is of much concern to a cook, but utmost importance is to be given to the ability to prepare the meal as it should to really make it of satisfaction to everybody, also in concrete adequate concreting, curing and structure design are, of course, more important factors than the ingredients themselves. Any high quality material will fail in a poor concrete. The right way to influence durability is to produce well designed, properly mixed, placed and compacted concrete. High strength, low porosity, good watertightness yield excellent durability.

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3. STEPS IN CONCRETING

The potential qualities of concrete will only be realized if the entire concreting process is carried out properly. The manufacture of concrete comprises:

- Selection of materials
- Proportioning of materials (mix design)
- Mixing of concrete
- Transporting and placing of mix
- Compaction in place
- Curing, prevention of drying out
- Removal of formwork
- Quality control and testing

Quality of concrete depends on the following elements:

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Table 1 Elements of Quality Concrete

STRENGTH	ECONOMY
Quality paste	**Effective use of materials**
- proper cement and water content	- large maximum size aggregate
- appropriate cement type	- proper aggregate grading
Quality aggregate	- proper cement content
- sound particle	- accurate yield
- suitable grading and particle shape	- proper slump
Quality processing	**Ease of handling**
- proper proportioning	- workable mix
- efficient mixing	- homogeneous concrete
- sufficient vibration	- adequate vibration
- adequate curing	**Effective operation**
	- dependable equipment
	- effective methods, plant layout and organization
	- automatic control

DURABILITY	
Resistance to weathering	**Resistance to wear-mechanical abrasion**
- freezing and thawing	- wear-resistant aggregate
- wetting and drying	- proper cement and water content
- temperature variations	- high strength
- suitable aggregate (well graded, structurally stable, large max. size)	- adequate curing
- optimum entrained air	- dense concrete
- proper cement and water content	- special surface finish
- adequate consistency	**Resistance to adverse chemical reactions**
- homogeneous concrete, workable mix, thorough mixing, proper handling, sufficient compacting	- proper cement type and content
- adequate curing, favorable temperature, min. loss of moisture	- homogeneous concrete
	- adequate curing
	- stable aggregate
	- suitable admixtures

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4. VERSATILITY OF CONCRETE

Concrete is the most widely used building material in the world today. We find concrete in all kind of industrial, public and residential buildings, in underground structures, in transportation structuressuch as railroads, bridges, pavements and tunnels, in water and sewage treatment systems, power and offshore stations, dams, a.s.o.

The world production of concrete lies around 2 tons/year per every human being. The reasons for the wide usage of concrete are not difficult to find -

- the raw materials are in plentiful supply and available almost everywhere
- energy requirement for production is low
- costs are little if compared to other building materials
- a wide range of production and application technologies is available
- its great versatility covers a wide range of performance requirements.

Concrete competes with all major building materials - timber, steel, aluminium, asphalt, rock, plastic, glass, bricks, ceramics. Will concrete still have such a predominant role in the future? Is this predominance assured for the years to come? In the following Table 1 a comparison between concrete and its main competitive materials is reported.

Table 1 Position of concrete versus competitive materials

Application	Main competitive material	Position of concrete
Buildings, buildings frame	steel, timber	strong
Roads	asphalt, soil	weak
Bridges	steel, timber	strong
Dams	soil	strong
Partition walls	bricks, gypsum	weak
Pipelines	steel, plastics	balanced
Roofs	tiles, steel	weak
Tunnels	steel, natural stone	strong
Skyscrapers	steel	growing
Offshore structures	steel	strong
Foundations	masonry	strong
Sanitary structures	plastics, ceramics	balanced
Hazardous wastes treatment	steel	strong

In the following Table 2 a list of the various types of concrete and the different production methods is compiled.

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Table 2 Types of concrete and production methods

Concrete types	
Plain	Repair mortar
Reinforced	Repair concrete
Prestressed	Flowing fill
Lightweight	Heat resistant
High performance	Chemical resistant
Fibre-reinforced	Coloured
Mass	Architectural
Mortars	

Production methods	
Ready-mixed	Pumped
Precast	Slipforming
Site-mixed	Underwater
Shotcrete	Vacuum
Roller compacted	Autoclaved
Steam cured	

History proves that no single human development remains predominant forever. However, the raw material reserves for making concrete are assured for many years to come and it is unlikely that some other material will take over in the foreseeable medium to long term.

Moreover the full potential of concrete as a structural material has not yet been realised; its obvious drawbacks such as low ductility, low tensile strength and low durability in some harsh environments are a challenge to concrete technologists to find suitable solutions.

Its capacity to absorb recycled materials and solve the problem raised by the concern of excessive CO_2 production will increase his competiotivity against other building materials.

For these and many other reasons, we firmly believe that concrete will keep its predominant role even in the 21st century.

5. CONCRETE TERMINOLOGY

In the following, a list of the most commonly used terms in concrete technology is compiled.

- Fresh concrete - concrete after mixing; it is still workable and can be transported on site and poured into the mould.
- „Green" concrete - concrete which, after being placed and compacted, has set but only hardened to a limited extent.
- „Young" concrete - concrete in an intermediate stage between „green" and hardened concrete; the material properties of young concrete change rapidly (for example, the strength increases exponentially) and are already similar to those of hardened concrete.
- Hardened concrete - artificial stone which after hydration of the cement with water has reached the properties required for a specific type of structure.
- Mortar - a mixture of cement paste and sand (maximum grain size 5 mm).
- Bleeding - autogenous flow of mixing water within, or its discharge from, newly placed concrete or mortar caused by the settling of solid materials in the mass.
- Curing - maintaining moisture and temperature in „green" and „young" concrete during a specific period to assure satisfactory hydration and proper hardening of concrete.
- Water/cement ratio - the ratio of cement to aggregate by weight.
- Normal weight concrete - concrete having a unit weight in the order of 2000 to 2800 kg/m^3 made with aggregates of normal weight.

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- Lightweight concrete - concrete having a substantially lower unit weight (max. 2000 kg/m^3) than that made from gravel or crushed stone.
- Heavyweight concrete - concrete with exceptionally high unit weight (min. 2800 kg/m^3) usually made with heavyweight aggregates (possible application: radiation shields).
- In situ (cast-in-place) concrete - placed directly in the structure.
- Precast concrete - concrete cast in a place (e.g. mould at the factory) other than the structure.
- Ready-mixed concrete - concrete in fresh and unhardened state for delivery to the purchaser.
- Lean concrete - concrete of low cement content.
- Rich concrete - concrete of high cement content.
- Plain concrete - concrete without reinforcement.
- Reinforced concrete - concrete containing steel reinforcement and designed on the assumption that the two materials act together in resisting forces.
- Reinforcement - metal bars, wires or other slender members which are embedded in concrete in such a manner that the metal and the concrete act together in resisting forces.
- Prestressed concrete - concrete into which internal stresses of such magnitude and distribution are introduced that the tensile stresses resulting from the service loads are counteracted to a desired degree.
- Admixture - ingredient of concrete other than water, aggregates and cement, added just before or during mixing.

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7. REVIEWING QUESTIONS

1) What is concrete made of?
2) Explain the functions of the concrete constituents.
3) What is the composition of paste?
4) Explain the water/cement ratio and its influence upon the properties of concrete.
5) Name the most important properties of concrete.
6) Name the three stages of the concrete development.
7) Name the steps in concreting.
8) Explain the difference between concrete and mortar.

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Chapter 7

Concrete Main Components

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CONCRETE MAIN COMPONENTS

by H. Wolter
Raw Materials and Product Technology

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1. MAIN COMPONENTS OF CONCRETE

1.1 Definition of concrete

Concrete is a composite material

that consists of aggregate (gravel + sand), cement and water and frequently admixtures and/ or additives.

The aggregate consists of a conglomerate of generally different large grains, mostly natural rocks. In particular cases also of waste wood, polystyrene or metals (waste steel or other).

As binders hydraulic materials are used, generally ordinary Portland cement. In special cases asphaltic materials or artificial resin will be applied.

Aggregates are generally designated as either fine (sand: 0.025 to app. 5 mm) or coarse (gravel: from app. 5 to 32 mm or larger). A 'concrete' with sand only is named mortar.

The cement paste coats the particles of aggregate and fills the spaces between them. While fresh, the cement paste also provides the lubrication that reduces the friction between the aggregate particles and imparts workability to the fresh mix. When hardened, the paste (now called cement stone) binds together the particles of aggregate.

The following figure shows the

SYSTEM: Binder + Water + Aggregate = Concrete

General Case				
Concrete	=	**Filler**	+	**Binder**
Special Case				
OPC concrete	=	**(Gravel + Sand) aggregate**	+	**OPC paste**
Mortar	=	**Sand**	+	**OPC paste**
OPC paste	=	**OPC**	+	**Water**

1.2 Cement

The most important component of concrete, the cement, has been treated in detail in a separate chapter.

1.3 Aggregates

1.3.1 Aggregates from natural sources

Definition

Natural aggregates are a mixture of uncrushed and/or crushed rocks and minerals. It consists of particles of different (or sometimes approx. equal) sizes and are used with a binder (cement) and water to produce concrete or mortar [1].

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General

The mineralogical, physical and chemical characteristics of rocks are determined mainly by the events of their geological history. A knowledge of the ways in which rocks are formed, and of the various natural processes (whereby their original characteristics are altered) may therefore lead to a better understanding of those intrinsic properties which determine the suitability of a rock as a source of concrete aggregate.

Rock classification according to their formation:

- Igneous rocks
- Sedimentary rocks
- Metamorphic rocks

Igneous rocks

Igneous rocks are those which are formed by the solidification of molten masses, and many of their characteristics are determined by the rate and condition of cooling. Extruded volcanic rocks, i.e. those which have been ejected on the earth's surface, are cooled very rapidly, so that crystallisation of the component minerals is generally only partly effected, and the resulting rock consists of a mixture of crystalline ingredients and glassy matter.

On the other hand rocks of the intruded igneous type, i.e. those which are formed from molten rock which has been intruded into overlying rock masses to form sills (a tabular intrusion of the surrounding rock), laccoliths (a concordant intrusion that has domed the overlying rocks) and dikes (rock solidification as a tabular body in a vertical fissure) etc. have been allowed to cool slowly, so that all the several compounds of the rock mass have had time to become thoroughly crystallised [8, 9].

The igneous rocks are the primary rocks and from these all the other rocks are ultimately derived. The mineral constituents of igneous rocks have been classified as follows:

Table 1: Minerals of igneous rocks

Feldapatic silicates	**Free silica**
Orthoclase	Quartz
Plagioclase	Tridymite
Leucite	Opal
Ferromagnesian silicates	**Accessory minerals**
Pyroxene	Magnetite
Amphibole	Ilmenite
Biotite	Haematite
Olivine	Apatite

Sedimentary rocks

Sedimentary rocks are derived from the chemical or mechanical breakdown of older rocks. The fragments resulting from such disintegration accumulate in deposits, for the most part under water, and the particles may be cemented together by the deposition of other siliceous, calcareous, argillaceous or ferruginous materials to form a dense mass.

Chemical precipitates are crystalline, but by far the greater proportion of sedimentary rocks are made up of fragments of all sizes of earlier rocks. As in the case of sand or gravel beds, the fragments may lie loosely together, or may be firmly cemented as a compact material.

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The fragments of the older rocks may be sharply angular, but as they are often transported by wind or by water or by glacier for considerable distances, attrition may cause them to become smoothly rounded in their passage to the final deposit.

As a result of their mode of deposition, the structure of sedimentary rocks is almost invariably stratified, and is essentially different from that of igneous rocks. Glacial deposits or coral reefs are not stratified and in this respect differs from other rocks of this class.

Due to the geological and climatological changes, sudden radical changes often occur in the nature of the material being deposited. For this reason limestone are frequently interbedded with shale, sandstone with siltstones and quartzite with dolomite [3].

Metamorphic rocks

Metamorphic rocks are those resulting from the alteration in place of pre-existing sedimentary or igneous rocks. Such alteration is brought about by the application of high temperatures and/ or pressures to the rock mass.

A large group of rocks in this class is characterised by foliated structure, which is not to be confused with the stratification of sedimentary rocks nor with the flow banding of some lava, but which is the result of the segregation of one or more of the constituents minerals and their reorientation in form of parallel plates.

Metamorphic rocks vary greatly both in structure (the attitude and relative position of the rock masses of an area) and in texture (the general appearance of a rock, e.g. the size, shape, and arrangement of the constituent elements). Where the rock has been formed under high temperature and pressure, an equigranular texture and a massive internal structure results and lend great strength and toughness to the rock. Such rocks include many hornfelses and quartzites.

1.3.2 Classification

From the petrologic view of point the aggregates, whether crushed or naturally reduced in size, can be divided into several groups of rocks having common characteristics. The group classification does not imply suitability of any aggregate for concrete-making: unsuitable material can be found in any group, although some groups tend to have a better record than others.

Table 2: Classification of natural aggregates according to rock type [4].

Basalt Group	Flint Group	Gabbro Group
Andesite	Chert	Basic Diorite
Basalt	Flint	Basic Gneiss
Basic Porphyrites	Hornstone	Gabbro
Dolerites		Hornblende-rock
Spilite		Norite
		Serpentine
Granite group	**Gritstone group**	**Hornfels group**
Gneiss	Arkose	Contact-altered rocks
Granite	Greywacke	(all kinds except marble)
Granulite	Grit	
Pegmatite	Sandstone	
Syenite	Tuff	

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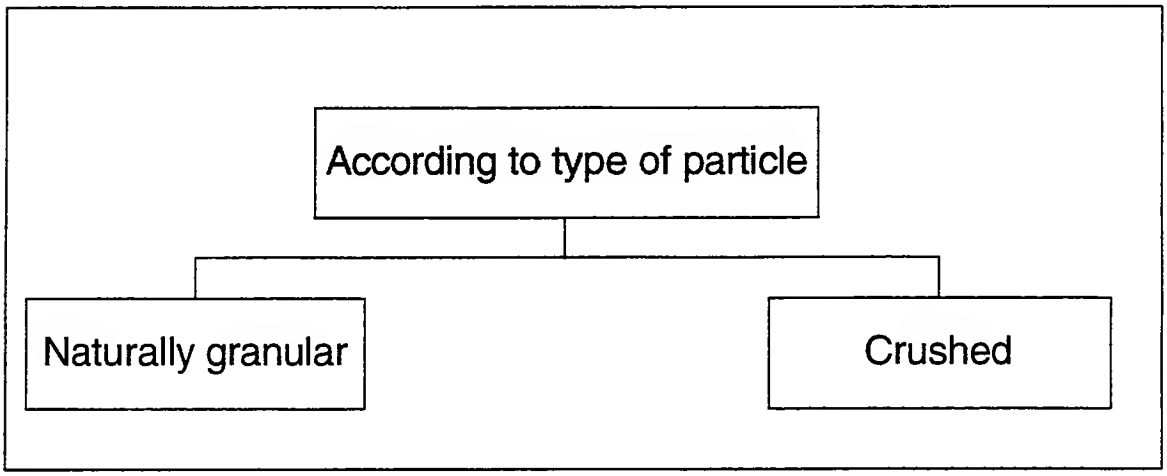
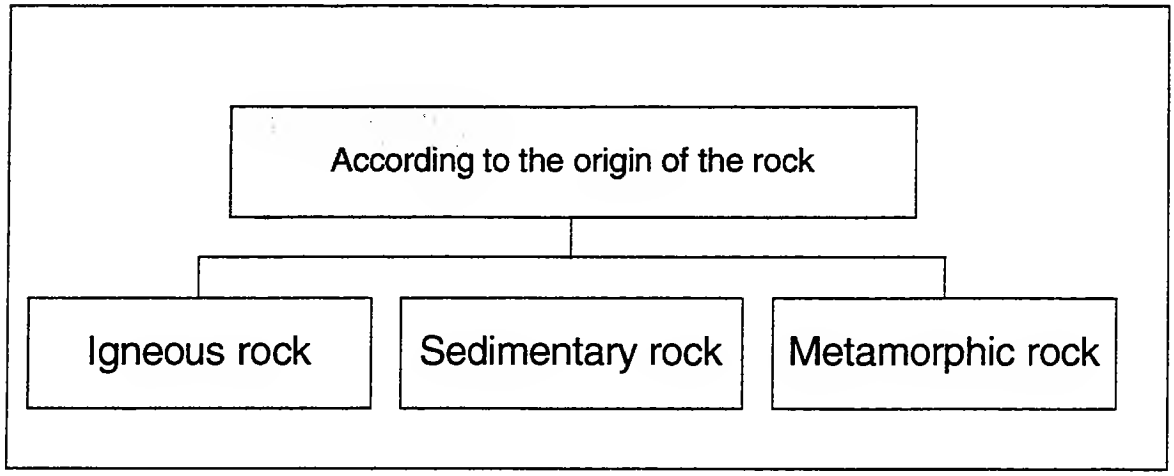
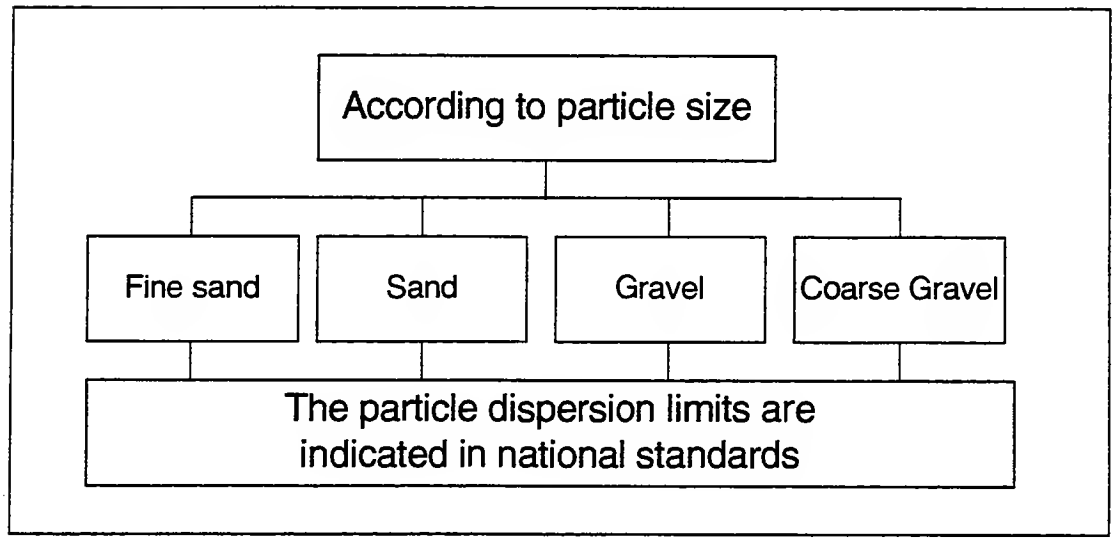
Pophyry Group	Limestone group	Quartzite group
Aplite	Dolomite	Ganister
Felsite	Limestone	Quartzitic sandstone
Granophyre	Marble	Re-crystallized quartzite
Porphyry		
Rhyolite		
Trachyte		

Schist group
Phyllite
Schist
Slate
All severely shared rocks

Figure 1: Classification according other parameters:







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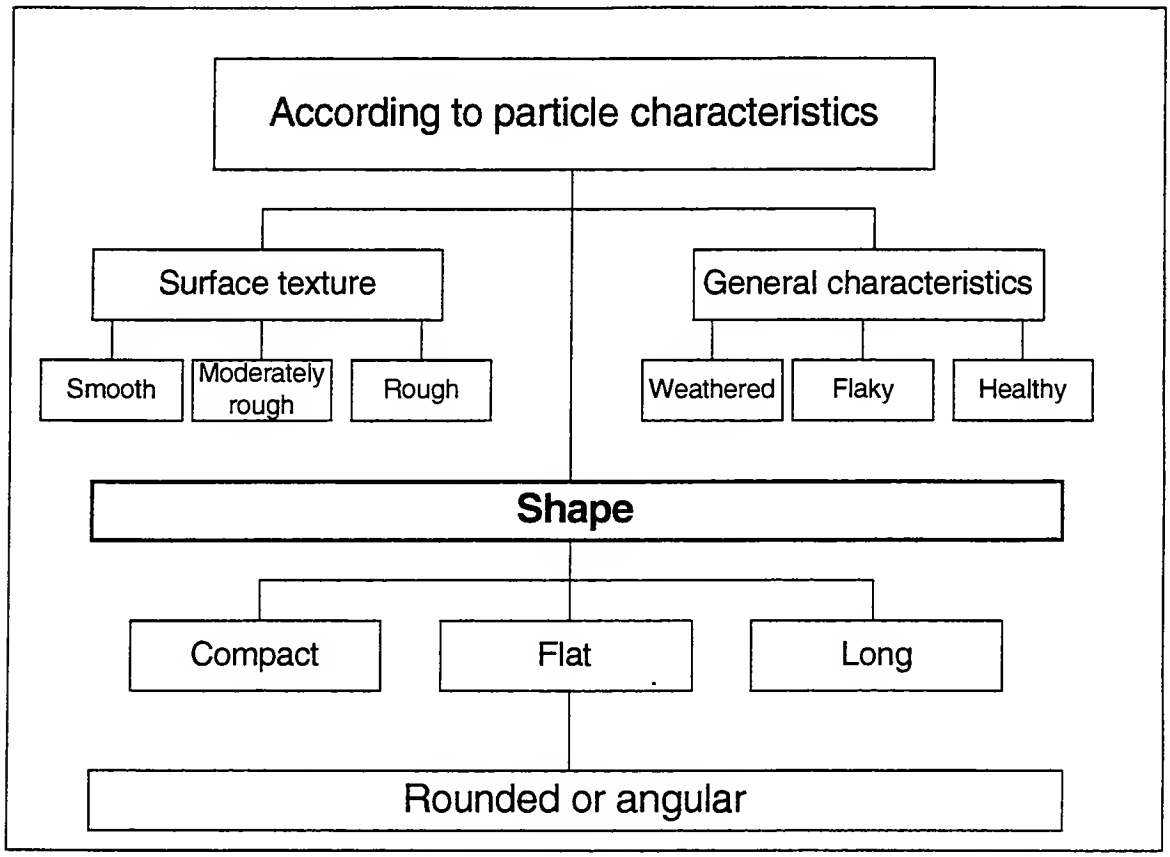

1.3.3 Chemical properties

While it is desirable that the aggregate should as far as possible be chemically inert, many natural aggregates contain substances which are deleterious in concrete. Substances considered chemically deleterious may be broadly classified into five groups:

Group 1:
Substances ***soluble in water***, which may be leached out of the aggregate thereby weakening it or promoting efflorescence in the concrete, e.g. common salt.

Group 2:
Soluble substances, or substances which become ***soluble in the cement matrix***, which may interfere with the normal hydration of the cement, e.g. humid acid.

Group 3:
Substances which ***react with the cement*** destroying its properties, e.g. sodium sulphate.

Group 4:
Substances which may ***react with the alkali*** constituents of the cement, e.g. opal (AAR, Alkali-Aggregate Reaction).

Group 5:
Substances which may ***cause corrosion*** of reinforcing steel, e.g. common salt.

1.3.4 Physical properties

Perhaps the most important concept developed from research is that the aggregate must be studied not only in its relation to the hardened cement paste, i.e. as a component of concrete, but in relation to the environment of the concrete during its service life. With this

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concept in mind, the physical properties of natural stone aggregates may be considered in terms of following factors:

Table 3: Properties of rocks

	Various properties			Strength Properties			
Rock Type	**Relative density [kg/m³]**	**Water Absorption [Vol. %]**	**Compres-sion str. [MPa]**	**10%-Crushing value [kN]**	**Modulus of rupture [MPa]**	**Modulus of elasticity [GPa]**	**Shear strength [MPa]**
Andesite	2,7 - 2,9	0,4 - 0,5	500 - 540	450	--	80 - 110	--
Basalt	2,8 - 3,0	0,1 - 0,3	190 - 470	340	14 - 24	49 - 98	--
Dolerite	2,8 - 3,1	0,1 - 0,7	160 - 375	180 - 340	14 -24	60 - 105	--
Dolomite	2,7 - 2,9	0,1 - 0,3	200 - 380	130 - 180	--	100 - 130	--
Felsite	--	0,2 - 1,5	360 - 450	230 - 380	--	70 - 90	--
Granite	2,6 - 2,8	0,2 - ,5	70 - 325	120 - 220	0 - 20	14 - 70	14 - 30
Hornfels	2,6 - 2,8	--	--	140 - 260	--	--	--
Limestone	2,6 - 2,9	0,2 - 0,6	20 - 240	180	4 - 15	21 - 71	8 - 21
Marble	2,6 - 2,9	0,2 - 0,6	20 - 240	--	4 - 24	48 - 96	9 - 45
Quartzite	2,6 - 2,8	0,2 - 0,6	105 - 480	160 - 280	12 - 24	64 - 86	--
Rhyolite	2,6 - 2,8	--	180 - 530	210 - 290	--	--	--
Sandstone	2,5 - 2,7	0,2 - 0,9	10 - 255	50 - 280	3 - 14	14 - 55	2 - 21
Syenite	2,6 - 2,7	0,4 - 1.5	95 - 445	--	--	55 - 72	--

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Strength

The compressive strength of cubical specimens of natural stone from which acceptable aggregate is derived, normally varies from about 70 to 400 MPa, while individual results as high as 540 MPa have been recorded [5].

With a wide range there appears to be only a poor correlation between the compressive strength of the aggregate and the flexural or compressive strength of the concrete.

As an index of overall quality the "10 % fines aggregate crushing value" or 10 % FACT value of coarse aggregates is useful (BS 812). However, it must be repeated that the usefulness of these tests is confined to a general assessment of quality and to the establishment of acceptance limits. New researches by Davis and Alexander have, however, shown a relationship between the aggregate type and the properties of concrete [6].

Elasticity

The modulus of Elasticity of concrete depends to a considerable degree on that of the aggregate from which it is made. The flexural strength of concrete is also depending on this property of the aggregate, and the use of an aggregate having a high elastic modulus will usually result in a concrete of high flexural strength, other factors being equal [7].

The drying shrinkage of concrete is reduced by the use of aggregate having a high elastic modulus. Low shrinkage, induced by the use of rigid aggregates may be undesirable as the restraint provided by the aggregate increases the cracking tendency of the paste, thereby reducing the durability of the concrete.

It would appear on balance that the reduction in durability resulting from the use of aggregate of high elastic modulus would in most cases more than offset any advantages which may be gained thereby.

Porosity

Many authorities consider that the size, abundance and continuity of pores in a rock particle are its most important physical properties. It is considered that the size and nature of the pores affect the physical strength of the aggregate, and control water absorption and permeability, thereby determining the durability of the aggregate in regard to freezing and thawing, and its resistance to chemical attack.

It has been previously pointed out that the strength of the aggregate over a wide range cannot be considered as an important factor. Furthermore, in tropic countries a high resistance of freezing and thawing is not often an important quality of concrete. This property should, however, be considered where impermeability or high resistance of chemical attack are important features of the construction. Sandstone and shale show an average porosity of 18 to 19 %, some limestone types 8 % and quality rocks 1 to 3 %. Individual results may be much higher [3].

Thermal expansion

In recent years it has come to be appreciated that the deterioration of concrete structures may be significantly affected by differences between the coefficients of thermal expansion of aggregate and the cement matrix in which it is embedded. In particular, the use of an aggregate of very low coefficient of expansion may lead to disintegration of the concrete. As the temperature of concrete, made with such an aggregate is lowered, the cement paste tends to contract more than the aggregate, with the result that tensile stresses are set up in the former, which may result in cracking.

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Bond characteristics

Perhaps one of the most important attributes of a concrete aggregate is its capacity for bonding strongly with cement paste. This factor has a significant influence on the flexural strength [7].

In a wider field the word "surface texture" should be replaced by "bond characteristics". The latter term embraces not only surface texture, but the extent to which cement paste can penetrate into pores are surface depressions, the angularity of planes of fracture, the friable of the surface, and the presence or absence of loosely bonded or friable coatings.

The effect of bond characteristics, while sufficiently significant in the medium strength range of mixes, becomes much more pronounced in the case of high-strength concretes.

Particle shape and surface texture

The shape of aggregate particles in both sand and stone, is one of the most significant factors affecting the behaviour of a concrete mix. Concrete water demand and water requirement on the design of concrete mixes, are strongly influenced by the shape of aggregate. Spherical, cubical or chunky shapes produce concrete having lower water demand than particles that are elongated or flaky.

Surface texture of the grains also affects water demand, although to a lesser extent than shape. Rough textures increase water demand due to greater surface area and to increase friction and mechanical interlock of the particles.

It seems that the shape and surface texture of aggregate influence considerably the strength of concrete. The flexural strength is more affected than the compressive strength Some data of Kaplan's research [7] are reproduced in the following table, but this gives more than an indication of the type of influence.

Table 4: Effect of aggregate properties

Property of concrete	Relative effect of aggregate properties		
	Shape	**Surface Texture**	**Modulus of Elasticity**
Flexural strength	31 %	26 %	43 %
Compressive strength	22 %	44 %	34 %

Thermal properties

Thermal properties which may be of significance are:

- specific heat (or heat capacity)
- thermal conductivity
- thermal diffusivity

All these properties are significant in the design of radiation shielding for nuclear power plants and in estimating loads on air-conditioning plant for buildings.

Resistance to abrasion

This is of significance in the choice of aggregate for pavements, industrial floors, channels, conveying abrasive materials and for certain type of silos.

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Soundness

The sulphate soundness test (ASTM, C 88 and SABS 1083) is not highly significant but provides a rough indication of the durability of an aggregate when subjected to expected environmental conditions. There is no evidence available to suggest that the mechanism of disruption of the test can be directly correlated with performance of an aggregate when subjected to these conditions. For this reason it is important that when an aggregate is to be evaluated on the basis of this test, it should be done by comparison with an aggregate of similar mineralogical composition and geological history and which has proved to be satisfactory in practice.

1.3.5 The grading

The grading of an aggregate refers to the distribution of particles of various sizes, and is determined by passing a representative sample through a series of standard sieves. Concrete aggregates are divided into two categories of fine aggregate or sand and coarse aggregate or stone. The boundary between these two is the 4 mm sieve but a fine aggregate may contain a small proportion of oversized particles and coarse aggregates may have some undersized material and still comply with the definition.

The German standard DIN 1045 sieves for fine and coarse aggregates are:

0.25 - 0.50 - 1.0 - 2.0 - 4.0
8.0 - 16.0 - 31.5 - 63.0 - mm

The sizes of the openings in the consecutive sieves are related by a constant ratio, the clear opening of each sieve size being twice or nearly twice that of the next smaller size.

Fineness modulus

The grading analysis provides data from which the fineness or coarseness of aggregate can be judged. The measure of fineness or coarseness is expressed in terms of an index known as the fineness modulus or FM. This is defined as an empirical factor obtained by adding the total percentages of material retained on each of the standard sieve sizes, except this amount smaller than 0,25 mm, and dividing the sum by 100. An example is given in the following table:

Table 5: Determination of fineness modulus

Sieve size [mm]	Mass retained on sieve [g]	Mass retained on sieve [wt-%]	Total retained on sieve [wt-%]*	Total passing on sieve [wt-%]
31,5	<1,0	0,0	0,0	100,0
16,0	1980,0	38,0	38,0	62,0
8,0	1250,9	24,0	62,0	38,0
4,0	781,9	15,0	77,0	23,0
2,0	469,1	9,0	86,0	14,0
1,0	312,7	6,0	92,0	8,0
0,5	156,4	3,0	95,0	5,0
0,25	156,4	3,0	98,0	2,0
< 0,25	104,2	2,0	(100,0)	0,0
Total	5212,0	100,0	548,0	---
Fineness Modulus **FM**		5,48		

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The FM is an index of the material, that is, a measure of the average particle size based on purely empirical classification. But the index in itself does not describe the grading. The FM indicates whether a material is fine, medium or coarse. A very fine aggregate grading with a maximum grain size of 31,5 mm is one having a modulus of 3,30 or less but in general run up to about 4.0. Medium aggregates are those with FM from 4.0 to 5.0 while coarse aggregates have higher fineness modulus (> 5.5).

Grading of sand

While the grading of a sand has a relatively minor effect on water demand, it has a major influence on the workability, cohesiveness and bleeding properties of concrete in its plastic state. Experience gained over many years has shown that the proportions of sand passing the 0.090, 0.125 and 0.25-mm sieves have the greatest effect on the properties of the mix while the contribution on the remaining fractions is less significant. Real values are shown in the following table:

Table 6: A guide to sand grading

Sieve Size [mm]	Percentage passing	
	Suggested outer limits	Preferred limits *)
4,0	80 - 100	90 - 100
2,8	68 - 100	80 - 100
2,0	55 - 100	75 - 95
1,4	43 - 92	70 - 85
1,0	32 - 85	55 - 70
0,5	16 - 65	40 - 60
0,25	5 - 43	20 - 40
0,125	2 - 22	10 - 20
0,09	0 - 11	3 - 6

*) limits are suggested for pump concrete and for concrete used in sliding formwork

The dramatic increase in surface area as the particle size decrease is illustrated by a few examples. The effect of grading on concrete behaviour is not constant but depends on the cement content and workability of the mix. Generally it may be accepted that a lower cement content calls for a finer grading of the aggregates: in other words, the leaner the mix, the higher should be the proportion of fineness (the total sum of sand < 0,1 mm plus cement should be approx. 350 kg per m^3 concrete).

Table 7: Specific surface of various materials

MATERIAL	Specific Surface [m^2/kg]
32-mm concrete aggregate	0,07
Average grading (B32) of aggregates	2,7
Average concrete sand	3,5
Silt	35
Ordinary Portland cement	300
Kaolinite	5'000

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Blending of sands

The grading of sand has a more pronounced effect on workability than has that of the coarse aggregate. While sand from a single source may not meet the grading requirements, it is often possible to overcome this difficulty by using a combination of sands.

In the following table the grading is tabulated for the mixture of aggregates of 67 % of the crusher sand and 33 % of the pit sand:

Table 8: Grading of mixture of two sands

Sieve size [mm]	Percent passing		
	Crusher Sand A	Pit Sand B	Mixture 67 % A + 33 % B
4,0	82	100	88
2,0	56	100	71
1,0	45	100	63
0,5	33	97	54
0,25	19	58	32
0,125	7	31	15
0,090	3	18	8
FM	**3.58**	**1.14**	**2.77**

A comparison of concrete with crushed aggregates and concrete with rounded aggregates is shown in the following table:

Table 9: Properties of fresh concrete with crushed aggregates

Fresh concrete Crushed aggregate has in comparison with rounded aggregate:
• Better stability of stiff concrete after immediate demoulding • Comparing flowing properties during vibration • Comparable properties of pumped concrete • Less bleeding • Comparable or better concrete surface • Consistency (on DIN A flow table) less favourable

Table 10: Properties of hardened concrete with crushed aggregates

Hardened concrete Crushed aggregate has in comparison with rounded aggregate:
• Better flexural strength • Higher grip resistance • Less depth of water penetration

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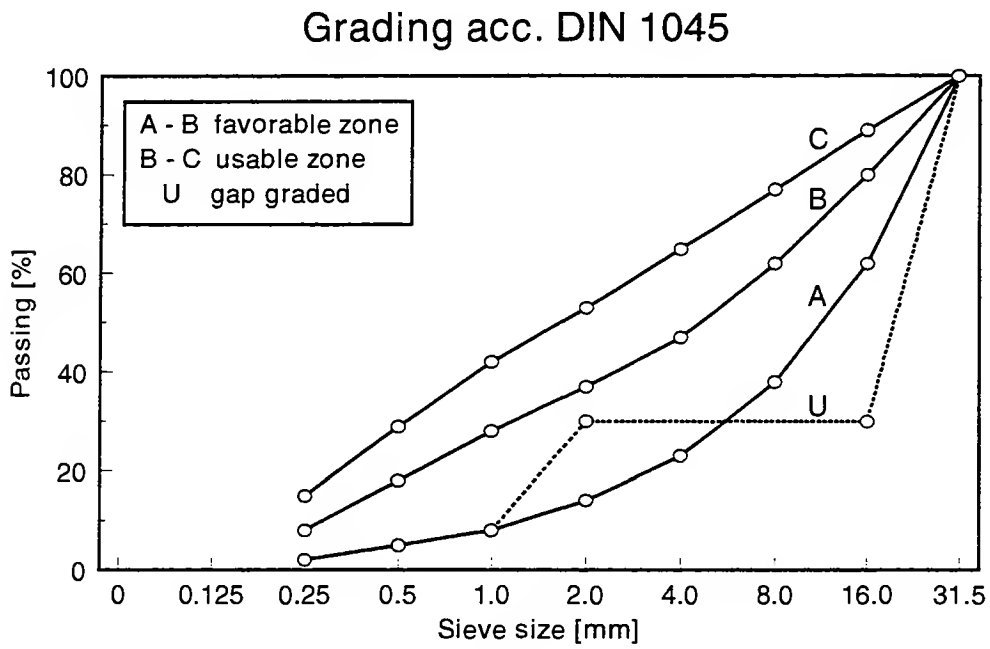
Grading of stone

The grading of coarse material has a smaller influence on the workability of a mix than that of the sand but haphazard grading cannot be permitted where the quality of the concrete is of importance. For this reason the material is screened into its various size fractions and recombined to conform with specified or suitable grading requirements. The following figure shows the grading limits according to DIN 1045.

Figure 2: Examples of particle size distribution for different gradings



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Gap-graded and continuously-graded concrete

The relative merits of concrete made with gap-graded and continuously-graded stone have been much debated [10, 11,12]. The technical advantages of each are listed below but, in comparing merits, it must be stated that economic considerations may outweigh the technical.

Table 11: Properties of gap-graded concrete

Gap-graded concrete [2,12] compared with continuous-graded concrete
• Less danger of particle interference. • Grater sensivity of consistence to change in water content. This makes for more accurate control mixing of water which in turn ensures more consistent strength results. • Stiffer mixes are more responsive to vibration.

Table 12: Properties of continuously-graded concrete

Continuously-graded concrete compared with gap-graded concrete
• Wetter mixes are less prone to segregation. • Less sensitive to slight changes in water content. This is an advantage where uniform workability is important. • Improve pumpability especially at higher pressure. • Improve flexural strength due to the increased surface area of graded stone.

1.3.6 Alternative aggregates

Aggregates from natural sources make up the bulk of the aggregates used in concrete. However, there are a number of alternative materials which can be used as aggregate and concrete technologists should be encouraged to explore their potential performance and economy for use in concrete.

Materials used for a greater or lesser extent of natural aggregate are:

- **Metallurgical slag**
  Metallurgical slags that have been found suitable for use in concrete include blastfurnace, ferromanganese, ferro-silicon-manganite, phosphate, chrome, copper and platinum slag [14].
- **Clinker**
  Well-burnt furnace residues from furnaces fired with pulverised fuel (BS 1165).
- **Expanded Clays** (shale and slate)
  When certain clays or shale are rapidly heated to the point of incipient in fusion, the material softens, becomes plastic and tends to entrap gases which are generated within its mass. Trade names are: *Haydite, Cel Seal, Aglite, Gravelite. Leca and Güllät.*
- **Sintered fly ash**
  In one of the manufacturing processes of low-density aggregate from fly ash, the ash is first palletised and then formed noodles are sintered at 1'000 to 1'200 °C.
- **Burnt-clay bricks**
  Burnt-clay aggregate made by crushing broken bricks is being used more and more to

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some extent in cast-in-situ concrete, but mainly for making precast concrete panels and concrete masonry units.

- **Colliery Spoil**
  In the UK in 1972, two plants were operating on colliery waste and producing an aggregate known as Aglite. It is an expanded low-density aggregate.
- **Exfoliated Vermiculite**
  Vermiculite is the geological name given for a group of micaceous minerals. It has the unique property that when heated to ground 1'200 °C, the flakes expanded (exfoliated) up to 15 times their original volume.
- **Perlite**
  Perlite and certain other volcanic glasses expand when heated to the point of incipient fusion. Expanded perlite is usually produced only in sand size. It has a maximum bulk density of 240 kg/m^3.
- **Glass**
  Waste glass is a potential aggregate for concrete and has been used in precast concrete elements. Waste glass is susceptible to alkali-aggregate-reaction and may produce concrete with a high expansion if used in combination with a high alkali content.
- **Recycled concrete as aggregate**
  Depletion of normal aggregate sources and waste disposal problems have made concrete reclaimed from the demolition of concrete structures an attractive proposition as aggregate. Useful data have been published by Buck [13].
- **Sawdust and Wood Wool**
  Concrete made from mixtures of Portland cement and sawdust is generally a rather unreliable material and its properties cannot be easily predicted.
- **Expanded polystyrene**
  Expanded polystyrene beads have a closed cellular structure and can easily be used to produce cellular concrete. Expanded polystyrene beads are extremely light (density 12 to 16 kg/m^3) so that they can segregate from the mix and are hydrophobic.
- **Remarks**
  Some of the alternative aggregates, such as air-cooled metallurgical slag are used merely because they are more economical than natural aggregates and can be used without detriment. In other cases, certain technical properties are required (e.g. low density, superior thermal insulation) and for these purposes aggregates are often "purpose-made" for example low-density aggregates such as bloated clay and shale, sintered fly ash and exfoliated vermiculite. Many of these materials are only used to make precast concrete products such as concrete masonry units.

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1.4 Admixtures

Admixtures are chemicals which are added in relatively small quantities to the basic constituents of concrete. The quantity of an admixture is usually measured against the quantity of cement, expressed as its percentage by mass.

It is convenient to make a distinction between admixtures and additives. Admixtures are used in quantities of maximum 5 wt-% (usually 0.5 to 2.0%). Additives such as fly ash, or other pozzolanas and blast furnace slag, are applied in quantities of more than 5% by mass of cement (usually 15 to 35%). Additives should therefore be considered as additional constituents of the concrete mix.

Admixtures are usually categorised according to their principal uses or according to their primary effects as follows:

- Improvement of workability of fresh concrete
- Control of setting time and early hardening
- Air-entrainment
- Other effects (stability, high cohesion, colouring, etc.)

The official **definition of an admixture** according to ASTM C125:

- An admixture is a material other than water, aggregates, hydraulic cement, and fibre reinforcement used as an ingredient of concrete or mortar and added to the batch immediately before or during its mixing.
  This definition encompasses a wide range of materials that are utilised in modern concrete technology. Subsections of admixtures are:
  - **Accelerating admixtures**
    Admixtures that accelerates the setting and early strength development of concrete.
  - **Air-entraining admixtures**
    Admixtures that causes the development of a system of microscopic air bubbles in concrete or mortar during mixing.
  - **Retarding admixtures**
    Admixtures that retards the setting of concrete.
  - **Water-reducing admixtures**
    Admixtures that either increases the slump of freshly mixed mortar or concrete without, increasing the water content, or that maintains the slump with a reduced amount of water due to factors other than air entrainment.

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1.4.1 Workability agents

Admixtures in this category are either called

- plasticizers
- superplasticizers
- water-reducing agents
- high range water-reducing agents.

The primary effect of the admixtures is an improvement of workability.

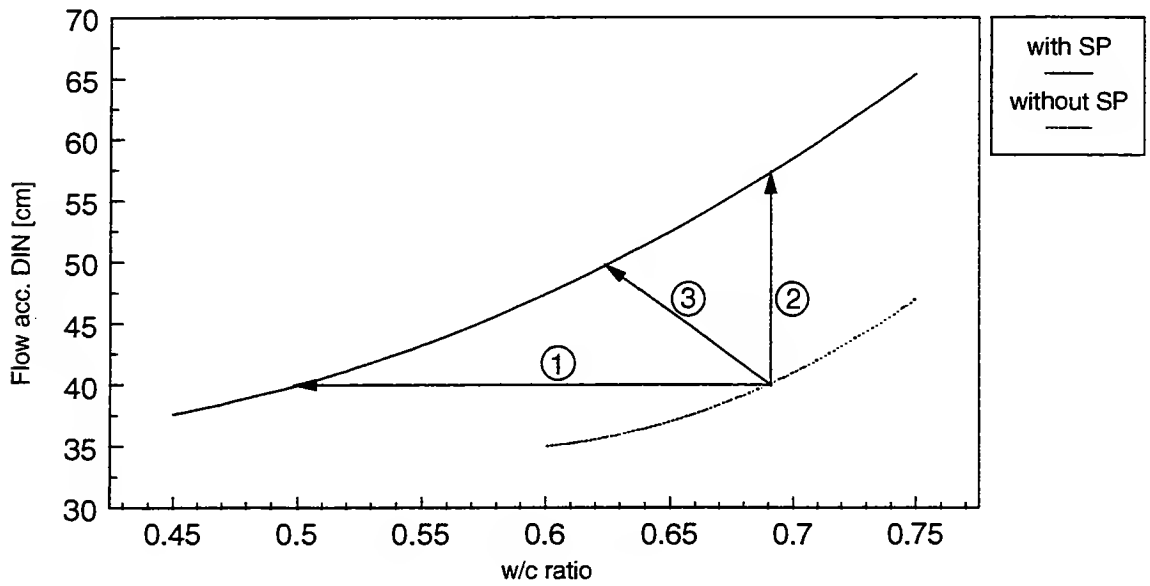
The following diagram shows the different purposes of using plasticizers in a typical concrete mix:

1 constant workability (flow); reduced w/c ratio and increased strength

2 constant w/c ratio and strength; increased workability (flow)

3 mix of both cases

Figure 3:



Admixtures in this group are all based on chemical compounds which affect the forces between solid particles suspended in water and reduces the surface tension of water. The admixtures contain surfactants which is being absorb on the surface of the particles. The strength of the adsorption depends on the type of particle. In case of Portland cement the

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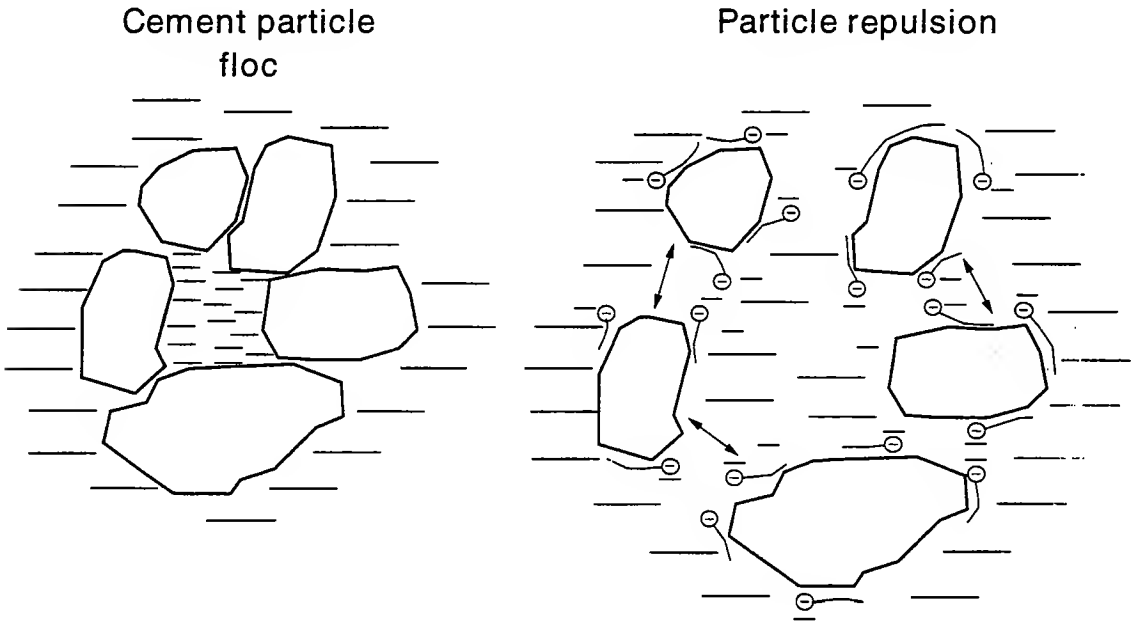
C_3A compound appears to provide the strongest attraction. The dispersing action is also enhanced by the development of a layer of adsorbed molecules of the plasticizers which separate the particles of cement; see following figure:

Figure 4:



Flocculated particles of cement before the addition of a super-plasticizer

The negative charges of the 'tails' of the molecules of the admixture adsorbed on to the cement particles generate repulsive forces and disperse the particles

The superplasticizers are broadly classified into four groups, namely:

- **SMF:** Sulphonated Melamine-Formaldehyde condensate
- **SNF:** Sulphonated Naphthalene-Formaldehyde condensate
- **MLS:** Modified Ligno Sulphonates
- **Others:** Polyacrylates, sulphonic-acid-esters, carbohydrate esters etc. A very **new invention is based on derivatives of maleic acid and vinyl monomers.**

Variations exist in each of these classes and some formulations may content a second ingredient.

Ordinary plasticizers

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Ordinary plasticizers are mainly based on lignosulphonate salts - a minority use salts of hydroxy-carboxylic acid. Calcium lignosulphonate and the better water soluble sodium lignosulphonate normally used as admixtures for concrete. The lignosulphonate molecules are in the form of polymers with molecular weights varying from approx. 20 to 30'000.

Dosage of lignosulphonate type plasticizers varies between 0.2 to 0.6% of the cement in the mix. The effectiveness of the lignosulphonate plasticizers depends on the composition of the particular Portland cement used. Cements with high content of C_3A are likely to require increase dosage of admixtures. The lignosulphonate admixtures appear to be less effective for cements with moderate high cement content than for mixes with low cement contents, below approx. 270 kg/m^3.

Lignosulphonate based plasticizers have the tendency to produce significant side effects, namely:

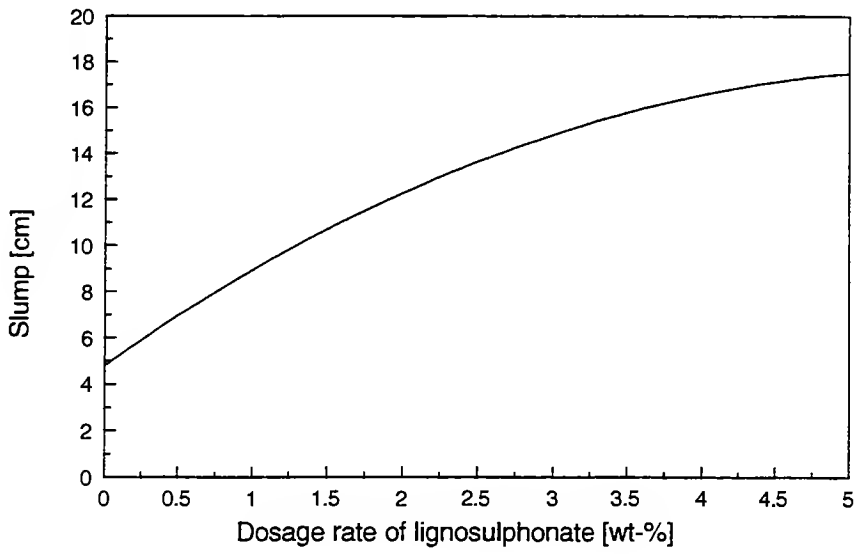
(1) Air-entrainment
The additional percentage of entrained air which can be generated by normal doses varies between 0.5 to 2.5 vol-% (slump approx. 6 cm, cement content approx. 300 kg/m^3).

(2) Retardation
The lignosulphonates themselves interfere with the hydration of cement and cause some retardation. This effect is more pronounced when the calcium lignosulphonate admixture is used. High dosages of such admixtures can lead to unacceptable retardation.

The hydroxy-carbolic plasticizers (mostly sodium salts of citric, tatratic, gluconic, and maleic acid) are not very common, and hydroxylated polymers are rarely found in concrete construction practice.

Overdosing the concrete with an ordinary plasticizer increases workability. The increase in non-linear and the effect varies with the cement content and the initial workability (slump) of the mix. The following graphic shows the increase of slump as a function of the dosage of an ordinary Portland cement:

Figure 5:



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Influence of the dosage rate of an ordinary lignosulphonate plasticizer on slump

Superplasticizers

Superplasticizers are based mainly on two types of polymers, namely the salts of formaldehyde naphthalene sulphonate and formaldehyde melamine sulphonate. The term superplasticizer indicates the much greater potential for increasing workability of concrete without undesirable side effects when compared to ordinary plasticizers.

The chemical composition of the superplasticizers differ from that of the ordinary plasticizers in that they do not delay the setting times and hardening of fresh concrete. On the contrary, some acceleration of the setting and hardening is usually observed. It is not entirely clear if the acceleration is due simply to a better dispersion of the cement particles or if other processes are also involved.

The accelerating effect of the superplasticizers appears to be primarily responsible for the relatively short periods of effectiveness of the admixture. Small additions of purified lignosulphonates are sometimes added to the admixture to moderate the rapid early loss of workability.

The periods of effectiveness of the superplasticizers have been quoted to vary between 15 minutes to 40 minutes. The period of effectiveness of the superplasticizer is usually measured as the time it takes for the workability of the mix to decrease to the level of the initial slump before the admixtures had been added.

It has been common practice in the ready-mixed concrete industry to delay the first dose of the superplasticizing admixture until the time the concrete reaches the delivery point. It is then possible to re-mix the concrete more than once, each time adding another dose of the superplasticizer and thus maintaining or even increasing workability without detrimental effect on the properties of hardened concrete.

The chemical nature of the superplasticizer determines its effectiveness in increasing the workability (slump, flow table spread, etc.). For e.g. to obtain a slump of about 25 cm from an initial value of 5 cm, it may be necessary to add 0.6% SMF or MLS-based superplasticizer whereas this could be accomplished with only 0.4% SNF-based admixture. In the following diagram the loss of workability is shown in a control concrete (containing no admixtures) compared with that containing MLS (**M**odified **L**igno **S**ulphonate), SMF (**S**ulphonated **M**elamine-**F**ormaldehyde) and SNF (**S**ulphonated **N**aphthalene-**F**ormaldehyde):

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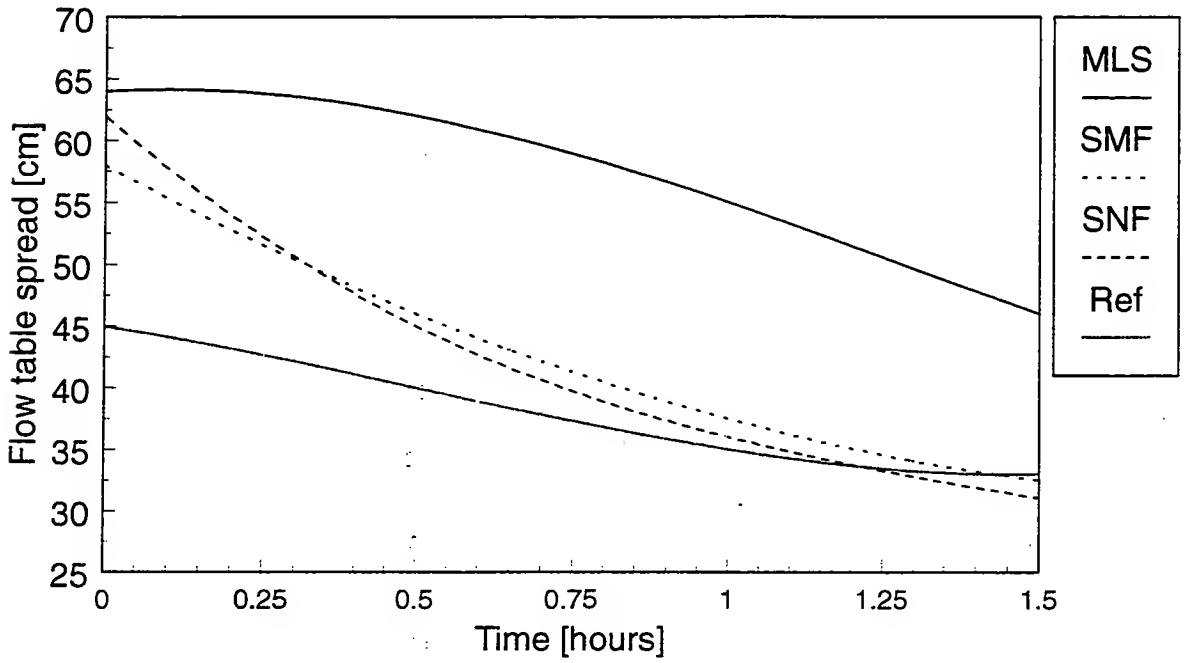
Figure 6:



Effect of MLS, SMF and SNF on loss of flow table spread

1.4.2 Air-entraining agents

The air-entraining admixtures are invariably organic substances which helps to generate microscopic bubbles of air in the fresh concrete. The majority of the commercially available ones are **based** on neutralised **wood resins** (vinsol) in which salts of abietic and pimeric acids are usual active chemical ingredients. A few air-entrained admixtures are based on the salts of fatty acids or other organic chemical compounds. It is also possible to use some types of the lignosulphonate based admixtures in higher dosages to produce simultaneous improvement of workability and generation of air-entrainment.

The technique of air-entrainment was originally developed in the USA primarily to increase the resistance of concrete to freezing and thawing and to the scaling effects produced by de-icing salts (mostly sodium or calcium chloride). The entrainment of air in concrete has been defined as the introduction into fresh concrete of air in controlled amounts and in the form of properly-sized bubbles (preferably within the size range 0.05 to 0.3 mm diameter).

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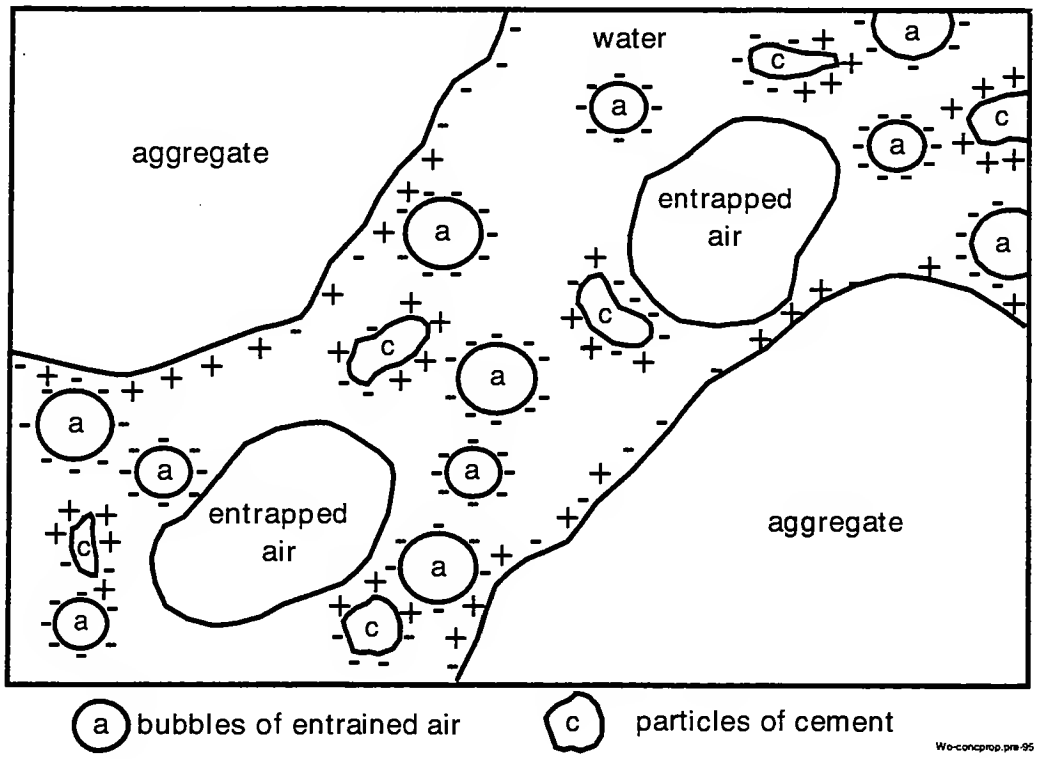
The entrained air in concrete should be clearly distinguished from accidentally entrapped air. The two differ in magnitude, amount, and properties of the air bubbles:

- **Entrained air**
  This air is intentionally incorporated by means of chemical admixtures, usually between 3 to 7% by volume and a maximum size of about 0.5 mm; entrained air produces discrete cavities in the cement paste so that no channels for the passage of water are formed and the permeability of the concrete is not increased. The voids never become filled with the products of hydration of cement as gel can form only in water.
- **Entrapped air**
  Accidentally incorporated air is called entrapped air. This air usually forms very much larger bubbles, some as large as the familiar pockmarks on the surface of the concrete. The amount is in general between 0.5 to 1.5% by volume.

The air-entrained admixtures contain surfactants which are adsorbed on to the surfaces of the cement grains. As the microscopic bubbles form during mixing of the concrete, they stabilise in their small sizes and are attracted to the layers of surfactant on the cement particles. A simplified diagram of the cement-water-air structure is shown in the following:

Figure 7:



A simplified structure of the cement - water - aggregate - entrained air system in an air-entrained concrete mix

Air entraining agents should be added as solution, dissolved in the mixing of the concrete. If other admixtures are also used, the air entraining agent should be added separately rather

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than mixed with the other admixtures, because sometimes there are reactions between materials that result in a decrease in the effectiveness of the air entraining agents.

The usual dosage rate of these materials is between 0.3 to 1.0 ml (density ≤ 1.1 g/cm^3) per kg of cement. This rate is roughly equivalent to 0.01%, solid admixtures substance to cement. The dosage rate can vary according to the composition of the mix.

Air entraining agents can be used with cements other than Portland. When used with blended cements, a larger amount of agent may be required to obtain the desired air content of the concrete.

1.4.3 Agents affecting setting and hardening

Accelerators

Accelerators have their primary application in cold weather concreting where they may be used to permit earlier starting of finishing operations. They reduce the time required for curing, and permit earlier removal of forms or loading of the concrete. Accelerators cannot be used as antifreeze agents since, at the allowable dosages, the freezing point will be lowered less than 2 °C.

Accelerating admixtures can be divided in three groups:

(1) Soluble inorganic salts

Most soluble inorganic salts will accelerate the setting and hardening of concrete to some degree; calcium salts generally being the most effective. **Calcium chloride** is the most popular choice because it gives more acceleration at a particular rate of addition than other accelerators and is also reasonably inexpensive. Admittedly it is very **corrosive** (as all water soluble chlorides) against the reinforcement of the concrete and therefore in many countries **forbidden** or limited at a particular level.

Soluble carbonates, aluminates, fluorides, and ferric salts have quick-setting properties. Sodium carbonate and sodium aluminate are the most common ingredients of shotcreting admixture formulations used to promote quick setting. Calcium fluoroaluminate can be used as admixtures to obtain rapid-hardening characteristics.

(2) Soluble organic compounds

A variety of organic compound have accelerating properties (although many more act as retarders), but triethanolamine, calcium formiate, and calcium acetate account for most commercial uses. They are commonly used in formulations of water-reducing admixtures to offset their retarding action.

Although triethanolamine is listed as an accelerator, recent research shows that its reaction with Portland cement is rather complex. It can cause retarding or flash setting, depending on the amount used.

(3) Miscellaneous solid materials

Solid materials are not often used for accelerating. Addition of calcium aluminate cements cause Portland cements to set rapidly, but strength development is poor. Concrete can be "seeded" by adding fully hydrated cement that has been finely ground during mixing to cause more rapid hydration. Finely divided carbonates (calcium or magnesium), silicate minerals, and silicas are reported to decrease setting time.

Retarders

Retarders can be used whenever it is desirable to set off the effects of high temperatures which decrease setting times. Prolonging the plasticity of fresh concrete can be used to advantage in placing mass concrete. Successive lifts can be blended together by vibration,

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with the elimination of cold joints that would occur if the first lift were to harden before the next were placed.

Retarders can also be used to resist cracking due to form deflection that can occur when horizontal slabs are placed in sections. Concrete that has set but has acquired little strength is liable to microcrack when subsequent pouring alters the amount of form deflection. If the plastic period is prolonged, the concrete can adjust to form deflection without cracking.

The composition of retarders can be divided into several categories, **based on** their chemical composition:

Lignosulphonic acids and their salts (1)
Hydroxycarboxylic acids and their salts (2)
Sugar and their derivatives (3)
Inorganic salts (4)

It will be noted that categories (1) and (2) also possess water-reducing properties, and these admixtures can be classified under both groups. Lignosulphonate-based admixtures are prepared from pulp and paper industrial wastes, and studies have indicated that most of the retarding properties of these admixtures may be due to compounds that belong to category (2) or (3). Some inorganic salts (borates, phosphates, and zinc and lead salts) can act as retarders but are not used commercially.

The influence of an admixture on air entrainment should be considered, particularly if the admixture also has water-reducing properties. Retarders may also increase the rate of loss of workability in fresh concrete (slump loss), even when abnormal setting behaviour does not occur.

Whenever a retarding admixture is used, some reduction in the 1-day strength of the concrete should be anticipated. Within 7 days, the strength should approach that of an unretarded concrete unless an overdose has been used (see following diagram):

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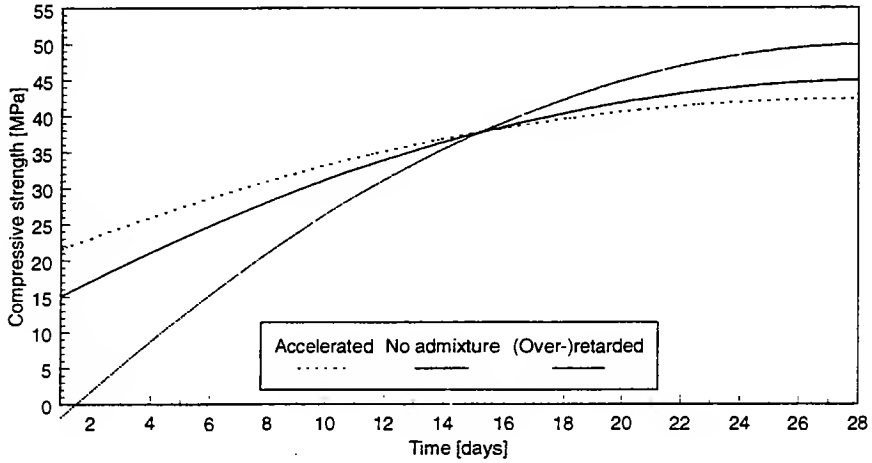
Figure 8:



Effect of retarders and accelerators on strength development of concrete

Retarding admixtures have been reported to increase ultimate compressive strength and, to a lesser extent, flexural strength. Although set-controlling admixtures are reported to increase drying shrinkage and creep, the effects depend on changes in mix design, time of hydration, and time of drying or loading.

1.4.4 Other admixtures

There are many other types of admixtures that are commercially available. The consumption of these various materials added together is less than the amount used of any single type so far mentioned. Some brief discussion of the more important kinds is warranted, however.

Bonding (or polymer) admixtures

Polymer latex emulsions are used to improve the bonding properties of concrete. This can be bonding between old and new concrete in repair work, or bonding between concrete and other materials, such as steel.

The brittle nature of concrete is an inherent property of the material and one that is overcome by the use of reinforcing materials. The very high porosity of concrete is also a disadvantage. Several approaches have been taken to improve concrete properties, resulting in quite different materials. Three different kinds of materials are often used:

PIC: **Polymer-impregnated concrete**
PIC involves filling the capillary pores of hardened concrete with a polymer

LMC: **Latex-modified concrete**
LMC is made by incorporating a polymer latex with fresh concrete, which im-proves the tensile properties of concrete.

Corrosion inhibitors

The incorporation of compounds that will prevent or reduce corrosion has been suggested for concretes in which corrosion of reinforcement can be a problem. Generally, these will be

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salts that **contain an oxidizable ion**, such as nitrites, thiosulphates, benzoates, stannous salts, and ferrous salts.

It is however, doubtful whether the use of inhibitors is really warranted, since they are not considered to provide protection in the presence of chloride ion, which is precisely the situation in which protection is desired.

Dampproofing admixtures

The term 'dampproofing' implies prevention of water penetration into dry concrete or the transmission of water through concrete. No admixture can actually prevent such movement of water, although it may reduce the rate at which it occurs.

Certain formulations **based on salts of fatty acids** (soaps) or petroleum products (mineral oils and asphalt emulsion) may give the concrete a water-repellent effect. Such materials have been used to prevent the penetration of rain into porous concrete blocks but are not likely to affect the performance of dense, well-cured concrete.

Expansive Cements

One of the major disadvantages of Portland cement concrete is its high drying shrinkage and its susceptibility to tensile strength when volume contraction is wholly or partially restrained.

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The development of expansive cements dates back about 60 years. Commercial production began in the US in the late 1960s, although total production remains quite small, about 500'000 tons annually. A standard specification, ASTM C845-90, covers expansive cements.

All three variants of present-day expansive cements (Type K, M, and S) are **based on** the formation of **ettringite** (hydrated calcium sulphoaluminate) in considerable quantities during the first few days of hydration. The material from which ettringite is formed differ substantially in each cement, but all require a source of calcium aluminate and sulphate ions. Expansion has also been achieved by using clinker with high content of free lime as admixture to cause expansion of a superplasticized concrete mix.

Successful use of expansive cements depends upon proper control of the expansion of the cement during hydration and is **sensitive to** a number of variables, for example:

Reinforcement -- Mix design -- Handling and curing

Grouting admixtures

Cement-based grout used for speciality applications such as **cementing oil well** contain different admixtures. Flocculating admixtures, thickeners, and mineral admixtures are used to prevent bleeding and segregation and to increase cohesion and retention of water during pumping. Retarders are commonly used to extend pumping times.

1.5 Mixing and curing water for concrete

1.5.1 Introduction

The quality of water is important because poor-quality water may affect the time of setting, strength development, or cause staining. Almost all natural water, fresh water, and water treated for municipal use are satisfactory as mixing water for concrete if they have no pronounced odour or taste.

Because of this, very little attention is usually given to the water used in concrete, a practice that is in contrast to the frequent checking of admixture, cement, and aggregate components of the concrete mixture.

1.5.2 Effect of impurities

A popular criterion as to the suitability of water for mixing concrete is the classical expression:

If water is fit to drink
it is all right for making concrete

This does not appear to be the best basis for evaluation, since water containing small amounts of sugar or citrate flavouring would be suitable for drinking but not mixing concrete, and, conversely water suitable for making concrete may not necessarily be fit for drinking.

The most extensive series of tests on the subject "Impurities in mixing water" was conducted by Abrams (ACI Proceedings, Vol 20, 1924, 442-486). Approximately 6000 mortar and concrete specimens representing 68 different water samples were tested in this investigation. Among the water tested were sea and alkali water, bog water, mine and mineral water, and water containing sewage and solutions of salt. Tests with fresh and distilled water were included for comparative purposes. Some of the more significant conclusions based on these data are as follows:

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The time of setting of Portland cement mixtures containing impure mixing water was about the same as those observed with clean fresh water with only few exceptions. In most instances, the water giving low relative compressive strength of concrete caused slow setting.

- Non of the waters caused unsoundness of the neat Portland cement paste when tested over boiling water.
- In spite of the wide variation in the origin and type of water used, most of the samples gave good results in concrete due to the fact that the quantities of injurious impurities present were quite small.
- The quality of mixing water is best measured by the ratio of the 28-day concrete or mortar strength to that of similar mixtures made with pure water. Water giving strength ratios that are below 85% should be considered unsatisfactory.
- Neither odour nor colour is an indication of quality of water for mixing concrete. Water that was most unpromising in appearance gave good results. Distilled waters gave concrete strength essentially the same as other fresh waters.
- Based on a minimum strength-ratio of 85% as compared to that observed with pure water, following samples were found to be unsuitable for mixing concrete:
  - acid water
  - lime soak water from tannery waste
  - carbonated mineral water discharged from galvanising plants
  - water containing over 3% sodium chloride or sulphate
  - water containing sugar or similar compounds

  The concentration of total dissolved solids in these waters was over 6'000 ppm except for the highly carbonated water that contained 2'100 ppm total solids. Very few natural waters other than sea water, contain more than 5'000 ppm of dissolved solids.
- Based on the minimum strength-ratio of 85%, the following waters were found to be suitable for mixing concrete:
  - bog and marsh water
  - waters with a maximum concentration of 1% sulphate
  - sea water (but nor for reinforced concrete)
  - alkali water with a maximum of 0.15% Na_2SO_4 or NaCl
  - water from coal and gypsum mines
  - waste water from slaughterhouses, breweries, gas plants, and paint and soap factories.

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1.5.3 Effects of algae on air content and strength

A rather extensive series of laboratory tests showed that the use of water containing algae had the unusual effect of entraining considerable quantities of air in concrete mixtures with an accompanying decrease in strength. The data in the following table were based on tests with 19-mm maximum-size aggregate concrete having a w/c ratio of 0.5 and a slump of 4.0 to 7.5 cm, with a constant ratio of coarse to fine aggregate:

Table 13:

Algae in mixing water [%]	Air in concrete [%]	Compr. strength at 28 days [MPa]
none (control mix)	2.2	33.3
0.03	2.6	33.4
0.09	6.0	27.9
0.15	7.9	22.8
0.23	10.6	17.8

In addition to the detrimental effect of strength, one of the important aspects of these data is that considerable quantities of air can be entrained in concrete by the use of mixing water containing algae.

1.5.4 Curing water

There are two primary considerations with regard to the suitability of water for curing concrete:

- One is the possibility that it might contain impurities that would cause staining, and the other
- that it might contain aggressive impurities that would be capable of attacking or causing deterioration of concrete. The latter possibility is unlikely, especially if water satisfactory for use in mixing concrete is employed.

In some instances the staining or discoloration of the surface of concrete from curing water would not be objectionable.

The most common cause of staining is usually a relatively high concentration of iron or organic matter in the water; however, relatively low concentrations of these impurities may cause staining, especially if the concrete is subjected to prolonged wetting by run off of curing water from portions of the structure.

Test data show that there is no consistent relationship between dissolved iron content and degree of staining. In some cases, 0.08 ppm of iron resulted in only a slight discoloration and in other cases, waters with 0.06 ppm of iron gave a moderate rust-coloured stain, while 0.04 ppm produced considerable brownish-black stain.

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3. APPENDICES

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3.2 Relevant ASTM Standards

3.2.1 Aggregate

C 33-90	Specification for Concrete Aggregates
C 40-84	Test for Organic Impurities in Sands for Concrete
C 70-79	Test for Surface Moisture in Fine Aggregate
C 88-90	Test for Soundness of Aggregates by Use of Sodium Sulphate or Magnesium Sulphate
C 123-90	Test for Lightweight Pieces in Aggregates
C 131-89	Test for Resistance to Degradation of Small-Size Coarse Aggregate by Abrasion and impact the Los Angeles Machine
C 227-87	Test method for Potential Alkali Reactivity of Cement-Aggregate Combinations (Mortar-Bar Method)
C 289-87	Test method for Potential Reactivity of Aggregates (Chemical Method)
C 294-86	Descriptive Nomenclature of Constituents of Natural Mineral Aggregates
C 330-89	Specification for Lightweight Aggregates for Structural Concrete
C331-89	Specification for Lightweight Aggregates for Concrete Masonry Units
C 332-87	Specification for Lightweight Aggregates for Insulating Concrete
C 586-86	Test method for Potential Alkali Reactivity of Carbonate Rocks for Concrete Aggregates (Rock Cylinder Method)
C 637-84	Specification for Aggregates for Radiation-Shielding Concrete
C 638-84	Descriptive Nomenclature of Constituents of Aggregates for Radiation Shielding Concrete
E 11-87	Specification for Wire Cloth Sieves for Testing Purposes

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Chapter 8

Fresh and Hardened Concrete

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FRESH AND HARDENED CONCRETE

By Horst Wolter, Product Development and Application, HMC

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1. GENERAL

The properties of a building material are of great importance with respect to the function for which the building was intended. Besides the required load capacity and durability, which are the main conditions, a building must protect against cold, heat, rain, wind, etc. A comfortable and pleasant living atmosphere should be created, which cannot be achieved simply by the design of the building or the application of the specific material, but by the favorable combination of several materials. The correct choice of material, however, can only be made if everybody involved with the construction has good knowledge of the material properties.

The properties of building materials such as bricks and steel will remain unchanged throughout the building process, while the final properties of concrete will only become established after placing. Cement is only an intermediate product of a building material.

New developments in concrete technology and building technique, based on many years of research, have made it possible to build such constructions as the 550 m tall TV tower in Toronto, prestressed concrete bridges with a span of up to 240 m and drilling platforms in the ocean. Flowing concrete has changed the method of placing. So far, concrete has been able, in most cases, to compete with other building materials.

All of these constructions and techniques have been made possible by the improvement of cement quality and on the methods of its application. Should further material requirements arise, would it be possible to adapt the concrete properties? Can the cement manufacturer contribute? He should at least make an attempt to meet any new requirements, because each successful concrete construction will favorably affect the cement production.

On the other hand, the cement manufacturer must be able to defend himself if defects in concrete are unjustly attributed to the cement. He can only be persuasive if he has thorough knowledge of concrete properties and concrete technology.

2. DEFINITION

2.1 Defintion of concrete

ACI has defined concrete as follows:

Concrete is a composite material consisting essentially of a binding medium, within which particles or fragments of aggregate are embedded.

In Portland cement concrete, the binder is a mixture of Portland cement and water.

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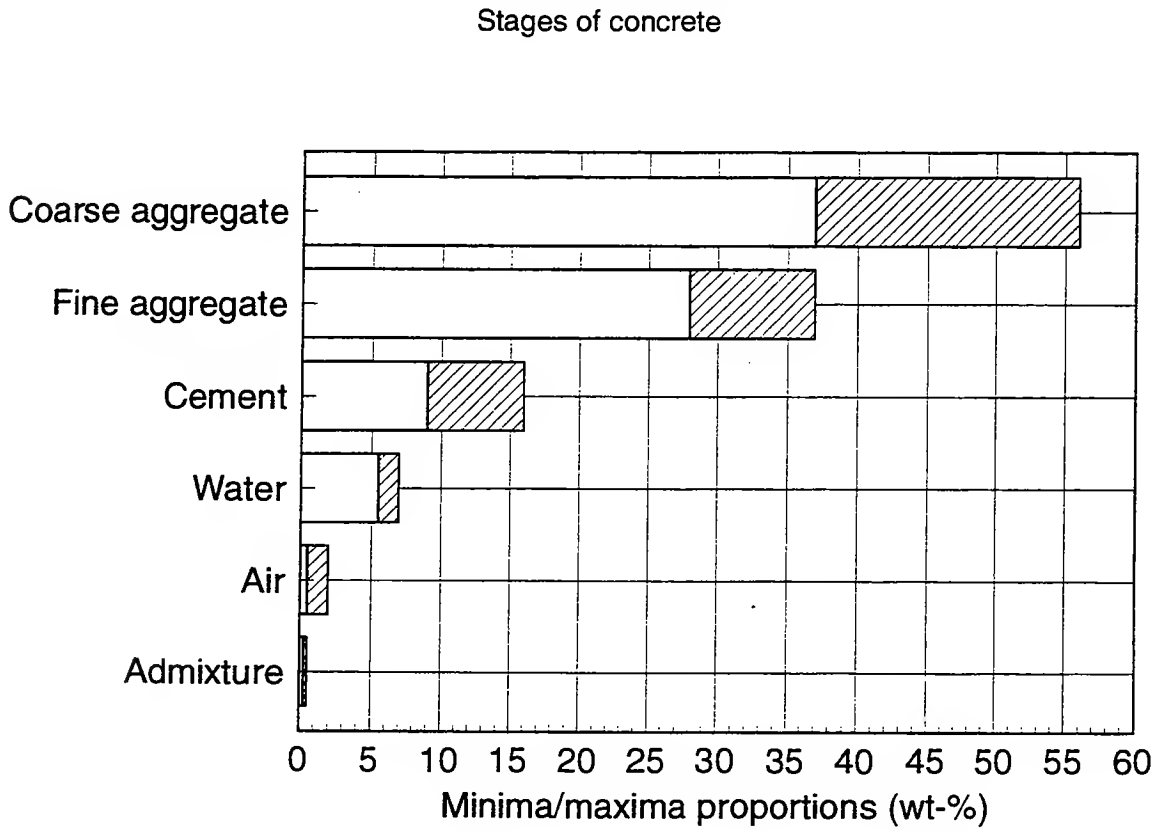
2.2 Composition of concrete

Fig. 1: Range of Proportions of materials used in concrete (by weight)



The properties of concrete depend very much on its age. Roughly three stages can be distinguished.

Fresh concrete
(a workable mass)

⇩

Transition
through intermediate stages
(sometimes called 'green' or 'young' concrete)

⇩

Hardened concrete
(an artificial stone which has reached the required properties for a specific structure)

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The transition from fresh to hardened concrete is a continuous process during which it changes from a workable mass to an artificial stone. The performance of the concrete at certain ages has a great influence on its applicability, and, therefore, it is of interest to know the concrete properties not only when placed (fresh) or used (hardened), but sometimes also during the entire development.

2.3 Green and young concrete, workability and resistance to loading of concrete

The concrete is called **'green'** as soon as it is compacted in the framework until its solidification by setting. When the concrete turns solid, it is called **'young'** until it reaches a certain degree of strength permitting the removal of the form.

For the practical behaviour of concrete in service, its mechanical properties are decisive. Its **resistance to loading** is essential in the hardened stage. It is, however, insignificant in fresh concrete where the **workability** is of main concern.

Workability is not well defined and, therefore, not directly measurable, because it includes a certain number of fresh properties. As an example, two definitions of workability from the same country (USA) follow:

- WORKABILITY is that property of freshly mixed concrete or mortar which determines the ease and homogeneity with which it can be mixed, placed and compacted and finished. (ACI definition)
- WORKABILITY is that property of concrete which determines the effort required to manipulate a freshly mixed quantity of concrete with minimum loss of homogeneity. (ASTM definition)

Resistance to loading is the ability of the hardened concrete to bear the service load (dead and live load); (dead load = constant load in structures due to the mass of the members, the supported structure and permanent attachments or accessories; live load = any load that is not permanently applied to the structure).

Resistance to loading can be specified and measured exactly for different types of loads. Usually, it is expressed as strength.

2.4 Strength of concrete

Strength is defined as maximum resistance to load that a member or structure is capable to develop before failure occurs. It is measured with reference to the cross section of the structure member.

$$\text{Strength:} \quad \sigma = \frac{F}{A} \quad \left[\frac{N}{mm^2} \quad \text{or} \quad MPa \right]$$

where: F = applied load
A = cross section on which the load is applied

We distinguish different types of strength according to the type of load exerted:

Compressive, tensile, flexural splitting, shear, torsion adhesive, and impact strength.

The loading resistance of various structural members results from the combination of different types of strength, according to the actual stresses, and from concrete quality.

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3. FRESH CONCRETE

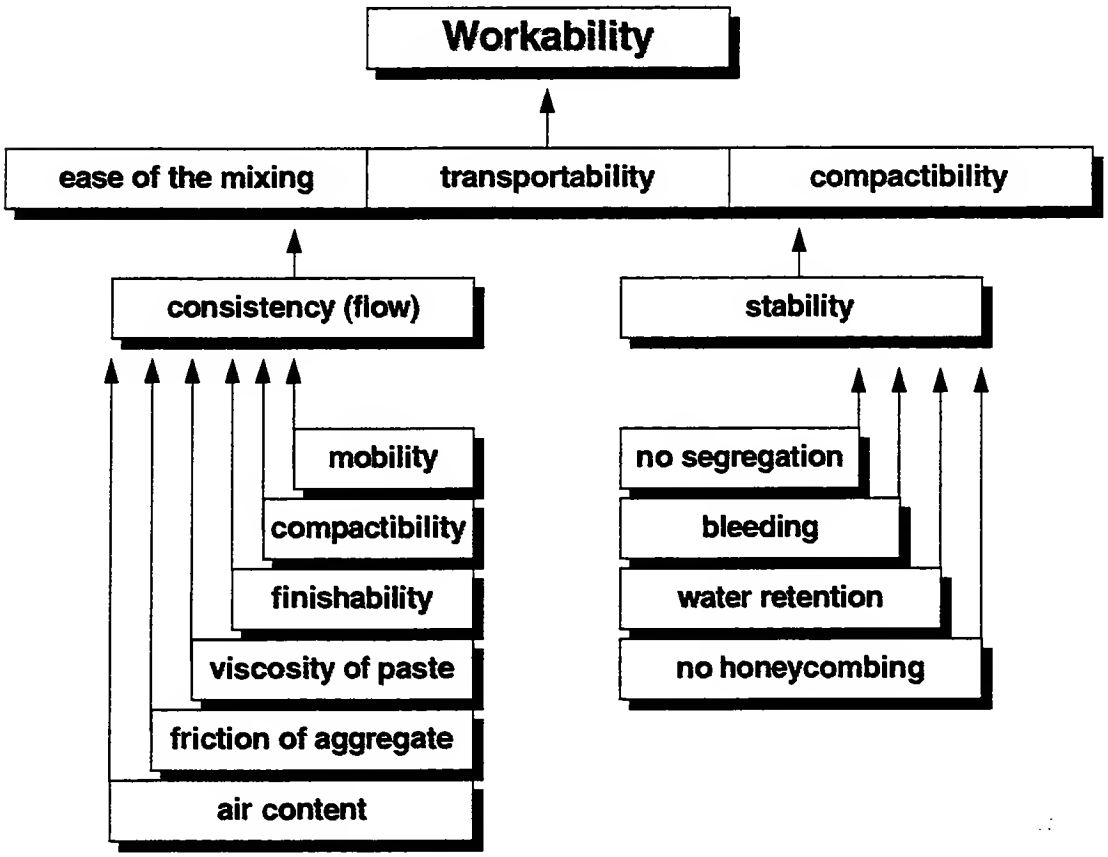
Fresh Concrete is the product that is obtained immediately after mixing the components.

3.1 Workability

The homogenized concrete mix, after it is taken from the mixer, must be suited to be transported to the destination point and to be placed into the moulds or formwork. The fresh concrete must completely fill the mould, even if the shape of the elements is very complicated and the interstices between the reinforcement are very small. A good compaction must be achieved and through all these steps the mix must remain homogeneous without segregation.

The ability of fresh concrete to meet all these requirements is called **workability** which includes a number of more or less defined properties (see following figure).

Fig. 2: Properties of fresh concrete related to workability



The workability influences not only all operations of placing and consolidating fresh concrete but also to a large extent, the quality of the hardened concrete. A dense concrete must contain a high amount of solid matter and very few voids filled with water, vapor or air. Very important concrete properties depend on the density.

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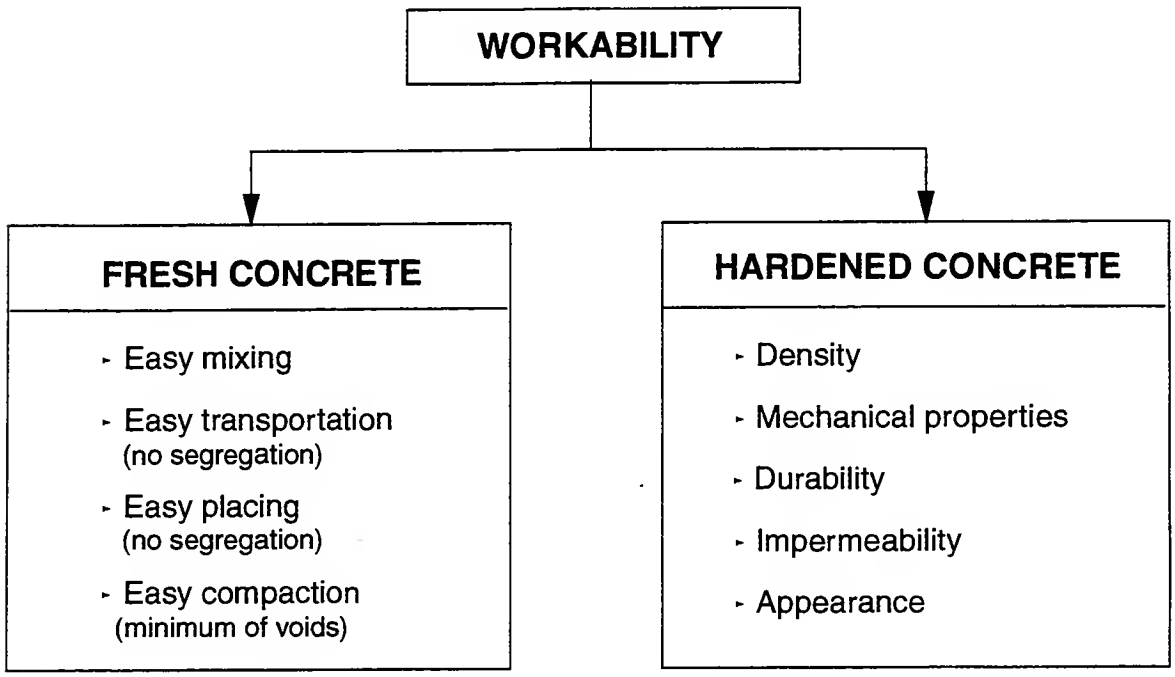
Fig. 3: Effect of the workability on fresh and hardened concrete properties



The workability requirements vary from one country to another. They depend on the level of the building industry, on the quality requirements and on the quality of the available materials. Furthermore, tradition, economical and subjective factors influence the technical demands.

To characterize the workability of concrete mixes, the following terms are usually used in Central Europe:

Terms for workability:
very stiff
stiff
plastic
soft (wet)
flowing (liquid)

However, those terms are not clearly defined. In England for instance, completely different terms are used:

- Low, medium, and high consistency.

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3.2 Influence on the Workability

The workability of concrete is influenced in various ways by its solid and liquid components and environmental conditions.

3.2.1 Effect of water

The workability is related to the water content that is available for the lubrication of a mix. By increasing the water content in the mix up to a certain limit, the workability of the mix can be improved, i.e. the mix will be more wet. If the water content exceeds a certain amount, there is the danger of segregation.

The water content of the mix is expressed in liters or kilograms of water per cubic meter of concrete (pound per cubic yard).

3.2.2 Effect of solid constituents

Solid matter (aggregate + cement) shows a double effect on the workability:

- The shape of the solid grains influences the mobility of concrete in the following manner:
  - Coarse particles impede the mobility by their angularity and friction;
  - The fine grains, on the other hand, improve it because they act almost like ball-bearings between the coarse grains. This action is distinctly noticeable when lime or fly ash are added to concrete: their spherical particles improve concrete consistency.
- By absorption of the surface, eventually also by chemical bond, part of the mixing water is fixed. Only the remaining 'mobile' part influences concrete consistency.

The effect of aggregate properties on workability is considerable.

> **Note:**
>
> Grain size distribution, angularity and surface texture of the aggregates significantly affect concrete consistency.
>
> The effect of cement properties on workability is very small

The effect must be attributed to the water requirement of cement which can be determined with standard methods on the cement paste. The chemical-mineralogical composition has a stronger influence on the water requirement than fineness and grain size distribution. Grain size distribution shows only little variation in industrial cements.

If the amount of cement added to a concrete mix is increased at a constant water content, a stiffening action is the result because more water is absorbed. If, however, both cement and water are increased, so that the w/c ratio remains unchanged, the lubricating action of the additional cement paste increases the fluidity.

Thus, the workability of a given aggregate mix can be improved in two ways: by increasing the water content only, or by increasing the dosage of both, water and cement. In the first case the quality of hardened concrete will be inferior; in the second case the quality is maintained but at higher costs.

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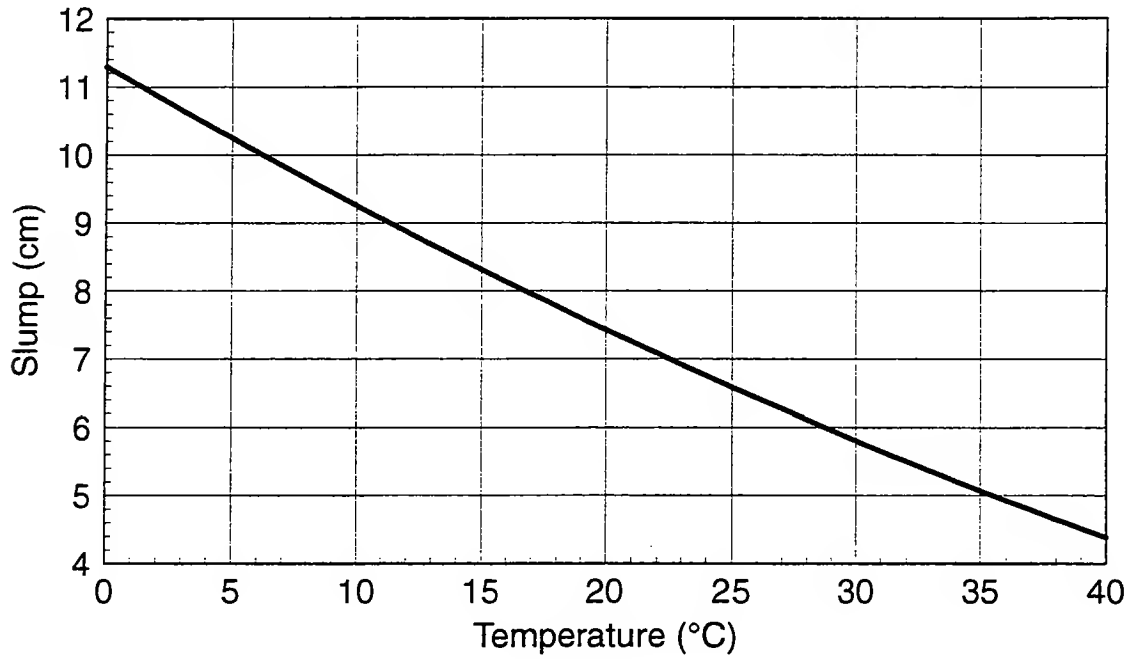
3.2.3 Effects of admixtures

Some admixtures such as plastifiers, water reducers and superplastifiers, can modify the concrete consistency even if added in small quantities (see chapter 'Concrete Main Components', paragraph 3.).

3.2.4 Effect of temperature

Increased temperature of the concrete and surrounding air contributes towards a stiffer consistency.

Fig. 4: Effect of temperature on the consistency (slump)



The consistency in the above mentioned figure was tested immediately after the mixing of the concretes according to the same mix design. Only the temperature of the materials (cement + aggregate + water) and the ambient temperature were changed.

3.3 Other properties of fresh concrete

3.3.1 Bulk density (unit weight)

The unit weight of fresh concrete gives some indication about the final void content which is responsible for final concrete properties. It is also a measure of the yield; i.e. it indicates the concrete volume produced with a specific amount of cement. The bulk density is determined by weighing a defined concrete volume. It depends very much on the water and less on the cement dosage. After compaction, the unit weight of ordinary concrete should be higher than 2300 kg/m^3.

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3.3.2 Air content

Ordinary concrete which is well compacted, has an air content of about 0.5 to 1.5%. The measuring of the air content is important if entrained air is used. Air-entrained concrete is produced by using either an air-entrained cement or an air-entraining admixture during the mixing of concrete. Air-entrained concrete contains 3 to 7 Vol. % pores. - Entrained air improves the workability of fresh concrete.

3.4 From fresh to hardened concrete - Stages of development of concrete

'Green' and 'young' concrete are the terms for specific intermediary stages during the transition of concrete from a workable, more or less plastic mass, to an artificial stone.

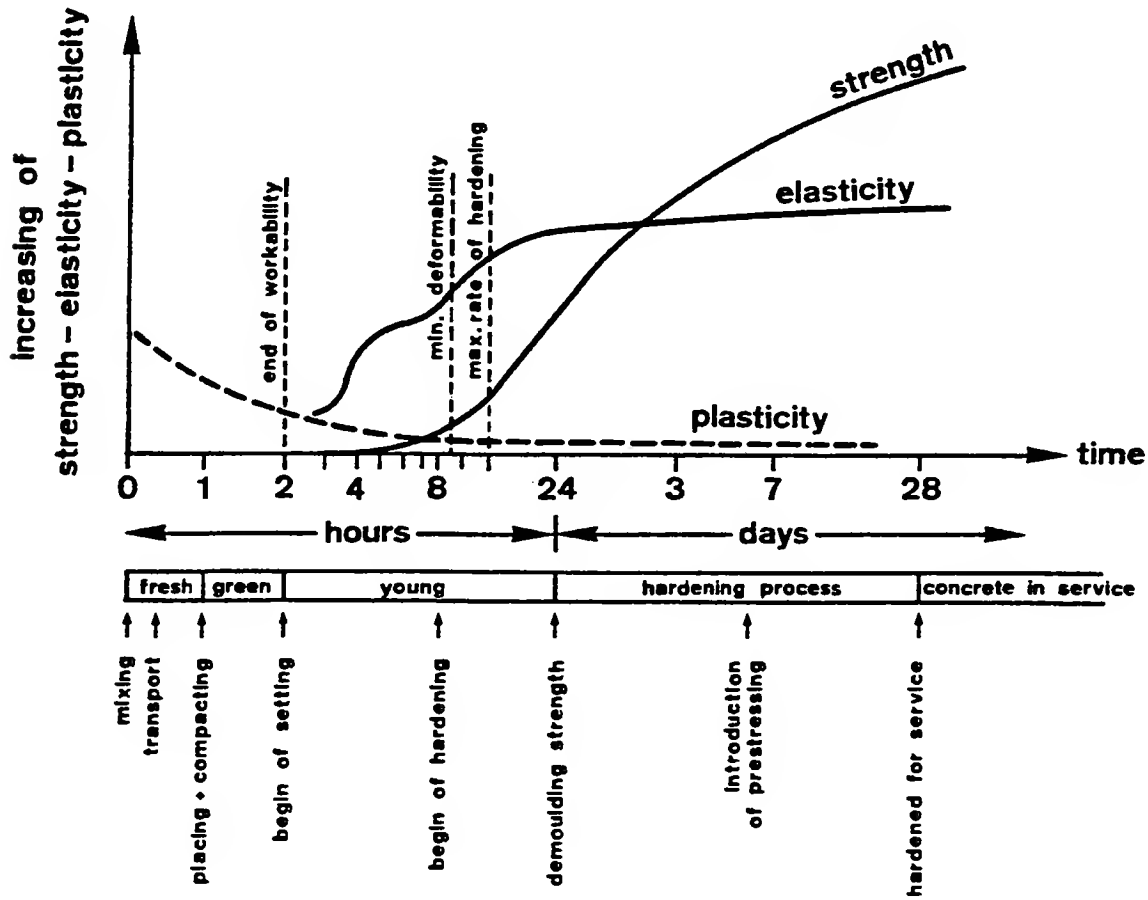
With the development of modern concrete technology and building methods, the properties of concrete at these intermediate stages have become important as well. The continuous development of hardened concrete is a sequence of periods of:

Concrete is a process:

Stiffening → Setting → Hardening

These periods are characterized by different growth rates of the various mechanical properties of the concrete (see following figure).

Fig. 5: Stages of development of concrete



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Stiffening
of concrete is a change in the workability (sometimes also called slump loss) and begins immediately after mixing, sometimes even during mixing.

This can have unfavourable consequences if fresh concrete is not placed properly after mixing, or, especially if ready mix concrete is used, in precast manufacturing, or on large building sites and placing has to be delayed.

Concrete should be placed as soon as possible after mixing, but often there may be a time lapse of up to one hour or even more between mixing and placing. In these cases, too rapid stiffening can impede placing.

> The time during which concrete is still workable - at the right consistency - is very important for transporting and placing.
>
> This time depends on the purpose for which concrete is used and on the method of placing.

Setting
is the beginning of the hydration process and is indicated by the transition from the plastic to the solid state within a relatively short period of time.

The resistance against deformation during setting increases rapidly, strength less rapidly.

> **Note:**
>
> The time of setting of concrete is not identical with that of cement measured on paste according to standards.

The reason is that the setting of concrete is influenced not only by the setting behavior of the cement, but also by varying cement and water contents, by temperature and type of construction.

Until setting, deformations of concrete are almost totally irreversible; the elastic part of deformations becomes predominant only after setting.

Hardening
is the subsequent improvement of mechanical properties of concrete after setting. Soon after setting, the strength begins to increase more rapidly than the elasticity (Young-modulus). The rate of increase reaches its maximum between 5 and 20 hours after the addition of water.

3.5 Properties of 'Green' and 'Young' Concrete

Green concrete:

The mechanical properties of green concrete are important in practice, especially in the precast industry, where blocks, tubes and other concrete products are moulded by pressing. The moulded pieces have to be removed from the press as soon as possible in order to make room for the next series. However, this is only possible if the cohesion of the concrete is high enough, even before setting, to bear its own weight without deformation. Sometimes the moulded pieces (e.g. blocks) are stored in several layers. In this case, the strength after demoulding will have to be higher, so as to carry the whole load.

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Young concrete:

Since young concrete at early age very often has to support a mechanical stress, it must have a minimum strength. Thus, the strength at this stage (after some hours) is decisive in the precast industry, where it determines the intervals at which precast elements can be demoulded, transported and piled up. It is also important in other building processes, for instance with prestressed concrete. The strength requirements vary according to the size and shape of the concrete element and the building method.

Early strength:

The compressive strength during the first few hours after placing is called early strength.

It must reach approx. 10 to 20 MPa for demoulding in reinforced concrete and

35 MPa for the release of the prestressing wires in the pretensioned, prestressed concrete.

4. HARDENED CONCRETE

4.1 General

Hardened concrete is the final building material as it is obtained after stiffening, setting and hardening of fresh concrete. Hardening, however, continues for many years, slowing down after a certain strength has been reached. Therefore, it is generally accepted to consider the properties of hardened concrete at the age of 28 days as characteristic. They are commonly used for the design of concrete structures.

Hardened concrete must meet various requirements and maintain its properties during a very long time. Most important are the load-bearing properties and the durability of concrete.

The resistance to loading of concrete depends on its strength. But a building can never be loaded to the strength limit. Structural members are designed in such a way that the calculated stresses in concrete do not exceed certain permissible working values. According to the function of the building, other special requirements may be of importance.

A large variety of concrete properties can be obtained by the choice of the components and the mix design. However, the nature of concrete, the availability of the materials and economical considerations limit these possibilities.

4.2 Strength

According to the type and direction of load and stress, various types of strength react in concrete:

- **Compressive strength:**
  Compressive strength is the most important characteristic of hardened concrete. High compressive strength is, in most cases, accompanied also by an improvement of the other properties. compressive strength determinations show the best reproducibility. Therefore, compressive strength is considered as a general measure of concrete quality. The compressive strength of commonly used concrete after 28 days is in the range of 10 to 70 N/mm^2. The low values (~ 10 to 20 N/mm^2) are used for plain concrete, the medium values (~ 20 to 45 N/mm^2) for reinforced concrete and the high values (~ 50 to 70 N/mm^2) for prestressed concrete and precast elements.

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- **Tensile strength:**
  Tensile strength is relatively low and amount to only approx. 1/10 of the compressive strength of hardened concrete. Because the tensile strength is low, it is practically not taken into consideration for structural design. Tensile stresses in the construction are carried by steel reinforcements.
- **Flexural strength**:
  It is difficult to determine the tensile strength of concrete and therefore bending or flexural strength is measured.
- **Impact strength;**
  Impact strength plays a significant role in special applications, e.g. for piles to be driven into the ground.

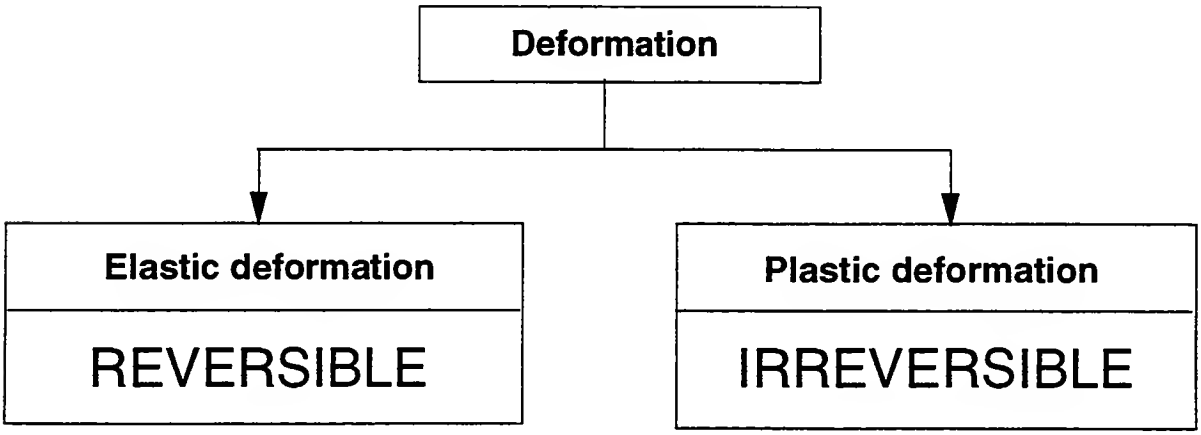
4.3 Deformation under load: modulus of elasticity and creep

Even momentary loads cause deformation of concrete. If they exceed certain limits, there is a risk of cracking.

> **Note:**
> Concrete is a plastic-elastic material

Its deformation is always composed of the two components: elastic and plastic.

The elastic deformation disappears when the load is removed, the plastic deformation remains.



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In hardened concrete, the main part of deformation is elastic.

A quantitative measure of elasticity is the ratio between stress and corresponding strain. This ratio is termed **modulus of elasticity E** (Young modulus = initial tangent modulus).

$$E = \frac{\sigma}{\varepsilon}\ [GPa] \qquad \varepsilon = \frac{\Delta L}{L_0}\ [1]$$

The modulus of elasticity may be measured in tension, compression or shear. The modulus in tension is usually equal to the modulus in compression and is frequently referred to as Young Modulus of elasticity (Table).

Table 1: Modulus of Elasticity of Different Building Materials

Material	Modulus of elasticity [GPa]
Steel	200 to 230
Aluminium	74
Copper	130
Natural stone	12 to 80
Concrete	20 to 50
Mortar	5 to 20
Timber	6 to 15

Due to the fact that concrete is neither ideally plastic nor ideally elastic material, the manner in which its modulus of elasticity is defined is somewhat arbitrary. Various forms of the modulus, which are used, are illustrated on the stress-strain curve in following diagram:

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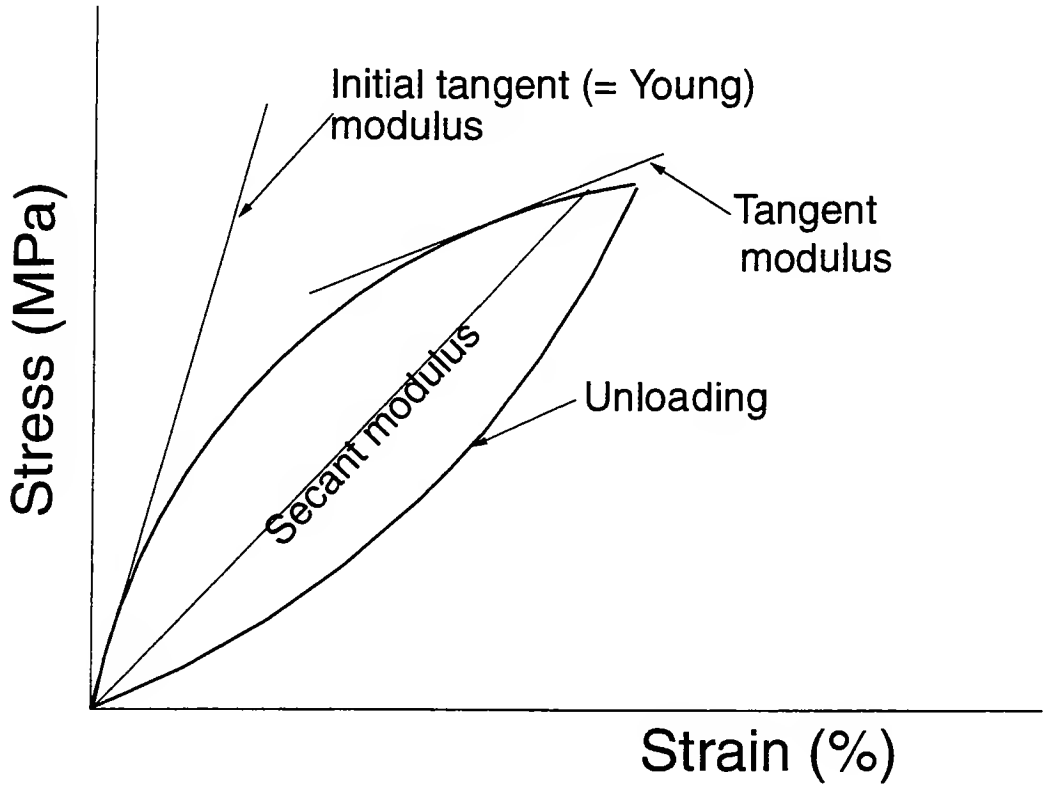
Fig. 6: Typical stress-strain curve for concrete



It also has to be distinguished between the 'dynamic E-modulus' and the 'static E-modulus'. The dynamic E-modulus is applied when concrete is exposed to oscillation. It can be calculated, for instance, from the rate of propagation of ultrasonic impulses and is a measure for the progression of deformation resistance. Using certain formulas, the compressive strength can be estimated from these results.

Creep:
Plastic deformation becomes more pronounced with longtime loading. The irreversible deformation by longtime loading is called creep.

> **Definition of plastic deformation:**
>
> The modulus of deformation is the ratio between the applied load and the irreversible deformation, expressed as function of time.

The creep is strongly influenced by the stress/strength ratio. With higher stress/strength ratio the creep increases. It also depends on other factors, such as temperature, humidity, etc. Creep attains approx. 1.5‰ per 1 MPa load during 1 year. Besides being a disadvantage, creep is also useful because it diminishes internal stresses. On the other hand, it reduces the effect of prestressing.

The deformability of concrete has certain limits:

- If the stress grows beyond these limits, the concrete cracks or breaks.
- Strength and resistance against deformation dictate the limits:

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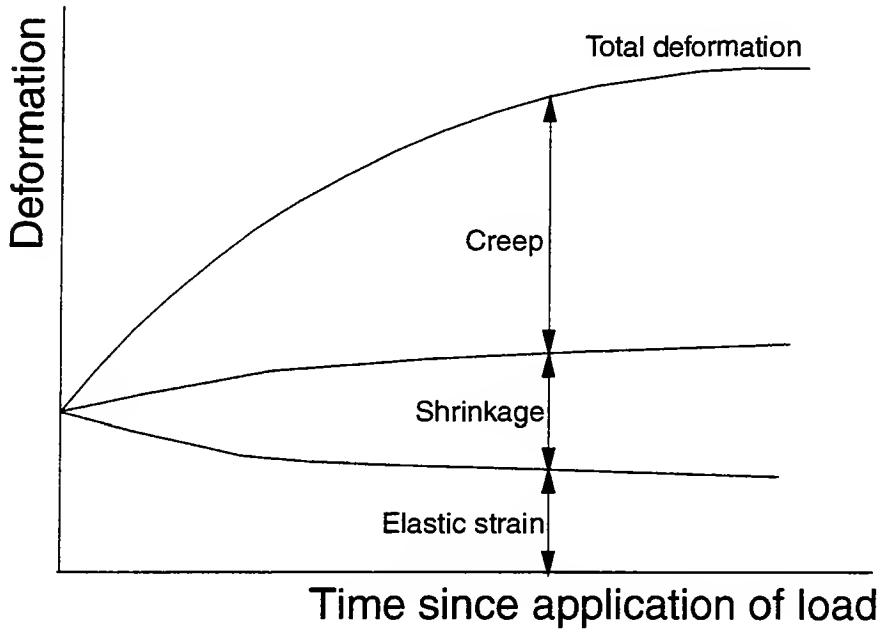
- The higher the strength and the lower the resistance against deformation, the higher the limits of deformation.

$$\text{Deformation limit: } \varepsilon_{lim} = \frac{\sigma_{lim}}{E} \cdot 1000\,(‰)$$

Where: σ = strength, E = Young modulus

Fig. 7: Time-dependent deformations in concrete subjected to a sustained load



The deformability of hardened concrete is in the range of about 1‰. Another characteristic of the deformability is the ratio of the transverse strain to the longitudinal compression or of the transverse contraction to the longitudinal strain.

$$\text{Poisson's ration} = \frac{\varepsilon_{transverse}}{\varepsilon_{longitudinal}}$$

While the modulus of elasticity indicates only the alteration of size in one direction, the Poisson's ratio indicates to what extent the load modifies the shape and the volume of the concrete (compressibility). It decreases with time and is about 0.2 in hardened concrete.

Poisson's ratio and modulus of elasticity describe completely the deformation behavior of a material.

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4.4 Density (weight of volume unit)

The dry density of the dry solid mass and the bulk density of the concrete have to be distinguished.

Density:

The density (specific gravity) - the weight per unit volume of a dry and pore-free substance (mass) - depends above all on the mix design as the density of cement (ρ = 3.0 to 3.15 g/cm^3), aggregate (ρ = 2.6 to 2.7 g/cm^3) and water ((ρ = 1.0 g/cm^3) vary little. The specific gravity of concrete is in the order of 2.3 to 2.5 g/cm^3.

Bulk density:

The bulk density (unit weight) - the weight of concrete per unit volume including voids - (about 2.2 to 2.4 t/m^3 for ordinary concrete) depends on the compaction of the concrete.

The ratio

$$\frac{Specific\ gravity - Bulk\ density}{Specific\ gravity}$$

indicates the amount of voids in the bulk volume which are filled with water or air.

Voids weaken all mechanical properties of concrete and are, furthermore, decisive for the impermeability and thus the durability of concrete. Therefore, it is most important to compact the concrete as firmly as possible. Well compacted concrete contains not more than 1 to 2% voids of its volume.

4.5 The durability of concrete

It is essential that concrete keeps its shape and size and does not suffer any deterioration of its substance which could also cause volume changes (soundness).

As was shown in previous chapters, the mechanical properties of concrete generally continue to improve with time. Concrete resists moisture and putrefaction as well as high temperatures (between 200 and 300°C). It does not burn. These are very important advantages that concrete has over other building materials.

In this paragraph, some factors influencing the durability are discussed.

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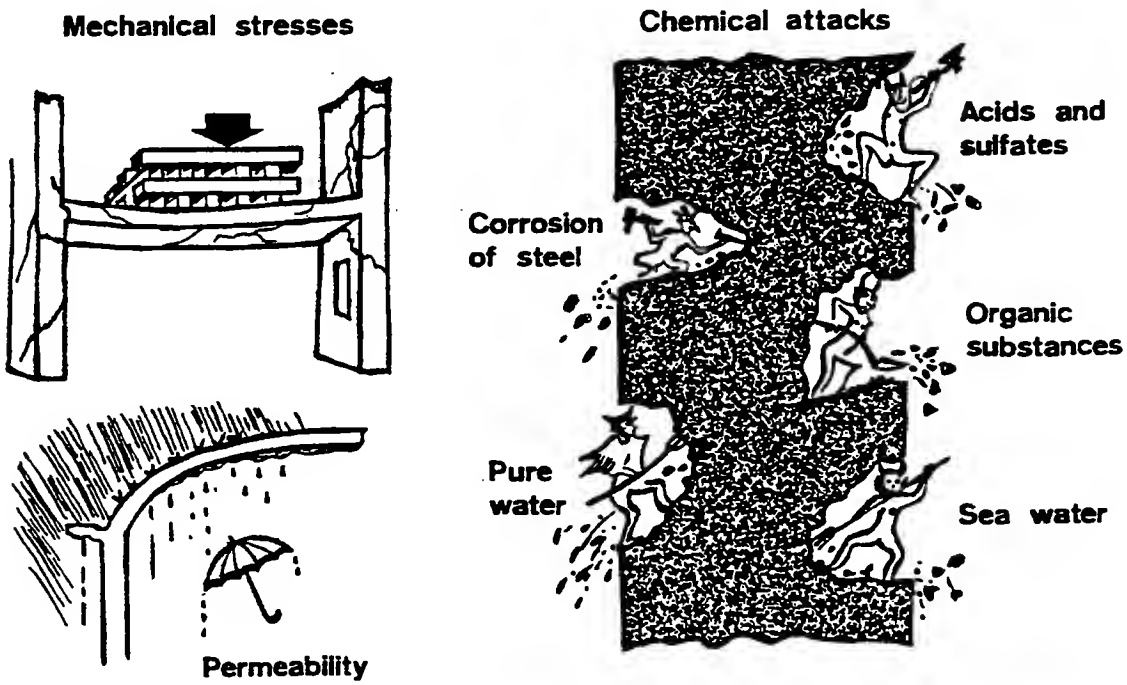
Fig. 8: Durability of concrete



4.5.1 Volume changes

Concrete is always subject to changes of volume which can be detrimental if they exceed certain limits.

4.5.2 Volume changes due to physical stress

There are several types of stress which may cause volume changes:

- **Temperature Changes**
  Temperature changes cause expansion and contraction in concrete as in all other materials. These changes are in the same order of magnitude as in steel (approx. 10^{-5} for 1°C). If these volume changes are impeded, high stresses result, causing cracks. Changes due to the freezing - thawing cycle are very hazardous. At freezing temperatures, the water volume in concrete increases by about 9%. Due to this expansion and thermal stress, concrete can disrupt. The risk is heightened if de-icing agents (salts) are employed, because of additional complicated, detrimental effects. The best way to avoid this reaction is to created voids allowing the water to expand. This is done by enlarging pore volume by about 3 to 7 Vol.% with the aid of air-entraining agents (air-entrained concrete), (see chapter 'Concrete Main Components', paragraph 3.4).

- **Moisture Changes:**
  Moisture changes - desiccation or absorption of water in hardened concrete - cause shrinkage or swelling. These changes continue until an equilibrium of moisture with the surroundings is reached which usually takes several years. Maximum final shrinkage (determined as a change in length) of the hardened concrete is 0.2 to 0.65 mm/m, maximum swelling through storage in water at 20°C, 0.1 to 0.3 mm/m depending on atmospheric conditions.

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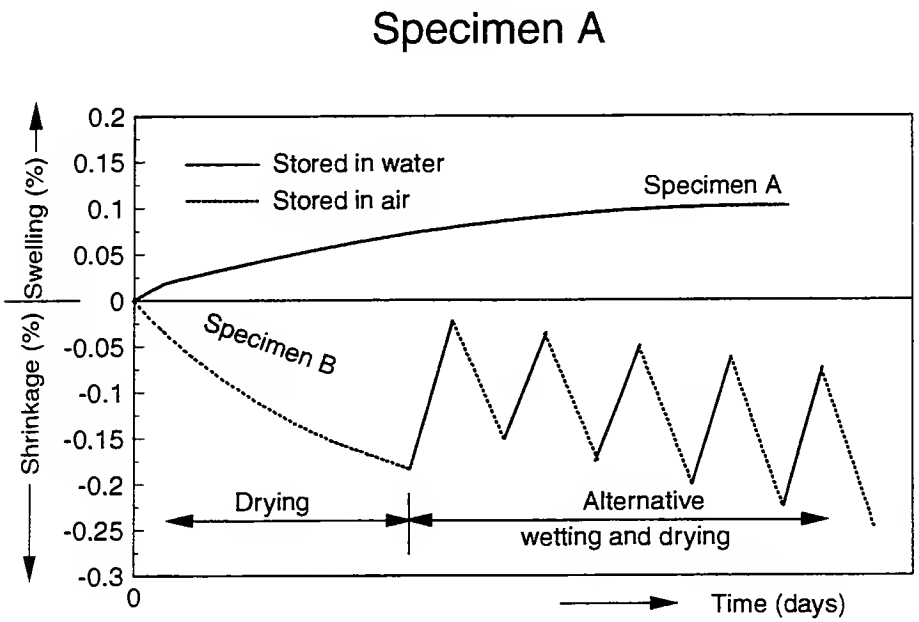
Fig. 9: Moisture movements in concrete - swelling and shrinkage



Much higher and also more dangerous is shrinkage before the setting of concrete - plastic shrinkage. At intensive desiccation it can reach several per mill (‰). If a lot of water is allowed to evaporate during the critical period, namely when concrete has only low strength and is not prevented from shrinking, e.g. through reinforcement, plastic shrinkage-cracks may result.

To avoid the early shrinkage-cracks, concrete must be protected against desiccation. This can be achieved by covering the concrete with burlap or plastic sheets or by spraying it with water.

4.5.3 Volume changes due to alterations of the substance

Chemical alterations of a substance result in volume changes, causing stresses and possibly cracks.

In concrete mainly the cement stone or the reinforcement (corrosion) are attacked while aggregates are generally more resistant.

The volume of cement stone decreases during hydration because the volume of the hydration product is about 3 to 11% smaller than the original volume of cement and water. This chemical shrinkage has to be distinguished from drying shrinkage. Chemical shrinkage enlarges the gel pores and generally does not cause cracks.

A higher risk for concrete is the expansion of the cement stone caused by the reaction of free lime, periclase and sulfate. These compositions produce unsoundness.

Subsequent reactions between cement and reactive aggregates in hardened concrete can have a deteriorating effect on concrete, especially if alkalis in the cement react with reactive silica in the aggregates.

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From the external chemical attacks the most important reactions are those with CO_2 in the air, with acids, sulfates and ammonium salts.

Table 2 shows the most important detrimental influences.

Table 2a: Effect on Concrete by Various Chemical Agents: ACIDS

Acid type	Effect on Concrete
Acetic	Disintegrates slowly
Acid waters	Natural acid waters may erode surface mortar, but usually action then stops
Carbolic	Disintegrates slowly
Humic	Depends on humus material, but may cause slow disintegration
Hydrochloride	Disintegrates
Hydrofluoric	Disintegrates
Lactic	Disintegrates slowly
Muriatic	Disintegrates
Oxalic	None
Phosphoric	Attacks surface slowly
Sulfuric	Disintegrates
Sulfurous	Disintegrates
Tannic	Disintegrates slowly

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Table 2b: Effect on Concrete by Various Chemical Agents: SALTS and ALKALIES

Salts and Alkalies (solution)	Effect on concrete
Carbonates of - Ammonia - Potassium - Sodium	None
Chlorides of - Calcium - Potassium - Sodium - Strontium	None unless concrete is alternately wet and dry with the solution
Chlorides of - Ammonia - Copper - Iron - Magnesium - Mercury - Zinc	Disintegrates slowly
Fluorides	None except ammonium fluoride
Hydroxides of - Ammonia - Calcium - Potassium - Sodium	None
Nitrates of Ammonium	Disintegrates
Nitrates of - Calcium - Potassium - Sodium	None
Potassium permanganate	None
Silicates	None
Sulfates of Ammonia	Disintegrates
Sulfates of - Aluminium - Calcium - Cobalt - Copper - Iron - Manganese - Nickel - Potassium - Sodium - Zinc	Disintegrates; however, concrete products cured in high pressure steam are highly resistant to sulfates

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Table 2c: Effect on Concrete by Various Chemical Agents: PETROLEUM and OILS

Petroleum	Effect on concrete
Heavy oils[t] - below 35 deg Baume	None
Light oils[t] - above 35 deg Baume	None - Require impervious concrete to prevent loss from penetration, and surface treatments are generally used
Benzine Gasoline Kerosene Naphtha High octane gasoline	None - Require impervious concrete to prevent loss from penetration, and surface treatments are generally used.

Table 2d: Effect on Concrete by Various Chemical Agents: COAL TAR SESTILLATES

Tar derivatives	Effect on concrete
Alizarin Anthracene Benzol Cumol	None

Steel reinforcement corrodes when it is attacked by moisture or acids. This not only diminishes the strength of the steel bars, but also damages the concrete due to an increase in volume of the corrosion products. Well compacted concrete, as an alkaline medium with pH-values above 9.5, protects steel against corrosion.

Impermeability (or low permeability) of concrete to water, other liquids and gases is very important for the protection of the steel reinforcement as well as for some concrete structures. In most cases, a well compacted concrete has a sufficient impermeability.

To protect concrete against specific attacks of aggressive water, chemicals and gases, a special surface treatment with chemically resistant coating is necessary. In some cases, a special concrete with special cements or of a special composition is required.

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Table 3: How can we improve the durability of concrete?

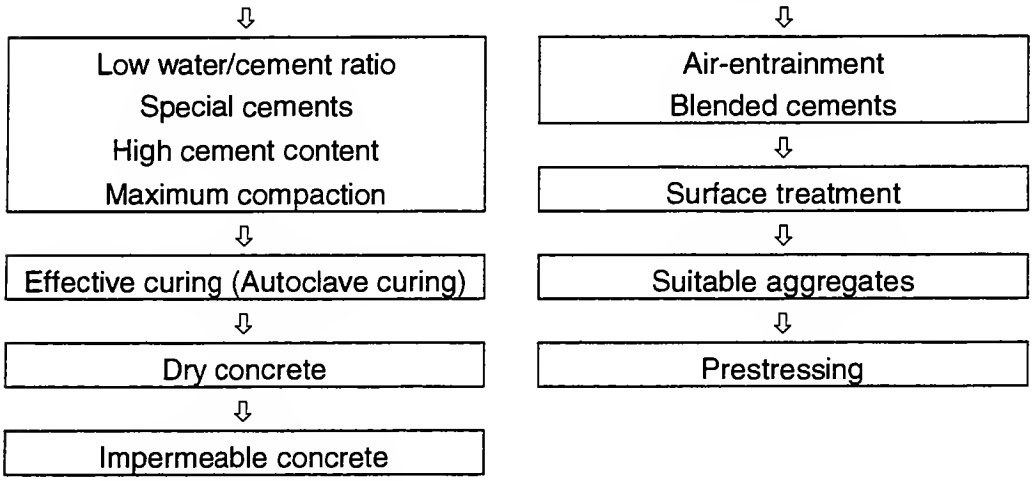
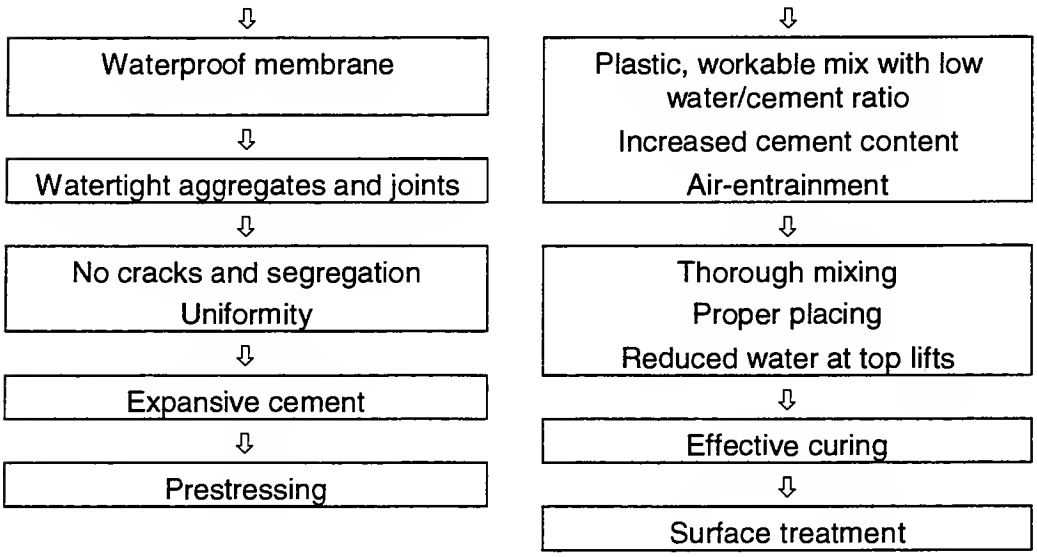


Table 4: How can we improve the impermeability of concrete?



4.6 Thermal properties

- **Thermal conductivity**
  Thermal conductivity is the ability of hardened concrete to transmit heat. It is usually measured in W/mK or kcal/mh°C. For ordinary concrete the values lie within the range of 1.30 to 2.10 W/mK (1.10 to 1.80 kcal/mh°C).

- **Specific heat**
  Specific heat is the measure of heat capacity and is defined as the amount of heat necessary to change the temperature of 1 kg concrete by 1°C. It varies between 0.84 to 1.17 kJ/kgK (0.20 to 0.28 kcal/kg°C).

- **Coefficient of thermal expansion**
  Coefficient of thermal expansion indicates either a change in volume, or usually a change in length in relation to temperature changes. The very similar coefficient of thermal expansion of concrete and steel is one of the reasons for practical utilization of steel as reinforcement in concrete. Standards and building codes of most countries prescribe the same value for steel and concrete (10^{-5}/°C).

Resistance to high temperature
Concrete, if properly designed, placed and cured, performs remarkably well with regard to

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fire, heat and blast resistance ('Concrete does not burn').
Temperatures below 250°C scarcely affect the concrete strength, but is subjected to about 300°C, there are marked signs of strength loss (Table 4a). The strength loss is considerably lower, if the aggregate does not contain silica, but is composed e.g. of limestone, basic igneous rock, and particularly of crushed brick and blast furnace slag. Low conductivity of concrete (lightweight concrete) as well as special cement (high-alumina) improves the fire resistance.

Table 5: Loss of compressive strength of ordinary concrete under continuous expoure to high temperatures

Temperature (°C)	<150	250 to 320	430 to 500	600 to 650	>820
Reduction in compressive strength (%)	negligible	15-25	40-50	75-85	disinte-grated

4.7 Aesthetic properties

In constructions where it is intentionally left exposed (without stucco), concrete should be appealing to the eye, i.e. color and texture should be decorative.

4.8 Acoustic properties

Sound protection is an important factor in many types of buildings due to their design. The concrete as a relatively heavy material (ρ = 2.2 to 2.4 g/cm^3) has generally favorable acoustic properties.

Generally it can be distinguished between:

- Sound insulation (sound conductivity) and
- Sound absorption.

Sound insulation:
For sound insulation two types of measurements and technical values are important for the building design:

- Sound transmission loss (STL) is defined as the number of decibels (dB) by which the level of airborne sound intensity is decreased when transmitted within the structure. STL of concrete walls and slabs is directly proportional to their thickness.
- Sound impact loss: the impact noise (i.e. by footsteps on the floor, appliances mounted on the wall) and their level at different distances from the origin is measured. The values are expressed also in dB.

Sound absorption:
The coefficient of sound absorption indicates the difference between the reverberation time and frequency of an empty standardized room and the same room after it has been furnished with a certain amount of absorptive material.

5. FACTORS INFLUENCING THE STRENGTH OF CONCRETE

The concrete properties are the result of a combination of many factors. Some can be attributed to the material itself, but just as important are the external influences. As has been mentioned previously, the most important and most characteristic property of concrete is its

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compressive strength after 28 days which can also serve as an indication for other properties. Therefore, this paragraph will be mainly concerned with concrete strength.

The main factors influencing the strength are (see also Fig. 10):

Strength influencing factors:

- Properties of the constituents
- Mix proportions (mix design, formula)
- Way of handling fresh concrete
- Conditions under which the concrete hardens

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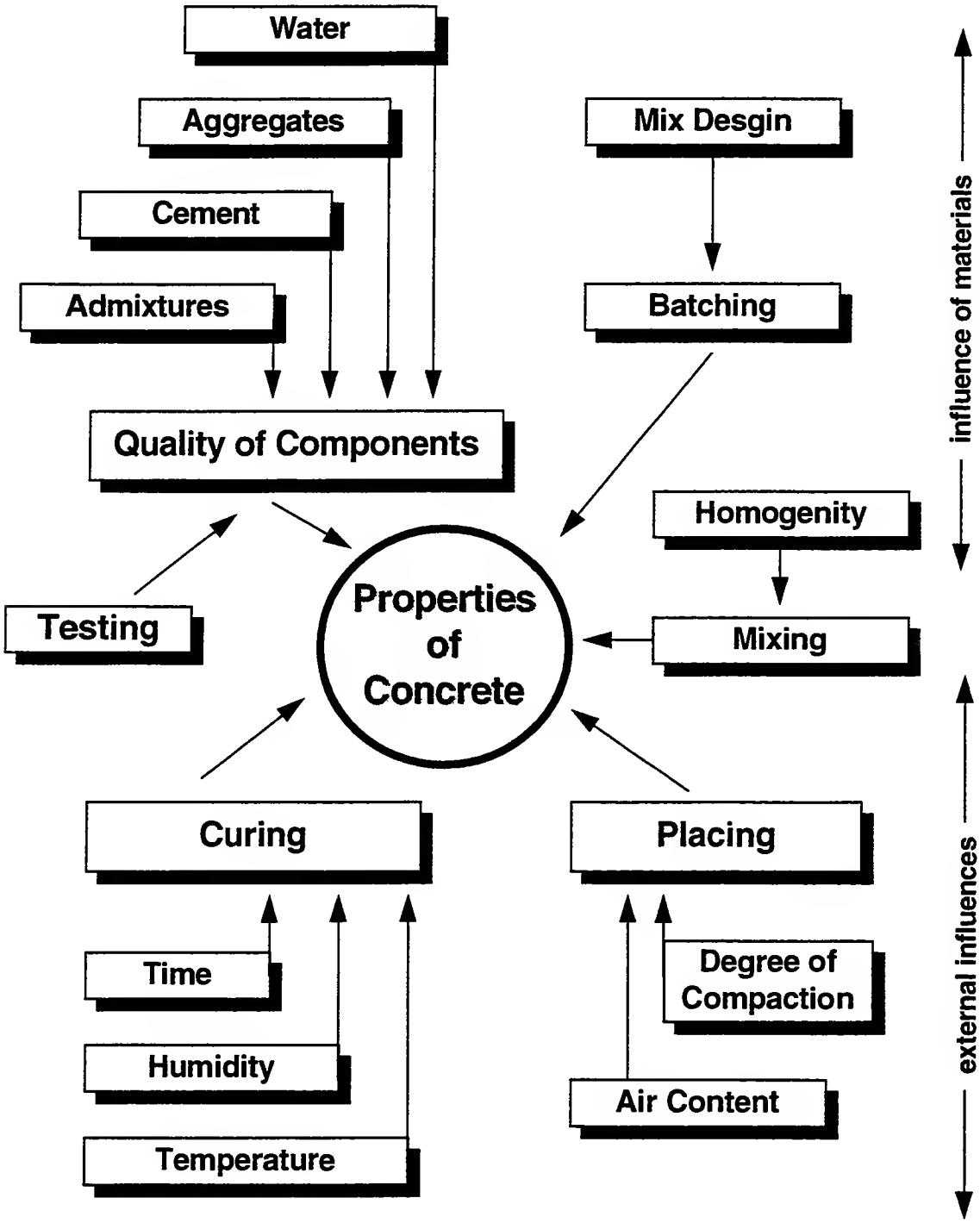
Fig. 10: Factors influencing the strength of concrete



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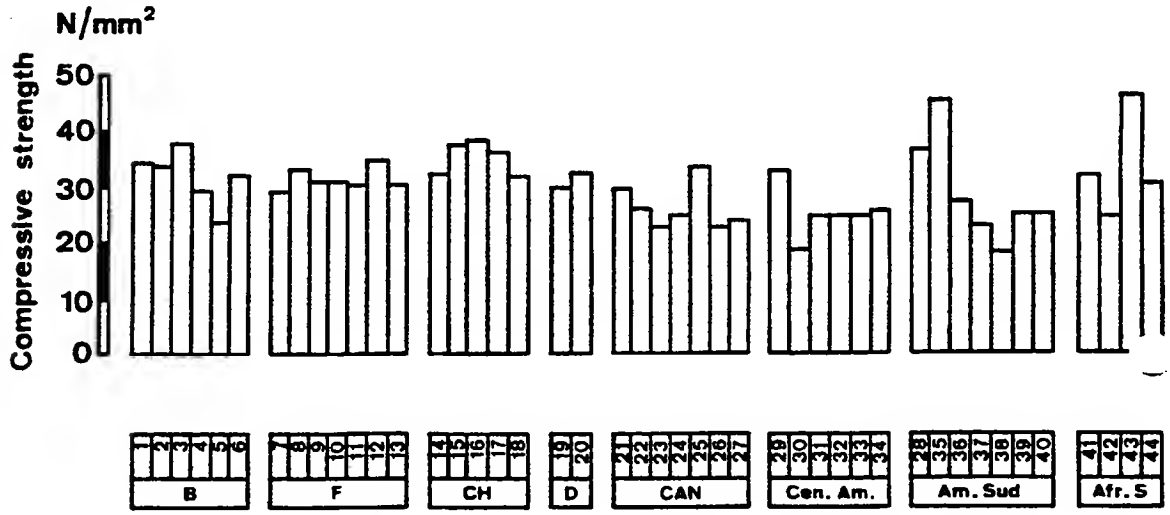
One and the same concrete may show various values when tested with different methods. Thus, the method of testing can influence the test results but not the actual properties of concrete.

The following figure gives an indication of the range in which strength values can vary; it shows the results of tests carried out on specimens of the most common cements produced in ready-mix plants which are affiliated with "HOLDERBANK".

Fig. 11: Compressive strength after 28 days of samples of 44 Ready-Mix Plants



5.1 Influence of the constituents

5.1.1 Aggregates

Aggregates represent the largest part of concrete in volume (60 to 75%) as well as in weight. Compressive strength of natural aggregates in normal weight concrete exceeds 2 to 3 times the strength of cement stone. Thus, it is the strength of the cement stone or the bond that limits the concrete strength, the cement stone being the weaker constituent.

Aggregates of various origin - if they are clean and free of deleterious substances - develop practically equal concrete strength if used with the same cement and mix formula (see following figure).

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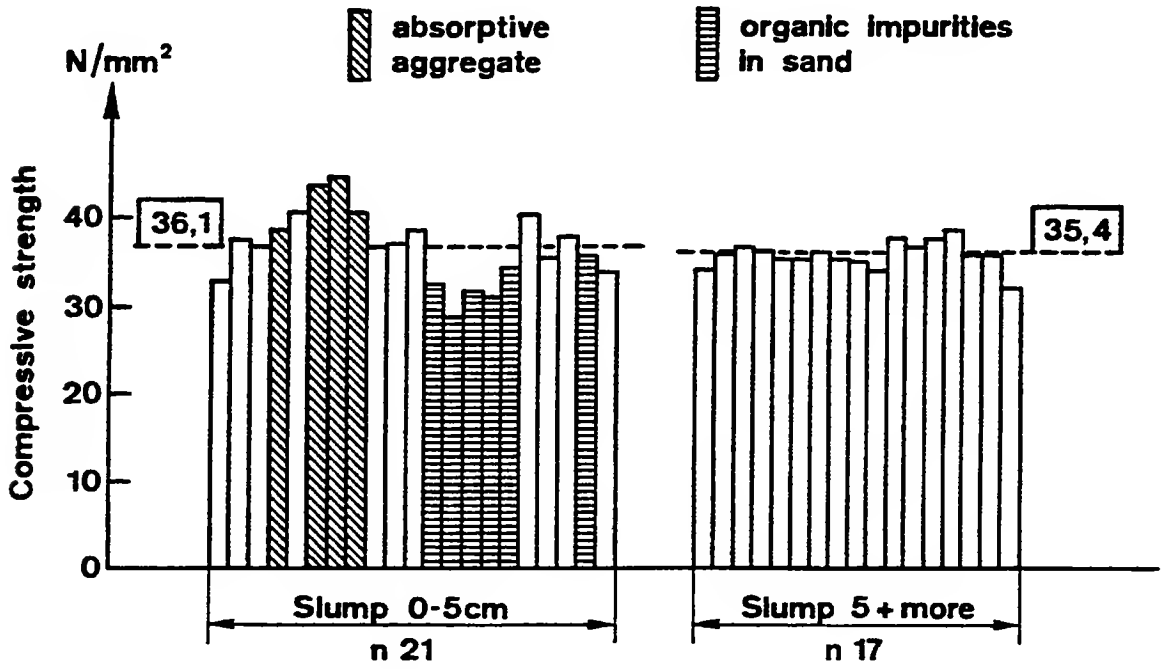
Fig. 12: Effect of aggregate on compressive strength concrete with 350 kg/m^3 cement and w/c ratio of 0.55



Note:

The strength of aggregates does not have a influence on the compressive strength of ordinary concrete if they meet basic quality requirements.

The concrete made of various aggregates with equal strength show very different consistencies This is obviously due to the grading, angularity and surface roughness of the aggregates, requiring various amounts of water to obtain concretes of the same workability. The water requirement is mostly influenced by the amount and properties of sand (fine aggregates). Grading, the maximum size and shape of the aggregates determine the volume of voids in a compacted aggregate mix as well. The percentage of void volume (varying between 20 to 40%) indicates the volume of cement paste (cement plus water) required to obtain a good concrete mix.

Note:

Grading and shape of the aggregates determine the required amount of water and cement and thus strongly influence the strength of concretes of equal consistency.

Other properties of aggregates such as: porosity, unsoundness or lack of chemical resistance affect the concrete durability.

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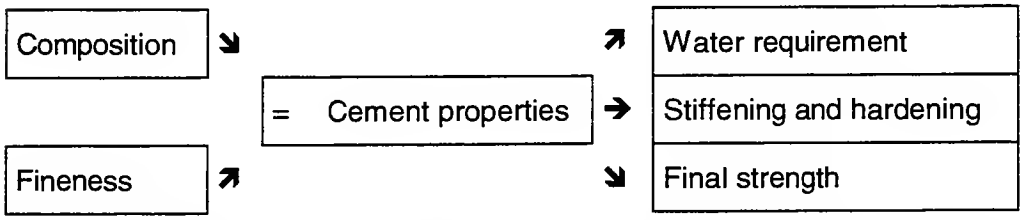
5.1.2 Water

Water represents the second largest portion of the volume of fresh concrete. The quality of water only exerts influence on concrete strength if it contains deleterious impurities.

The required water content of a concrete mix depends on the other constituents (cement, aggregate, admixture).

5.1.3 Cement

Cement has a primary and direct effect on concrete strength by its material properties:



The water requirement of cement measured on paste of normal consistency has only little influence on the water requirement of concrete (Fig. 13), much less than the aggregate.

Table 6: Water requirement of standard cement paste and concretes of different Consistency

Consist-ency	OPC Cement A		OPC Cement B		OPC Cement C	
	Water Require-ment (%)	Vebe Consist-ency (sec)	Water Require-ment (%)	Vebe Consist-ency (sec)	Water Require-ment (%)	Vebe Consist-ency (sec)
Ref.: Standard cement paste	***24***	–	***29***	–	***29.75***	–
Concrete: stiff	52	14.8	53.5	15.0		
Stiff-plastic	53.5	10.0			55	10.5
Plastic	55	5.0	57.5	4.5	58	5.6

Concluding from the strength test results on standard cement mortar, only a rough estimate of the concrete strength can be made.

With increasing standard mortar strength, concrete strength (with the same formula and aggregate mix) generally tends to increase slightly, especially after 28 days (see diagram below). The correlation between strength of concrete and mortar is poor; the ratio of both varies strongly with each individual cement. Cements with higher mortar strength can have even lower concrete strength.

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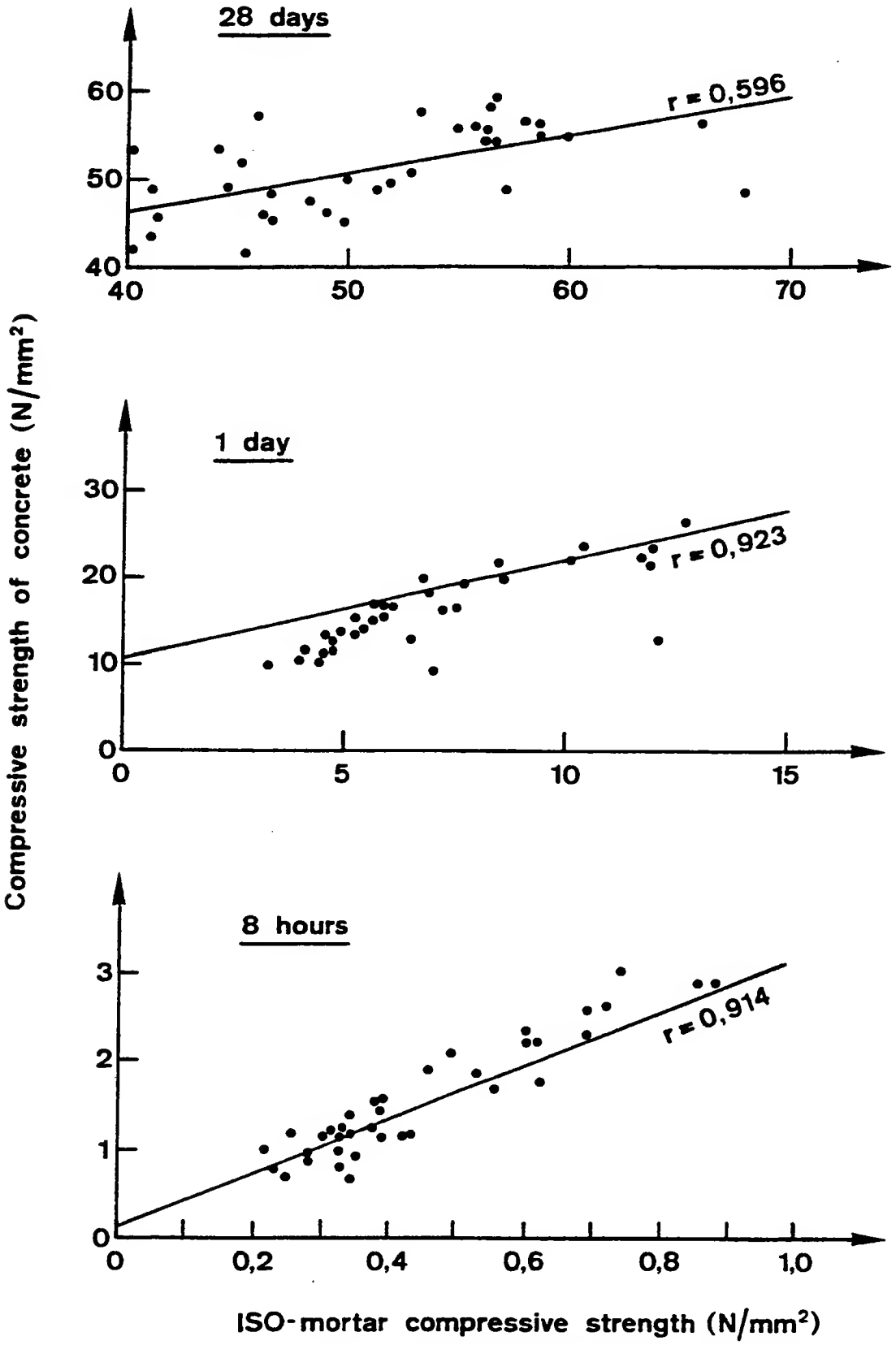
Fig. 13: Relation between concrete and standard mortar strength (determined in Concrete Laboratory of the TC-MD on 37 OPC)



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Note:

It is not possible to deduce from the standard mortar strength of various cements the strength of concretes of equal composition.

Important statement:

The ***hardening rate*** of concrete is much faster than that of standard mortar, as can be seen in the following figure. This is of utmost practical importance because on the job site under normal conditions (t = 15 to 25°C) the concrete reaches a certain percentage of its final strength much sooner than standard mortar.

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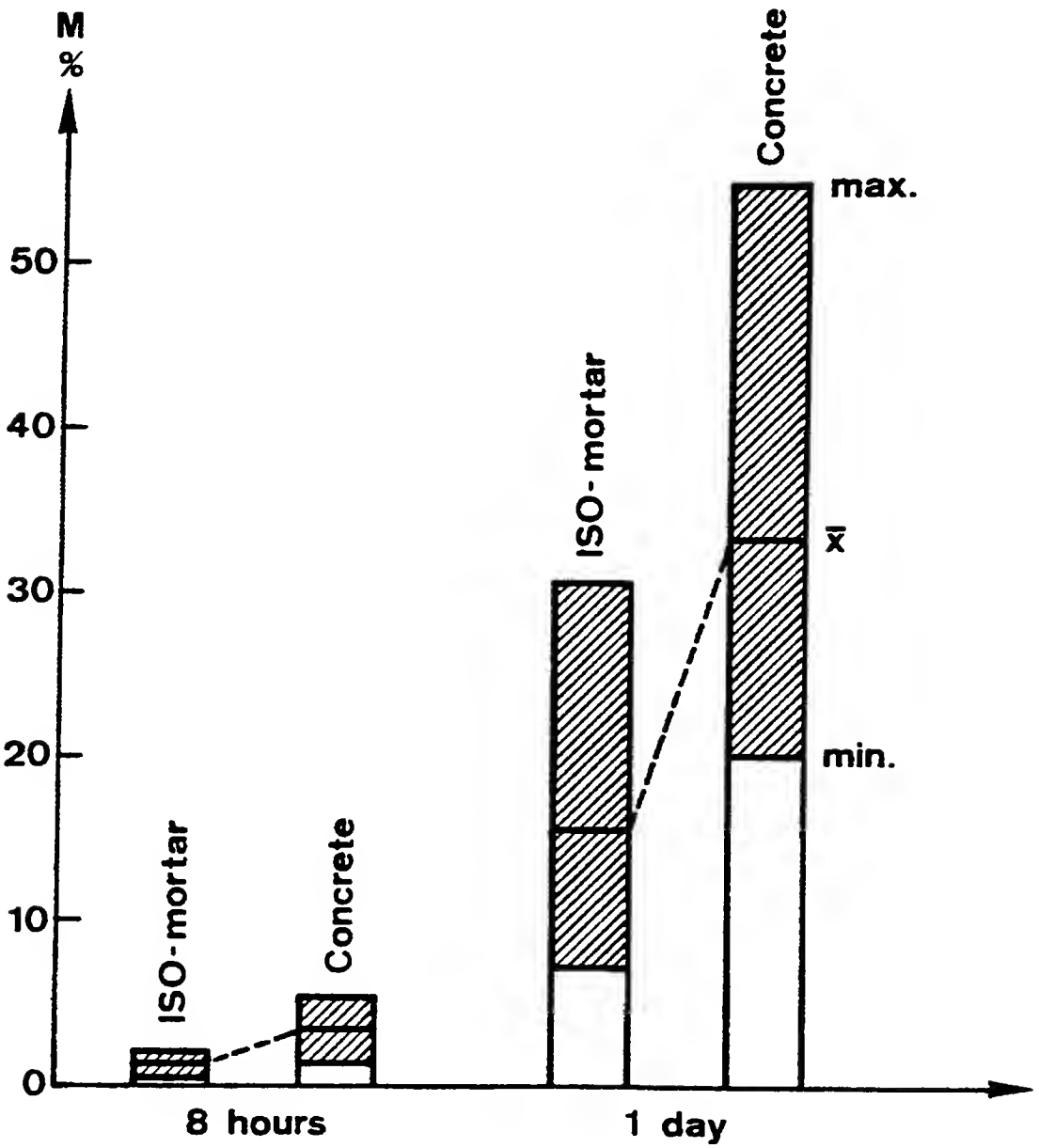
Fig. 14: Maturity Percentage of Mortar and Concrete



(Tested on 37 OPC from the «Holderbank» Group plants)

Due too different rates of hardening, the effect of cement properties on early strength is much stronger than on final strength. Finer grinding results in only slightly higher final strength but considerably higher early strength. With different cement compositions (above all blended cements) a concrete showing lower initial strength may have the same or even higher final strength than a concrete with higher initial strength. For more information regarding the influence of the cement properties, see paper **'Cement'**.

The influence of admixtures has already been described in paper **'Concrete Main Components'**.

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5.2 Influence of mix proportions

5.2.1 Water / Cement ratio

As already mentioned, the concrete strength depends essentially on the strength of the cement stone, and is only partially determined by the cement strength as measured on standard mortar. There must be other factors as well. The strength of mortar made with a given cement depends on its compaction, i.e. on the part of volume occupied by solid material and by voids (gel pores and capillaries).

The water requirement of cement for complete hydration is about 25%; another 15% is contained in the gel pores. If concrete contains less than 40% water, cement does not hydrate completely. Furthermore, since an extremely dry concrete cannot be compacted adequately, a deficiency of water causes inferior strength. For reasons regarding the workability, however, the concrete usually contains more water than the cement requires.

The following figure shows how the compaction of mortar depends on the ratio of the water content to the cement content:

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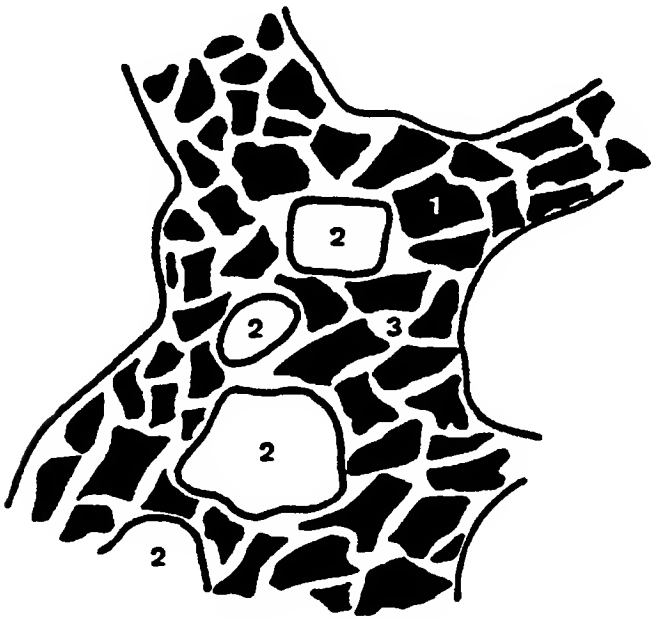
Figure 15: Structure of fresh cement mortar



Water/Cement ratio 0,5

1 Cement grain

2 Sand grain

3 Water and Air

Water/Cement ratio 0,7

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This weight proportion is usually termed **water/cement ratio (w/c ratio)**. While standard mortar has always the same prescribed w/c ratio, the water content of concrete varies according to the desired consistency and the water requirement of the aggregate and of the cement to obtain this consistency. The next two diagrams show the effect of the w/c ratio on the concrete strength.

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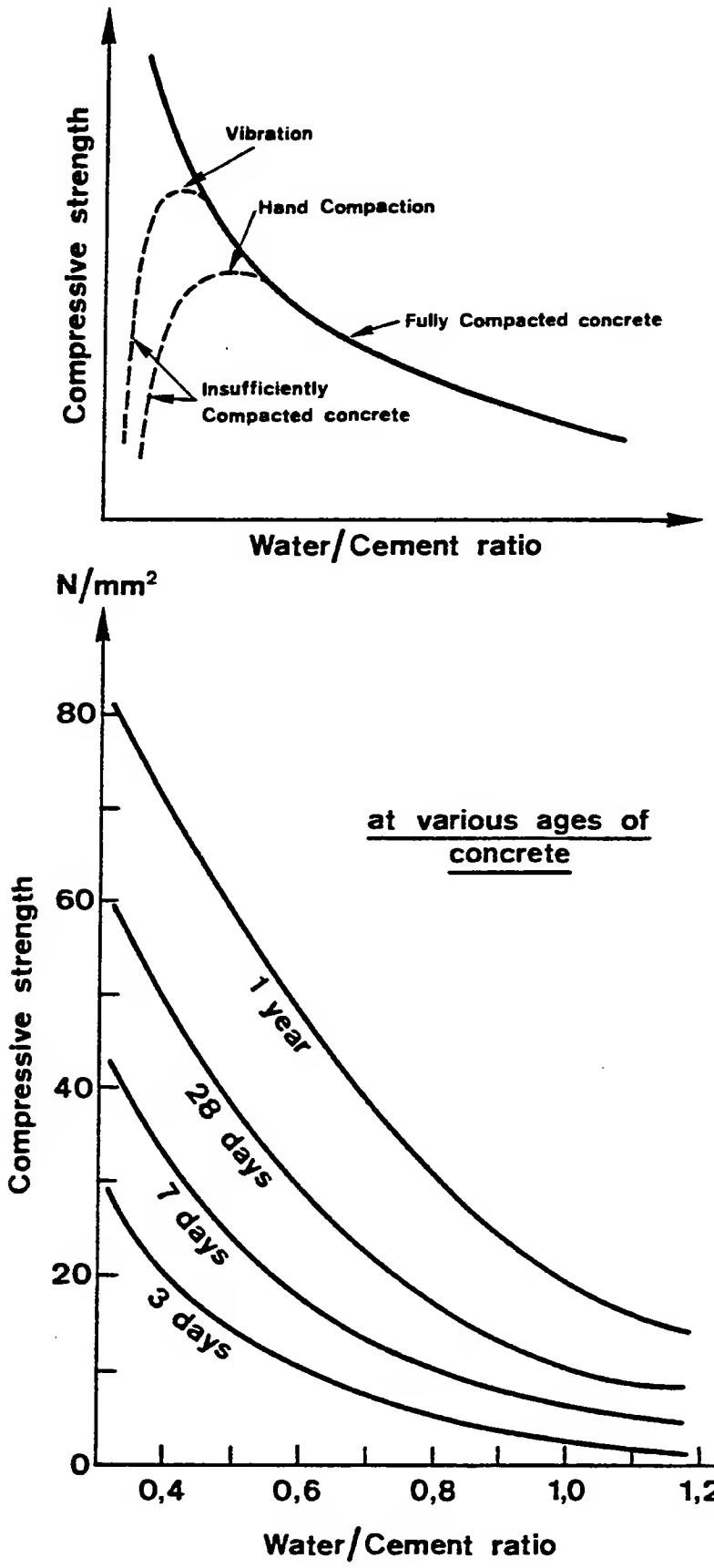
Fig. 16: Relation between compressive strength and water/cement ratio of concrete



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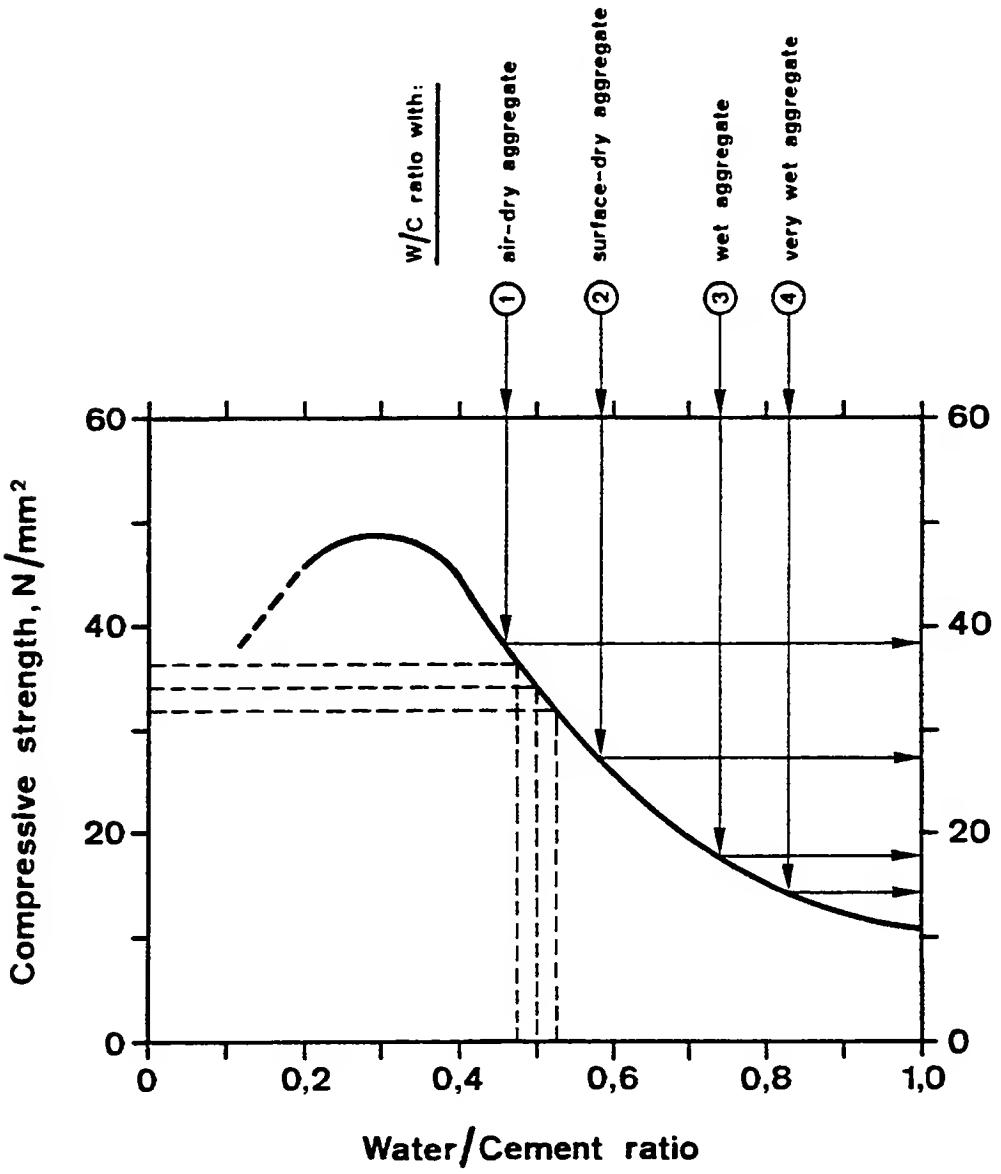
Important statement:

For the best concrete quality it is important to choose the lowest possible w/c ratio that still enables the concrete to be perfectly compacted.

In the concrete standards of different countries the maximum permissible w/c ratio for different types of structure is fixed (see tables later: Requirements for concrete acc. to DIN).

One of the most common causes of poor quality of concrete is the addition of too much water. Following figure shows how an inaccurate measurement of the quantity of mixing water or an uneven moisture content of aggregates may impair the strength of concrete. The moisture content of the aggregates must be deduced from the amount of water to be added.

Fig. 17: Influence of change in water content on compressive strength



Bolomey's Equitation

Various formula have been proposed to predict concrete strength on the basis of mortar strength and w/c ratio e.g. by Féret, Abrams, Bolomey, etc. The formula which is the simplest and most frequently applied, is the Bolomey's equation considering the reciprocal value of the w/c ratio, the cement/water ratio:

Bolomey's Equation:

$$\chi_{concrete} = \chi_{cement} \cdot \left(\frac{c}{w} - 0.5 \right) + b$$

where: $\chi_{concrete}$ = (predicted) concrete strength

χ_{cement} = (observed) mortar strength

a, b = constants depending on the mortar and concrete testing method, age and curing specimen, aggregate quality; cement content and cement characteristics not included in the χ_{cement} etc.

Based on the coefficient of correlation of 0.918 it can be assumed that effects other than the cement/water ratio and mortar strength have a negligible influence on the 28 day concrete strength (see following figure). To predict the early strength, the equation is less suitable since the relation between mortar and concrete strength is influenced also by the faster hardening rate of concrete in comparison to mortar.

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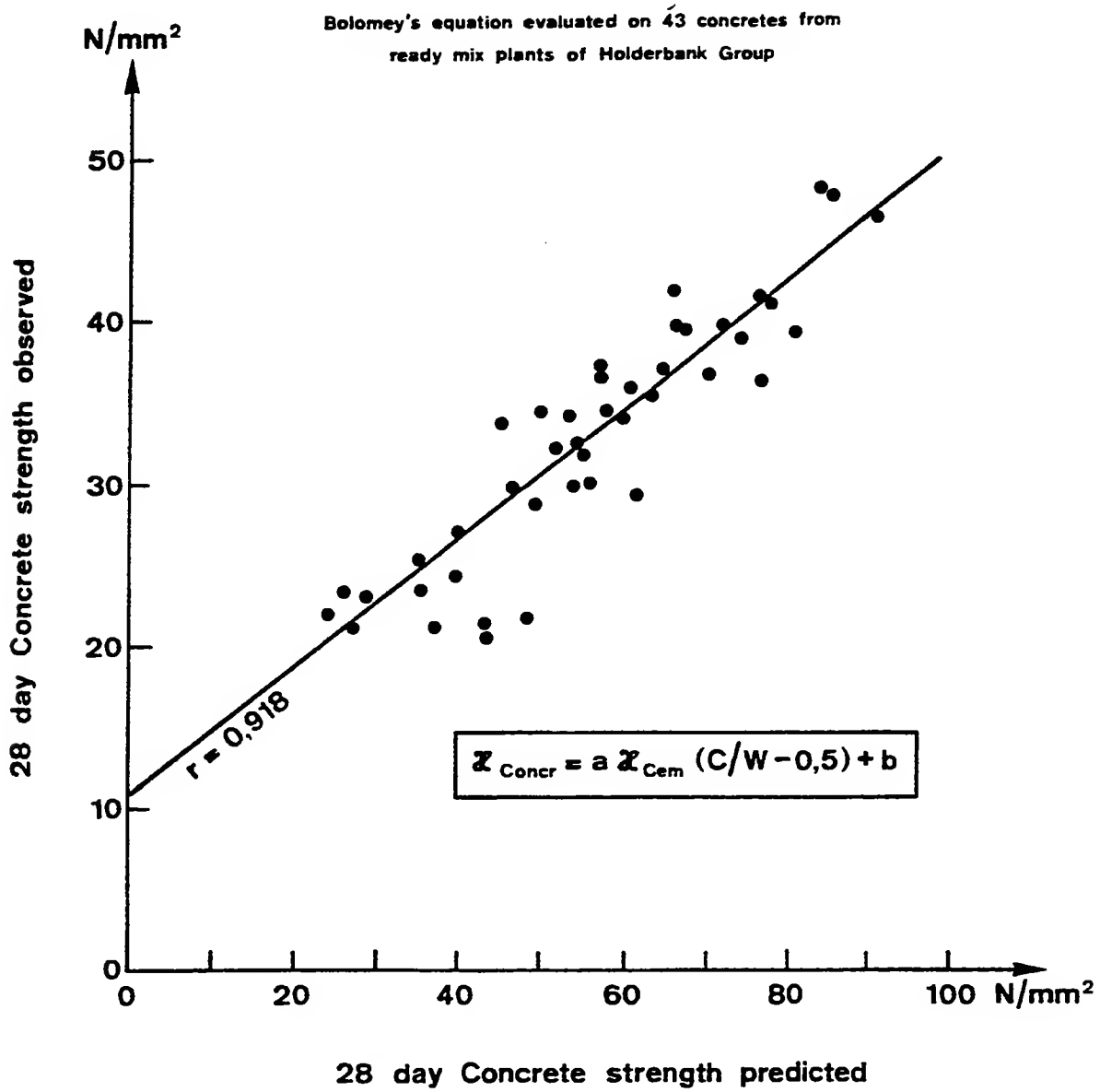
Fig. 18: Relation between calculated and observed concrete strength



The calculated concrete strength is only obtained, however, if the concrete is perfectly compacted (porosity < 1%).

5.2.2 Cement content

The content of cement or cement paste in concrete must be sufficient to envelop the aggregate grains in order to reduce their friction, to glue them, to fill the voids between them and also to protect the reinforcement against corrosion. Thus, an insufficient amount of cement impairs the workability and compaction of concrete and as a consequence, its strength and durability.

Note:

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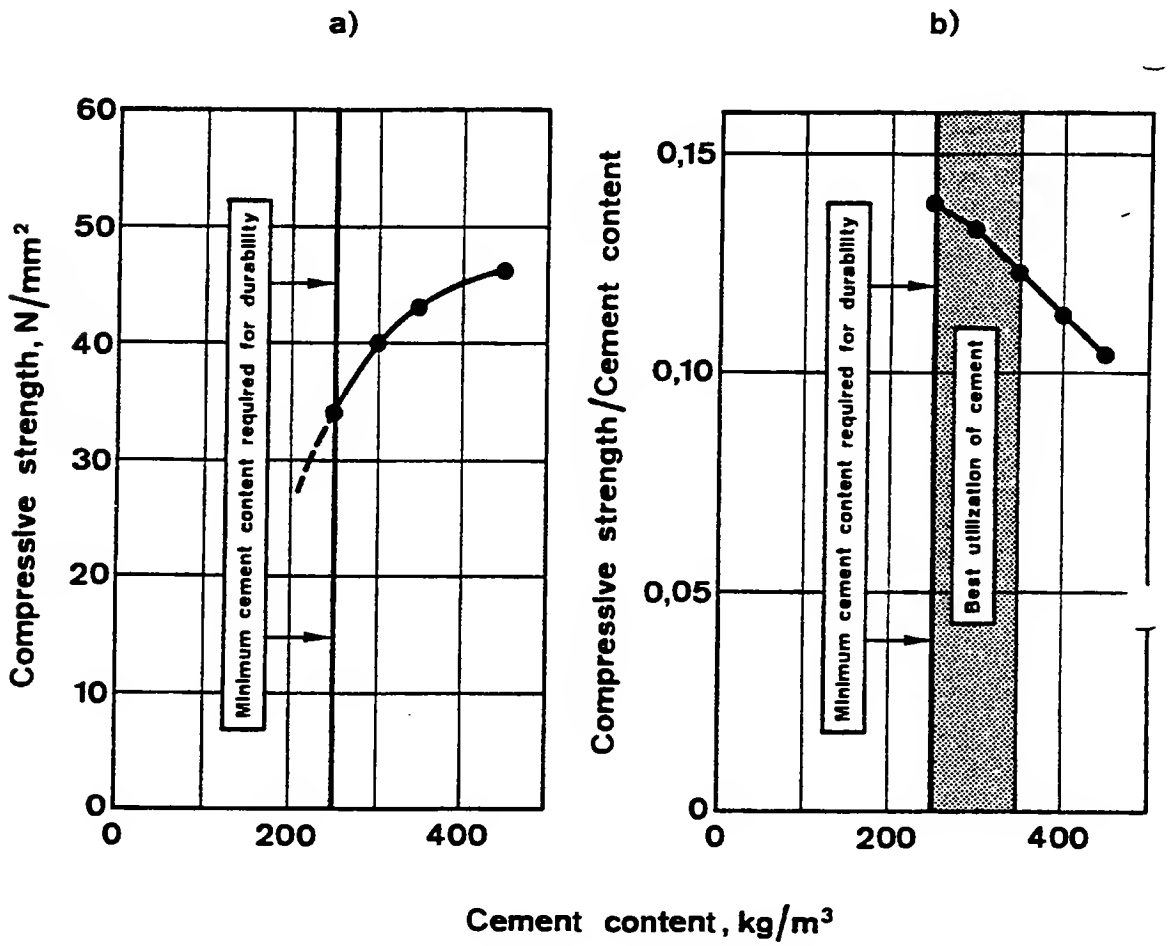
The cement dosage should not be below a certain minimum (usually between 250 to 350 kg/m^3 depending on the purpose of the construction) in order to guarantee impermeability and durability, even though less cement would be sufficient for the needed strength.

After all voids are filled, a surplus of cement paste does not increase the concrete strength any further since cement stone has a lower strength than aggregate (Fig. 20a). By increasing the cement content above an optimum, the ratio between compressive strength and cement content decreases (see figure). If the cement content in concrete is sufficient for full compaction, the strength of fully compacted concrete is given by the strength of the cement stone. It depends only on its w/c ratio and on the standard cement strength.

Fig. 19: Utilization of cement in concrete at constant workability



5.2.3 Aggregate/Cement ratio

The optimum **'aggregate/cement'** ratio is not only considered from a technical but also from an economical point of view since aggregate is stronger and cheaper than cement. Thus, the grading of aggregate is most important, not only for workability but also for the minimum volume of voids. It makes it possible to obtain the required strength with less cement.

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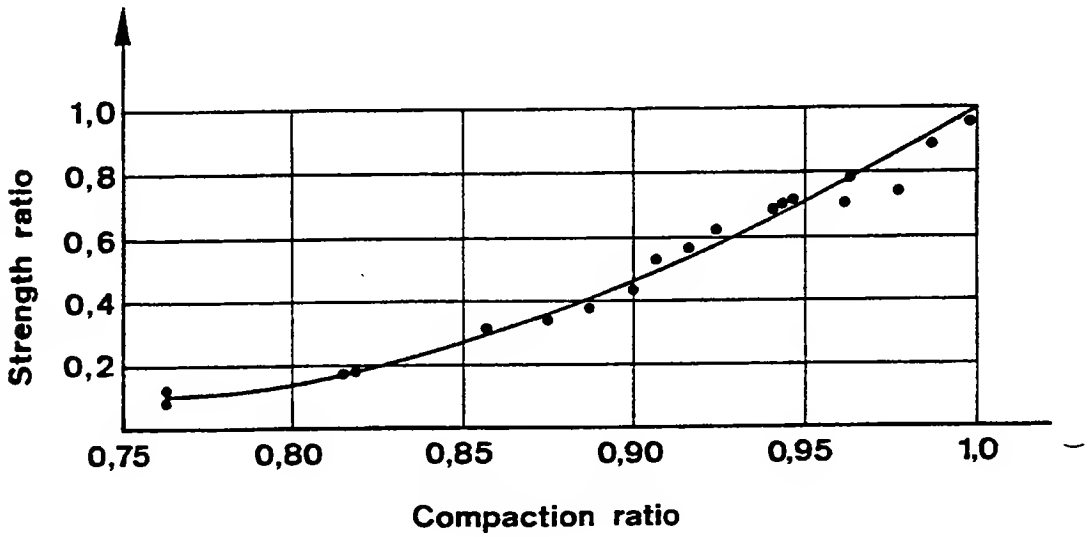
5.3 Influence of handling

It is most important that the consistency of concrete permits full compaction with the tools that are available on the job site. The need for compaction becomes apparent in the following figure:

Fig. 20: Relation between strength and compaction



Another prerequisite for the optimum utilization of the material properties is the homogeneity of the mix. To maintain uniformity within one batch and between several batches, accurate weighing, thorough mixing and careful transporting and placing is necessary. These measures also prevent segregation.

5.4 Influences of curing

5.4.1 Moisture

After placing, the subsequent conditions strongly influence the development of the concrete properties. As for the hydration of cement water is required, loss of moisture must be prevented or even additional moisture supplied. The next figure shows how concrete strength is impaired by the lack of moisture, especially during the first 3 days.

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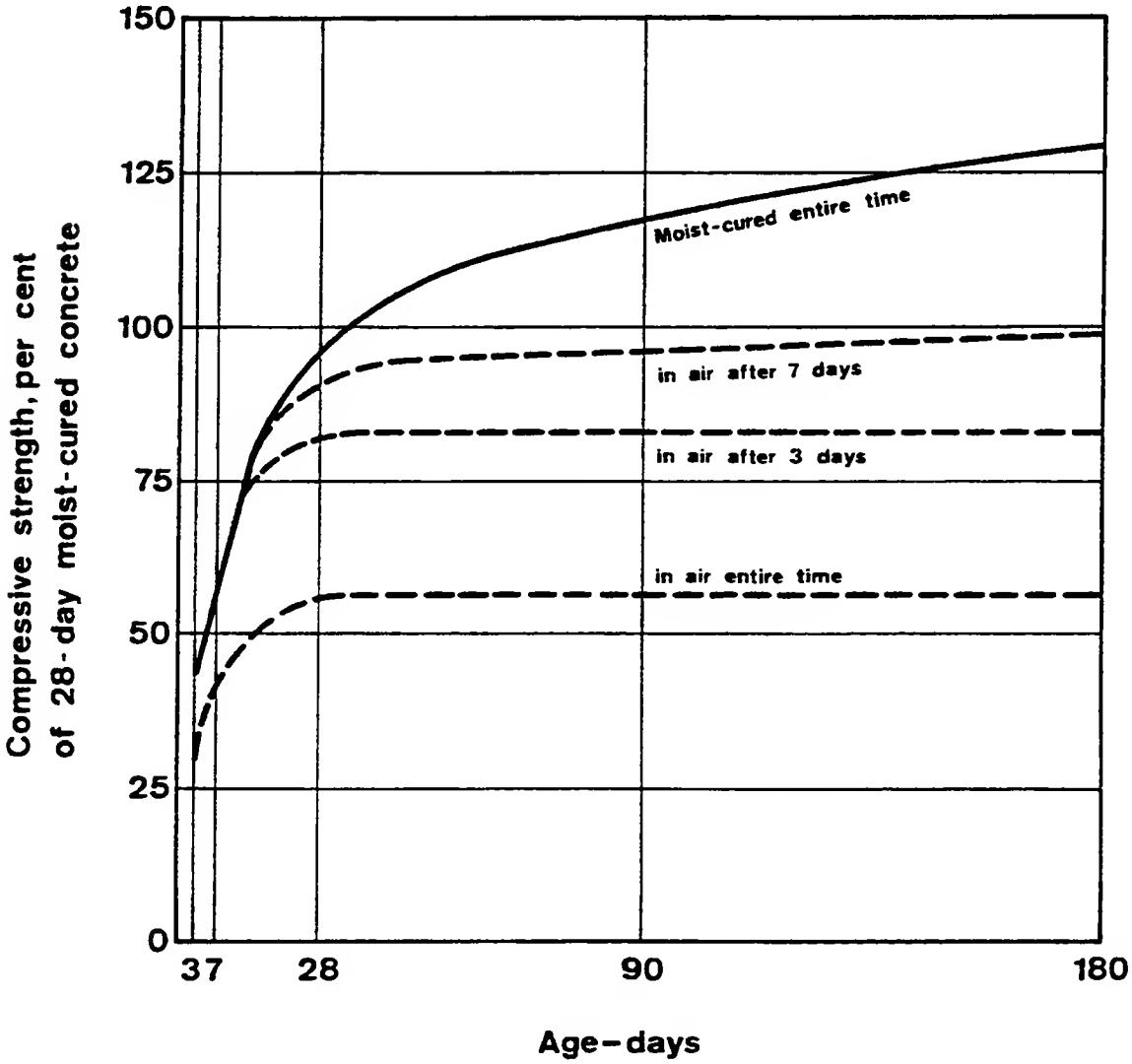
Fig. 21: Strength of concrete increase as long as moisture is present forhydration of cement



Concrete that hardens submerged in water has the highest strength. Once it is hardened, dry concrete has higher strength than moist concrete.

5.4.2 Temperature

Higher temperatures accelerate setting and hardening of concrete, like other chemical processes(see following figures). Seasonal and even daily temperature fluctuations alter the hardening characteristics of concrete.

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Fig. 22: Effect of temperatures on the compressive strength of concrete at various ages (T ≥ 20 °C)

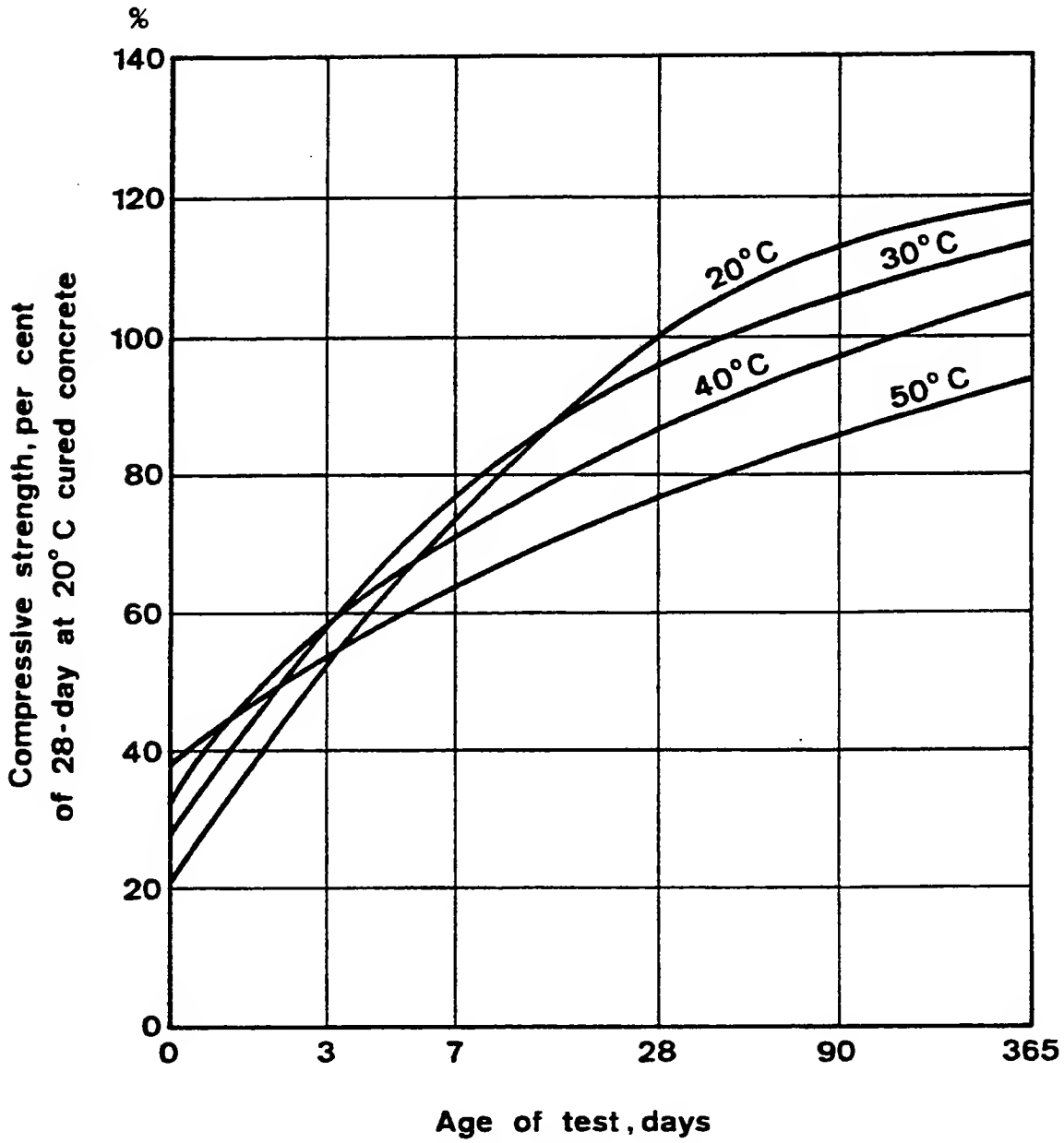
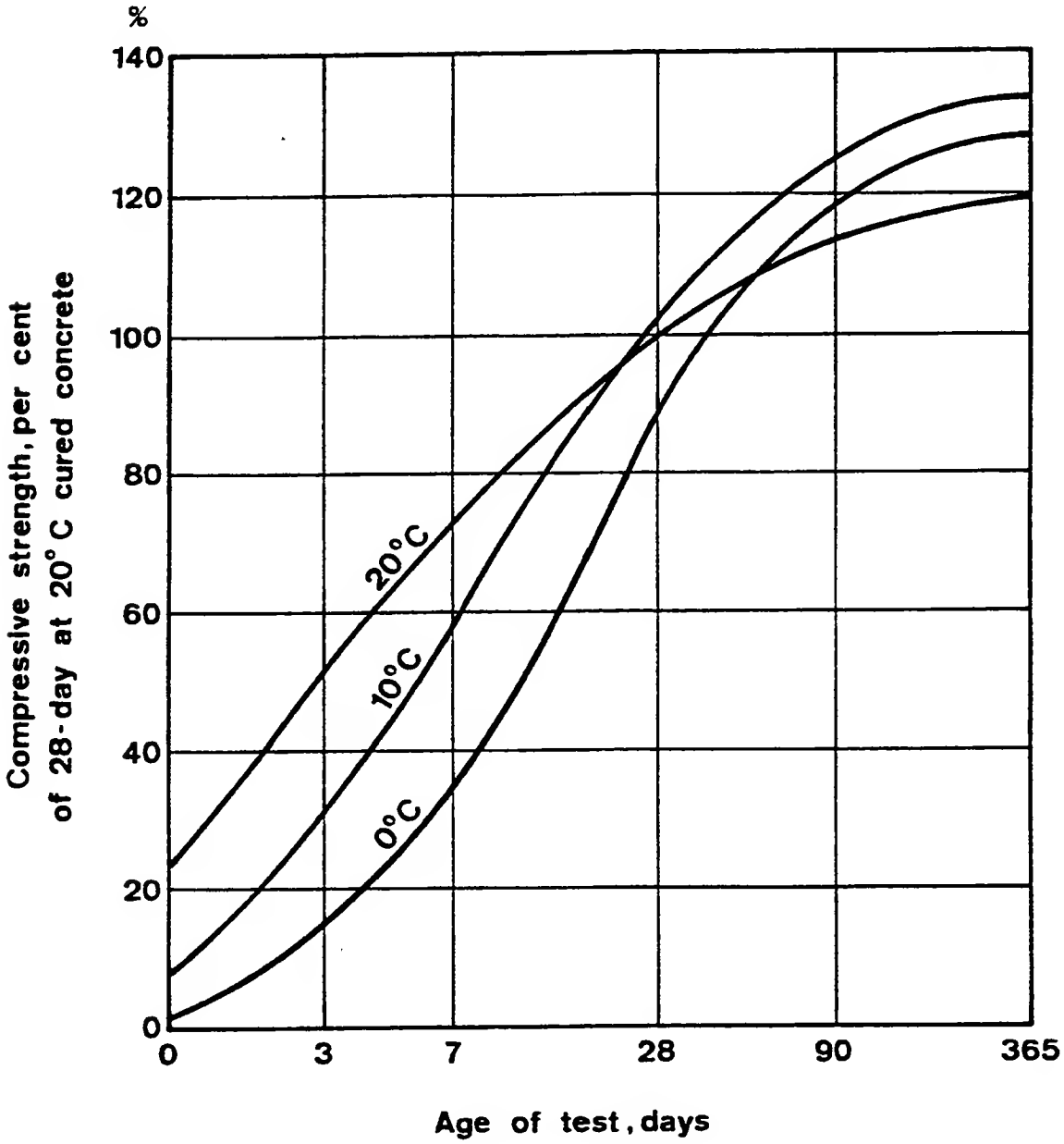


Fig. 23: Effect of temperatures on the compressive strength of concrete at various ages (T ≤ 20 °C)



Note:

- The temperature during the first 24 hours approximately strongly affects the initial strength development of concrete and it predetermines the later hardening and even the final strength. Thus, it is futile to maintain high curing temperatures after the first 24 hours.
- Concrete cured at low temperature achieves a slightly higher final strength than the concrete cured at 30 °C or more and vice-versa.

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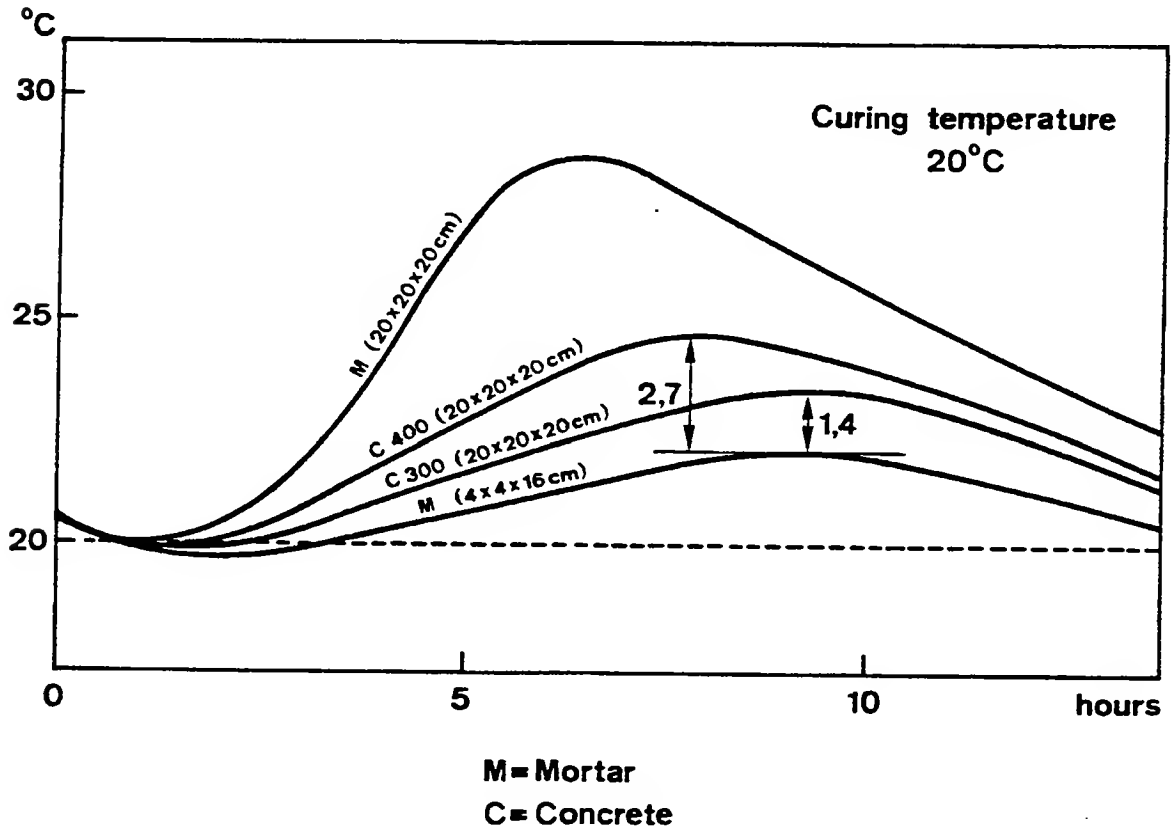
Accelerated hardening obviously creates structures of hydration products that are less favorable for later hardening.

The effects of temperature depend not only on external curing conditions, but also on the temperature within the concrete created by accumulation of hydration heat.

The development of concrete temperature depends on cement properties, the composition of concrete and the size and shape of the concrete element. This fact is illustrated the following figure which compares the temperatures of mortar and concrete and of prisms and cubes. If concrete is protected against heat-losses and 'autocuring' takes place; some mass-concrete, the internal temperature of which may rise up to 90 °C, has to be cooled to prevent cracks caused by internal stress.

Figure 24: Temperature of mortar and concrete specimen



M = Mortar
C = Concrete

It is important to note that sooner or later the temperature or curing effects may activate the potential for hardening of various cements; the potential as such, however, cannot be increased by these measures.

5.5 Influences of testing methods

The influence of the testing method can sometimes be decisive for the final test result.

Specimen size, moluding, curing and testing procedure are some of the influencing factors. The above mentioned figure demonstrates how test results of temperature evolution measurements depend on specimen size and concrete composition. The differences between results on test specimens and concrete in construction can be very significant. Thus, results of specimen tests are never identical to those obtained from tests on concrete in situ. They merely give some indications. (See separate paper: 'Testing'.)

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6. FACTORS INFLUENCING THE DURABILITY OF CONCRETE

In paragraph before the most important influences on concrete strength were discussed. The sum of other important characteristics of concrete in use, expressed as durability, may be related to strength in a general way, but it also affected by factors not significantly associated with strength.

Concrete must be able to endure those exposures which may deprive it of its serviceability - freezing and thawing; becoming wet and dry; heating and cooling; chemical attack by de-icing agents, sea water, salts and other substances.

Resistance to some of these may be enhanced by the application of certain ingredients or measures:

- **Alkali-Aggregate Reaction:**
  Low alkali cement, pozzolans or selected aggregate to prevent harmful expansion due to the alkali-aggregate reaction which occurs in some areas when concrete is exposed to a moist environment.
- **Sulphates:**
  Sulfate resisting cement or pozzolanas for concrete exposed to sea water or sulfate-bearing soils.
- **Abrasion:**
  Aggregate with a minimum of soft particles where resistance to surface abrasion is required.
- **Aggressive agents:**
  Use of a low water/cement ratio will prolong the life of concrete in practically every exposure-condition by reducing the penetration of aggressive liquids and gases. The water permeability also depends on the w/c ratio.
- **Freezing and thawing:**
  Resistance to severe weathering, particularly to freezing and thawing, and to salts used to remove ice, is greatly improved by the incorporation of a proper distribution of entrained air.

7. CONCLUSIONS

There are many factors which influence the properties of hardened concrete. The next figure summarizes the effect of these various factors.

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Fig. 25 Factors Influencing the Properties of Concrete

PROPERTIES \ FACTORS	Cement	Aggregate	Admixture	Mix ratio	Placing	Curing	Surface treatment	Construction design
Early strength	●	•	⊙	●	●	●	–	–
Final strength	⊙	•	•	⊙	●	⊙	–	–
Shrinkage	•	·	⊙	⊙	•	●	•	–
Cracking	•	•	⊙	⊙	⊙	●	⊙	⊙
Setting	●	–	●	⊙	·	⊙	–	–
Workability	•	⊙	⊙	●	–	–	–	–
Bleeding	●	●	⊙	⊙	•	–	–	–
Abrasion	·	⊙	•	●	⊙	⊙	⊙	•
Water permeability	·	⊙	⊙	●	●	⊙	⊙	⊙
Freeze thaw resistance	·	⊙	●	●	⊙	⊙	⊙	⊙
Corrosion resistance	⊙	·	•	●	⊙	⊙	●	⊙
Heat of hardening	⊙	–	•	⊙	–	⊙	·	●
Surface appearance	•	⊙	•	⊙	●	⊙	●	⊙
Heat conductivity	–	●	–	⊙	⊙	–	–	•

– no effect
· very small effect
• small effect
⊙ medium effect
⊙ strong effect
● very strong effect

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Chapter 9

Concrete Mix Design

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CONCRETE MIX DESIGN

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1. GENERAL

Mix design is a process that consists of two interrelated steps:

(1) Selection of the suitable ingredients (cement, aggregate, water and admixture) of concrete and

(2) Determination of their relative quantities to produce as economically as possible concrete of appropriate workability, strength and durability

Although many concrete properties are important, most design procedures are aimed at achieving a specified compressive strength at a given workability; it is assumed that if this is done, the other properties will also be satisfactory. An exception represent the resistance to freeze-thaw and other durability problems (for instance sulphate resistance), which require special attention in the mix design.

2. BASIC DESIGN CONSIDERATIONS

2.1 Costs

The cost of concreting is made up of material costs, plant and labour expenses. Except for some special concretes, the costs of labour and equipment are, however, to a large extent independent of the type and quality of concrete produced. It is therefore the material costs that are most important in determining the relative costs of different mix designs. Since cement is much more expensive than aggregate (see Table 1), it is clear that minimising the cement content is the most important single factor in reducing concrete costs (see also Figure 1).

Table 1: Prices of concrete constituents in Switzerland per ton (approx.)

Concrete constituent	Price (CHF)
OPC cement silo	~~110.--~~ 150
OPC cement bag	125.-- 180
Sand fraction 0/4 mm	20.20
Gravel fraction 4/8 mm	16.90
Gravel fraction 8/16 mm	12.--
Gravel fraction 16/32 mm	9.40

To economise on material costs, the proportioning should minimise the cement content without scarifying concrete quality. Since the quality depends primarily upon the w/c ratio, the water content should be reduced to lower the cement content. Some of the steps to minimise water and cement contents are to use:

- the stiffest practicable mixture
- the largest possible maximum size of aggregate
- the optimum ratio of fine and coarse aggregates

The cement reduction is, however, often restricted by specifications stating the minimum cement content (see Table 2).

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Table 2: Minimum cement content in concrete for different national standards (kg/m^3)

Type of concrete	Germany DIN 1045	Switzerland SIA 162	France* NF P-18-305	USA	England
Plain concrete	≥ 100	--	--	--	--
Reinforced/pre-stressed concrete	280 (OPC 25) 240 (OPC 35)	300	for class 250 and higher: $(250 + B)/D^{1/5}$	none	none

* B = compr. strength in bar; D = max. aggregate size

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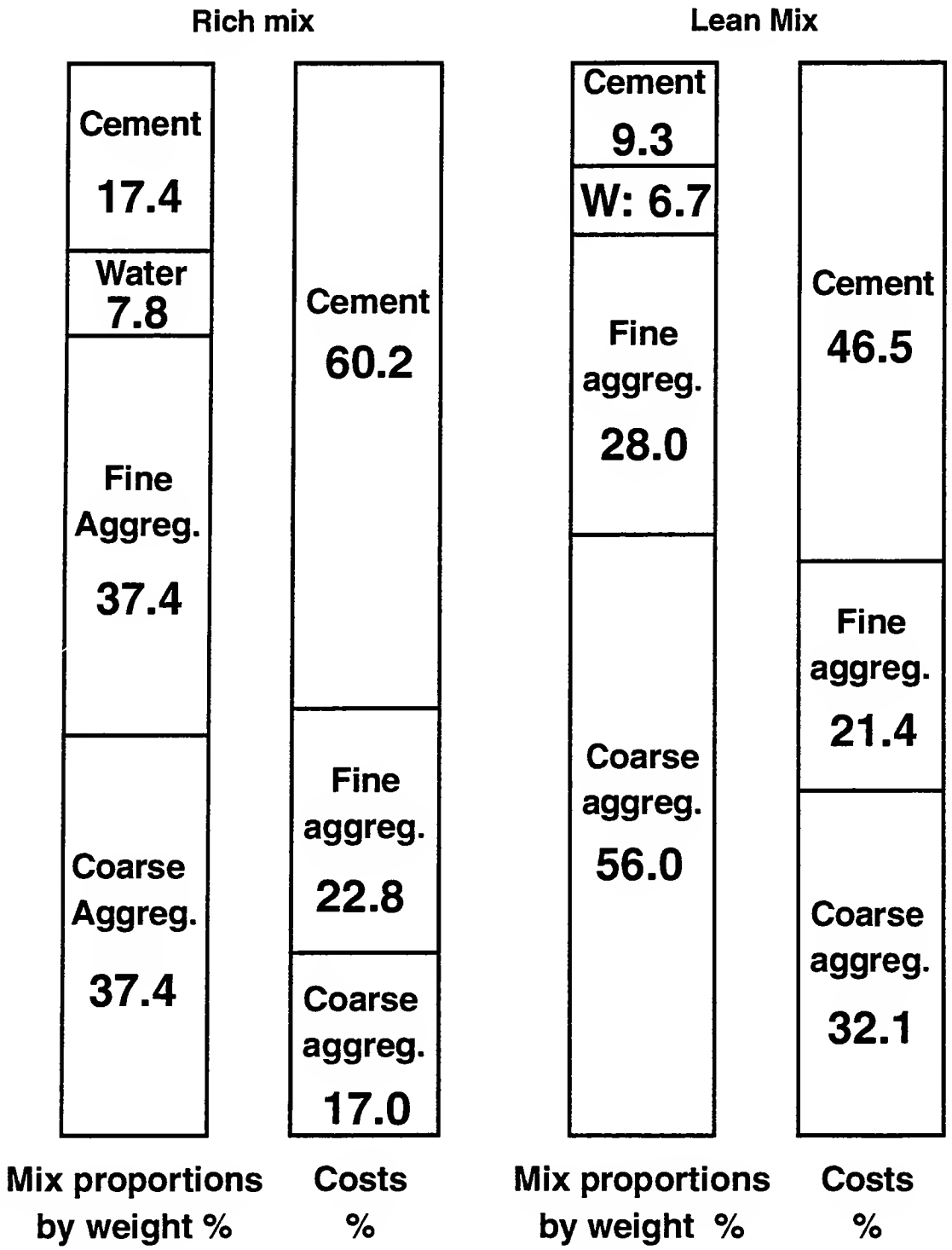
Figure 1: Costs of concrete: material costs



It should be noted here that in addition to cost, there are other benefits by using a low cement content; shrinkage will in general be reduced and there will be less heat of hydration. However, if the cement contents are too low, they will diminish the early strength

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and durability of concrete and will make uniformity of the concrete a more critical consideration.

Besides the cement reduction as such, there is also an interesting potential to reduce costs through the replacement of cement by mineral components (e.g. fly ash or ground blast furnace slag). Prerequisite is of course that mineral components of good quality are available at the concrete plant at convenient prices.

To determine the most economical mix proportions, the relative costs of fine and coarse aggregates should also be taken into consideration. Since admixtures to reduce the water requirement will increase material costs, it should be assessed in each case whether their use is justified by the savings in labour costs and cement cost eventually.

The economy of a particular mix design should also be related to the degree of quality control that can be expected on a job. At least on small jobs, it may be cheaper to overdesign the concrete than to provide extensive quality control that would be required with a more cost-efficient concrete.

2.2 Workability

Clearly, a properly designed mix must be capable of being placed and compacted properly with the equipment available. Finishability must be adequate, and segregation and bleeding should be minimised. As a general rule, the concrete should be supplied at the minimum workability that will permit adequate placement. The water requirement for workability depends mostly on the characteristics of the aggregate rather than of those of the cement. Where necessary, workability should be improved by increasing the mortar content rather than by simply adding more water or more fine material. Thus, co-operation between the mix designer and the contractor is essential to ensure a good concrete mix. In some cases, a less economical mix may be the best solution.

2.3 Strength and durability

In general, concrete specifications will require a minimum compressive strength. They may also impose limitations on the permissible w/c-ratios and minimum cement contents. It is important to ensure that these requirements are not mutually incompatible. It is not necessarily the 28 day strength that is most important; strength at other ages may control the design.

Specifications may also require that the concrete meet certain durability requirements, such as resistance to freezing and thawing, or chemical attack. These considerations may provide further limitations on the w/c-ratio or cement content and in addition may require the use of admixtures.

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3. SPECIFICATIONS FOR CONCRETE MIXES

There are three systems that can be applied to specify structural concrete. These differ with respect to the basis of specification, responsibility for the mix design and the parameter by which the concrete is judged for compliance, as indicated in Table 3.

Table 3: Types of concrete mixes and their method of specification

	Type of mix	Designed mix	Codified mix	Prescribed mix
Design stage	Basis of specification	Strength	Mix Proportions	Mix Proportions
	Strength classes	all	≤ 25 N/mm^2	all
Pro-duction stage	Responsibility for mix design	Producer	National code or specification	Purchaser
	Permitted materials	Any materials complying with National Standards	Restricted range of materials complying with Nat. Standards	any
	Strength testing for information	Basis of mix	Not usually necessary	Desirable especially for higher strength classes
Compli-ance stage	Basic parameters of concrete for compliance testing	Strength	Mix proportions	Mix proportions

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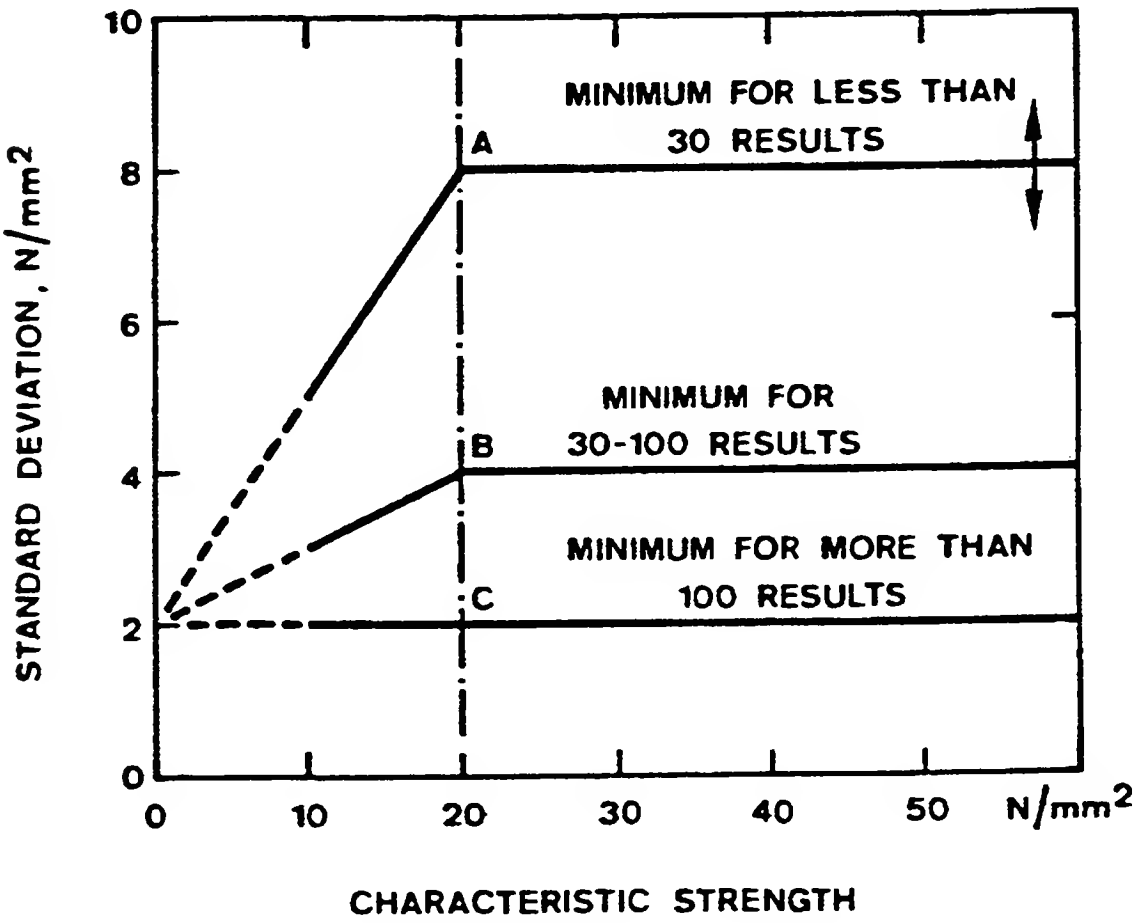
Designed mixes

The mix should be designed to have an adequate margin of strength between the specified characteristic strength or the designed mean strength based on the standard deviation obtained through experience. If there are no previous data, or if there are less than 30 results obtained under equal conditions (same plant, source of materials, and supervision), the proposed minimum value for standard deviation as per curve A (Figure 2) should be considered.

Figure 2: Proposed minimum values for the standard deviation



If there are between 30 and 100 results, the standard deviation should be the value obtained but not less than that given by curve B, i.e. for characteristic strength equal to or greater than 20 N/mm^2, the minimum standard deviation is 4 N/mm^2. For more than 100 results curve C gives the minimum standard deviation.

These values of standard deviation will be used for the calculation of characteristic strength.

Designed mixes are the most common types of mixes used in concrete practice.

Codified mixes

Codified mixes are recommended in National Standards or Codes and are based on the characteristics of the cement and aggregates available in each country. The adequacy of the structure is generally assured since the cement content will usually be greater than that of the corresponding designed mixes.

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Codified mixes are restricted to the lower strength classes of concrete and may be made only with restricted types of materials. They are sometimes used as approximate guidelines for designed mixes.

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Prescribed mixes

Prescribed mixes are applied when the purchaser wishes to specify the mix proportions to be used. There are a number of circumstances which call for such mixes; for example, if the purchaser knows that with local materials certain mix proportions will produce concrete of the required proportions, or if he requests the use of specific mixes because he wants a concrete with special characteristics (high or low density, etc.). Since the purchaser designs the mix, it is also his responsibility to ensure that the strength and other requirements for the safety of the construction are met.

4. PROCESS OF MIX DESIGN FOR NORMAL-WEIGHT CONCRETE

Before a concrete mix can be designed and proportioned, certain points of information should be known:

1) Size and shape of structural members
2) Spacing and diameter of reinforcement
3) Required strength
4) Exposure conditions (requirements for durability and chemical attack)
5) Placing and compaction methods
6) Basic material quality data on cement and aggregate

It is up to the design engineer and the architect to provide the information for points 1), 2) and 3). The information for points 5) and 6) can be obtained from the contractor firm and material supplier. As to point 4), the requirements for exposure conditions are normally specified in the standards.

The basic factors that have to be considered in determining the mix proportions are represented schematically in Figure 3.

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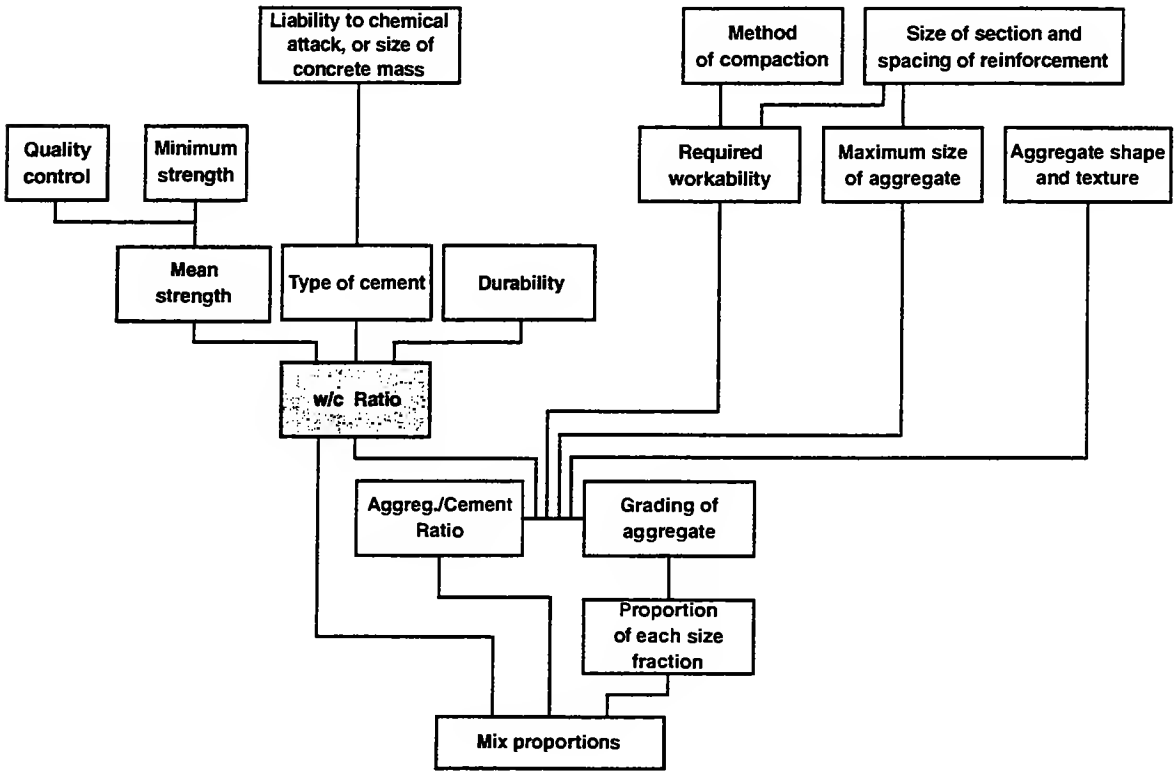
Figure 3: Basic factors in the process of mix design



The American Concrete Institute ACI recommends a maximum permissible w/c ratio for different types of structures and degree of exposure. The European Norm EN 206 for concrete describes as well the maximum permissible w/c ratio, minimum cement content, the minimum air content and range of aggregate grading for the different exposure conditions.

It should be explained that a design in the strict sense of the word is not possible: many of the material properties and effects in the concreting process cannot be truly assessed quantitatively, so that we are really making no more than an intelligent guess at the optimum combinations of the ingredients on the basis of the relationships established empirically. Therefore, we not only have to calculate or estimate the proportions of the available materials, but must also make trial mixes. Properties of trial mixes are checked and adjustments in the proportioning are made until a fully satisfactory mix is attained.

Trial mixes are usually relatively small batches made with laboratory precision so that a certain security factor should be calculated when using laboratory results on large job-site batches.

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The calculation of material quantities for mix design of ordinary normal-weight concrete is based on the absolute volume of the ingredients:

Volumes in $1m^3$ compacted fresh concrete:

$$\frac{C}{\delta_c} + \frac{A}{\delta_A} + \frac{W}{\delta_w} + P = 1000\left[dm^3\right]$$

C = cement content in kg/m^3
A = aggregate content in kg/m^3
W = water content in kg/m^3
P = air pores;
in compacted non-air-entrained concrete: $10 \div 20\ dm^3/m^3$;
in air-entrained concrete: $30 \div 70\ dm^3/m^3$
δ_c = bulk density of cement kg/dm^3
δ_A = bulk density of aggregate kg/dm^3
δ_w = bulk density of water = 1.0 kg/dm^3

Guide values for the above bulk densities are given in Table 4

Table 4: Guide values for bulk densities δ in kg/dm^3

Cements		**Aggregates**	
Portland cement	3.05 - 3.15 kg/dm^3	Sand, gravel	2.6 - 2.65 kg/dm^3
Blast furnace slag cement	3.0 kg/dm^3	Limestone	2.7 kg/dm^3
Pozzolanic cement	2.9 kg/dm^3	Basalt	2.9 kg/dm^3
		Volcanic gravel + sand	2.1 - 2.4 kg/dm^3
		Lightweight aggregate for structural lightweight concrete	0.4 ÷ 1.9 kg/dm^3

There are various methods to calculate a concrete mix. The difference between those methods is not to be attributed to the basic factors but rather to the mode of calculation and adjustment applied. As example, three methods to calculate the mix design of normal weight concrete are compared in Table 5.

Here, only the general principle of the mix design procedure shall be briefly presented (see also example of mix design in the next chapter):

First of all, the maximum permissible w/c-ratio for the required concrete strength has to be fixed; this value can be taken from the relationship w/c-ratio - concrete strength in function of the strength class of the cement.

Then, the required water content w (mixing water plus surface humidity of aggregate) has to be defined. The water content w can be estimated based on the needed consistency and the aggregate grading and modulus respectively.

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Table 5: Example of methods to calculate a concrete mix

	Country & Standard (Code)		
Step No.	**United States ACI**	**England Dep. of Environment**	**W. Germany DIN**
1.	workability = slump	mean strength (w/c-ratio)	cement quality
2.	max. size of aggregate	water content (workability)	consistency
3.	mixing water + (air content)	cement content	grading of aggregate
4.	w/c ratio	aggregate content	water content
5.	cement content	fine/coarse aggregate	concrete strength
6.	coarse aggregate content	trial mixes	w/c ratio
7.	fine aggregate content		cement content
8.	adjustment for aggregate moisture		weight of ingredients
9.	trial batch adjustments		trial mix
For details consult:	ACI 211.1-91, ACI Manual of Concrete Practice, American Concrete Institute, USA	A. Neville: Properties of concrete. Chapter 14. Mix Design., Longman, 1995	K. Walz: Herstellung von Beton nach DIN 1045. Beton-Verlag GmbH, Düsseldorf, 1971.

The necessary cement content c in the concrete mix can accordingly be calculated with the w/c-ratio and the water content w by means of the following formula:

$$c = w/(w/c)$$

Knowing the cement and water content, we can calculate the required amount of aggregate using the equation on the absolute volume of the ingredients (assumption on air pores p). Thus, the weight of oven dry aggregate A for 1 m^3 of concrete is:

$$A = \delta_A (1000 - c/\delta_c - w - p)$$

If humid aggregate is used, then the amount of surface humidity during concrete preparation has to be determined and to be included for the proportioning of the mixing water and the aggregate. The mixing water is calculated by subtracting the content on surface humidity from the water content w. For the proportioning of the aggregate, the amount A calculated according to the above equation has to be increased by the content of surface humidity.

If plasticising concrete admixtures (e.g. water reducers or air entrainer) are added to the concrete, the water content w for a given consistency is reduced. The fine air pores produced by the addition of air entrainer improve also indeed the workability, but they reduce at the same time the concrete strength; in case of air entrained concrete, the real water content has therefore to be adjusted for both the plasticising and strength reducing effect of the air entrainer.

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5. EXAMPLE OF MIX DESIGN CALCULATION

For illustration, a mix design shall be calculated here with the aid of Table 6 and Figures 4 to 6 from DIN standard 1045.

In our example, the concrete requirements shall be as follows::

- consistency: compaction factor 1.20
- strength class: B 35
- exposure condition: interior of building (thus no further limitations on w/c-ratio and cement content)
- max aggregate size: 32 mm
- aggregate grading: between sieve curves A and B (see Figure 6)

The available concrete components are:

- cement class Z 35 (density 3.10 kg/dm^3)
- aggregates:

  sand 0/2 mm (density 2.52 kg/dm^3)
  natural aggregate 2/8 mm (density 2.62 kg/dm^3)
  natural aggregate 8/32 mm (density 2.72 kg/dm^3)

  with the following grading:

Passing sieves (%)	0.25mm	0.5 mm	1 mm	2 mm	4 mm	8 mm	16 mm	32 mm
sand	6	50	80	97	100	100	100	100
aggr. 2/8mm	3	8	10	10	55	95	100	100
aggr. 8/32 mm	2	2	3	3	4	6	50	100

The average compressive strength tested on three cubes in one trial batch for the concrete strength class B 35 should be at least 45 N/mm^2 (see Table 6). Accordingly, the water/cement-ratio can be estimated from Figure 4; it amounts to 0.47.

Aiming at a grading of the aggregate mixture between A and B sieve curves, the estimated amount of aggregate fractions is:

- sand: 25 vol.%
- 2/8 mm: 23 vol. %
- 8/32 mm: 52 vol. %

The resulting grading of the aggregate mixture is thus:

Passing sieves (%)	0.25 mm	0.5 mm	1 mm	2 mm	4 mm	8 mm	16 mm	32 mm
	3	15	24	28	39	49	74	100

With the corresponding grading modulus 4.68 and the concrete compacting factor of 1.20, the water content w estimated from Figure 5 is 153 kg/m^3.

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The calculated cement content c is therefore:

153 : 0.47 = 325 kg/m^3 of concrete

The volume proportion of the oven dry aggregate mixture in 1 m^3 of concrete can then be calculated by means of the absolute volume of the ingredients (assuming an air content P of 15 dm^3):

A = 1000 - (c/3.10 - w - P) = 1000 - (325/3.10 - 153 - 15) = 727 dm^3

consisting of:

727 x 0.25 x 2.62 = 476 kg of sand
727 x 0.23 x 2.62 = 438 kg of fraction 2/8 mm
727 x 0.52 x 2.72 = 1028 kg of fraction 8/32 mm

That means 1942 kg of aggregate mix in 1 m^3 of concrete.

The mix design is thus:

aggregate	1942 kg/m^3
cement	325 kg/m^3
water	153 kg/m^3

resulting in 2420 kg/m^3 of fresh compacted concrete.

Table 6: Requirements for concrete in preliminary test

Group of concrete	Strength class	Required compr. strength (N/mm^2)[1), 2)]	Required consistency values[2)]
B I	B 5 B 10 B 15 B 25	≥ 11 ≥ 20 ≥ 25 ≥ 35	K 1: v = 1.30...1.26 K 2: v = 1.15...1.11 a = 39...40 cm K3: v = 1.06...1.04 a = 48...50 cm
B II	B 35 B 45 B 55	40 + m[3)] 50 + m[3)] 60 + m[3)]	according to requirements of the site, incl. margin

[1)] average compressive strength of three cubes of one batch
[2)] site-mixed and ready-mixed concrete, not for concrete in concrete works
[3)] choose margin m according to experience, otherwise m ≥ 5 MPa are to be used

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Figure 4: Relationship strength versus w/c ratio

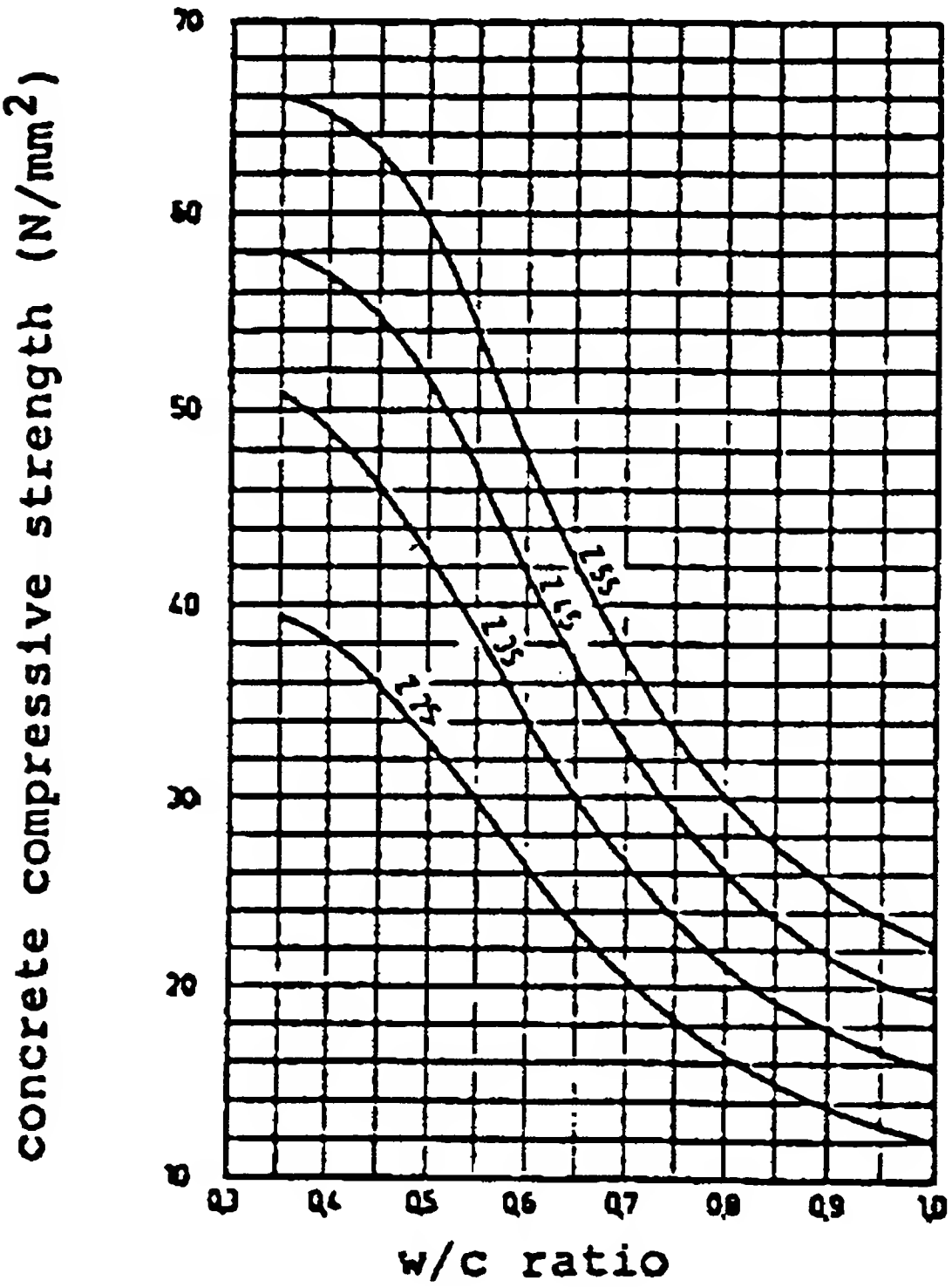
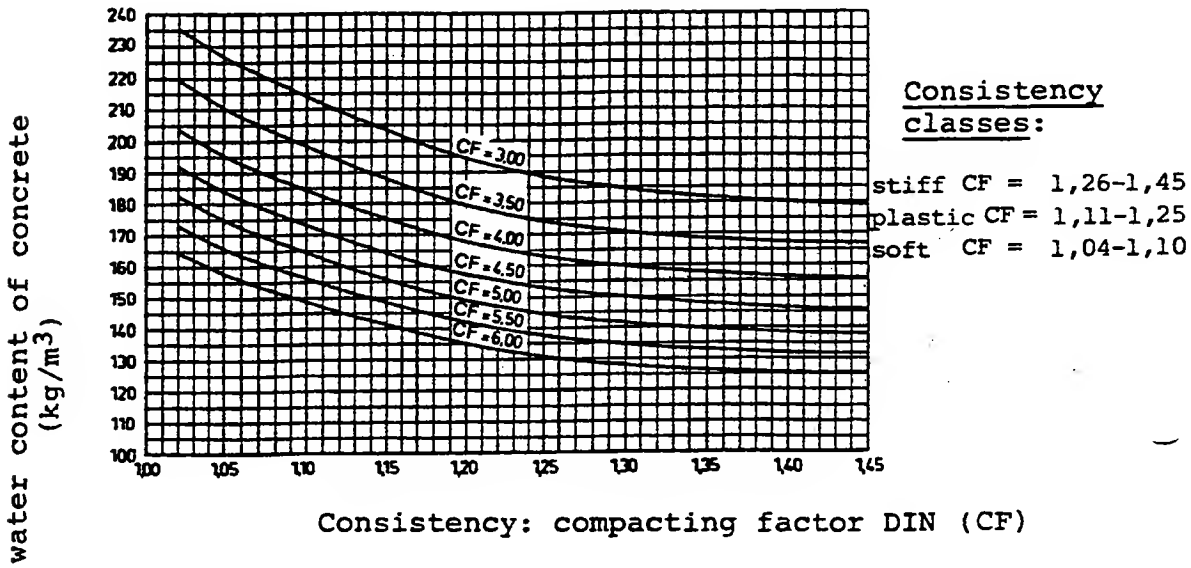


Figure 5: Relationship between consistency, grading modulus and water content of concrete



Correction of water content:

- by using crushed aggregate	+ 5 ÷ 10%
- by using plasticisers	- 5%
- by using air-entraining agents	- 1% per 1% of air content

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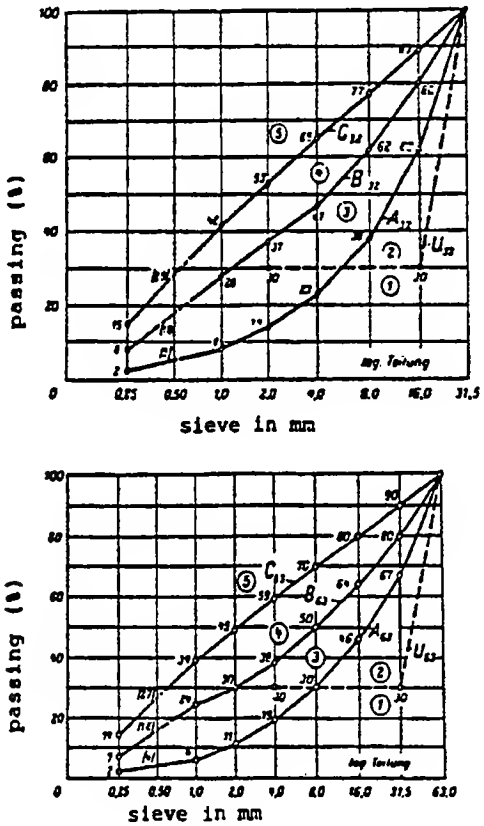
Figure 6: Standard grading curves



Classification of grading:

1	unfavourable
2	gap-graded mixes
3	favourable
4	serviceable
5	unfavourable

6. CONCLUDING REMARKS

There are several well established methods to calculate concrete mixes. Given the variability of the properties of ingredients and the difficulty in describing them, the results of these calculations are, however, really only guesses. On the other hand, such methods are concerned basically with the technical aspects of mix design without proper consideration of the economical side of the problem.

The establishment of a proper concrete mix design will always involve the preparation and testing of trial mixes and consecutive adjustments. The experience and knowledge on the influence of the various factors upon the properties of concrete can of course help to improve the first guess and limit the number of trial mixes to be tested.

To facilitate the selection of the optimum concrete mix design (also from the economic point of view), more and more computer based tools are used nowadays in practice. Such tools do not replace experience, but allow to explore more quickly the different alternatives and to arrive faster at the optimum solution.

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Chapter 10

Concrete Production Process

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1. GENERAL

Characteristics and properties of fresh and hardened concrete depend to a great extent upon mix design and quality of the constituent materials.

However, the importance of mixing, transporting, placing and curing techniques should not be neglected.

Improved production practices and techniques will then contribute considerably to achieving a good concrete. Each stage of concrete production is important and has an influence on the final concrete serviceability.

2. BATCHING OF MATERIALS

To produce concrete of uniform quality, the ingredients must be measured accurately for each batch. Most new specifications require that batching be done by weight rather than by volume because of the inaccuracies in measuring solid materials (especially damp sand) by volume. The weight system for batching provides greater accuracy, simplicity and flexibility. Flexibility is necessary because changes in the aggregate moisture content require frequent adjustments in batch quantities of water and aggregates. Water can be measured accurately by either volume or weight.

ACI specifications generally require that materials be measured within this percentage of accuracy:

cement	± 1%
aggregates	± 2%
water	± 1%
admixtures	± 3%

The ERMCO (European Ready Mixed Concrete Organisation) Code of good practice for Ready-Mixed Concrete gives recommendations for batching tolerance as listed in Table 1.

Table 1: Accuracy of batching acc. ERMCO

Material	Tolerance
Cement	± 3%
Coarse aggregate	± 3%
Fine aggregate	± 3%
Admixtures	± 5%
Water	± 3%

Equipment should be capable of measuring quantities within these tolerances for the smallest batch regularly used, as well as for larger batches.

The accuracy of batching equipment should be checked periodically and adjusted when necessary. Admixture dispensers should be checked daily since errors in admixture batching, particularly overdosing, can lead to serious problems in both fresh and hardened concrete.

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3. MIXING

The objective of mixing is:

- to coat the surface of all aggregate particles with cement paste
- to blend all the ingredients of concrete into a uniform mass
- to maintain uniformity of concrete at the discharging from the mixer

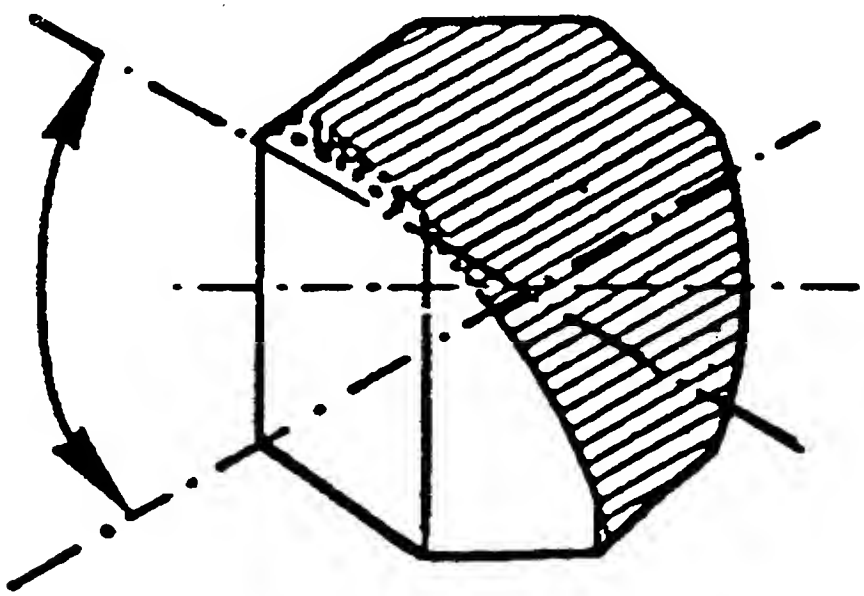
All concrete should be mixed thoroughly until it is uniform in appearance, with all ingredients evenly distributed. Mixers should not be loaded above their rated capacities and should be operated at approximately the speeds for which they were designed. If the blades of the mixer become worn or coated with hardened concrete, the mixing action will be less efficient. Badly worn blades should be replaced and hardened concrete should be removed periodically, preferably after each day's run of concrete.

3.1 Types of mixers

The method of discharging is one of the criteria for the classification of concrete mixers.

Tilting Mixer: Mixing chamber (drum) it tilted for discharging (see Figure 1).

Figure 1: Tilting mixer



Drum (non-tilting) mixer: The axis of the mixer is always in a horizontal position, and discharge is effected by inserting a chute into the drum or by reversing the direction of rotation of the drum, as applied in the truck mixer used in the ready-mixed concrete production (see Figure 2).

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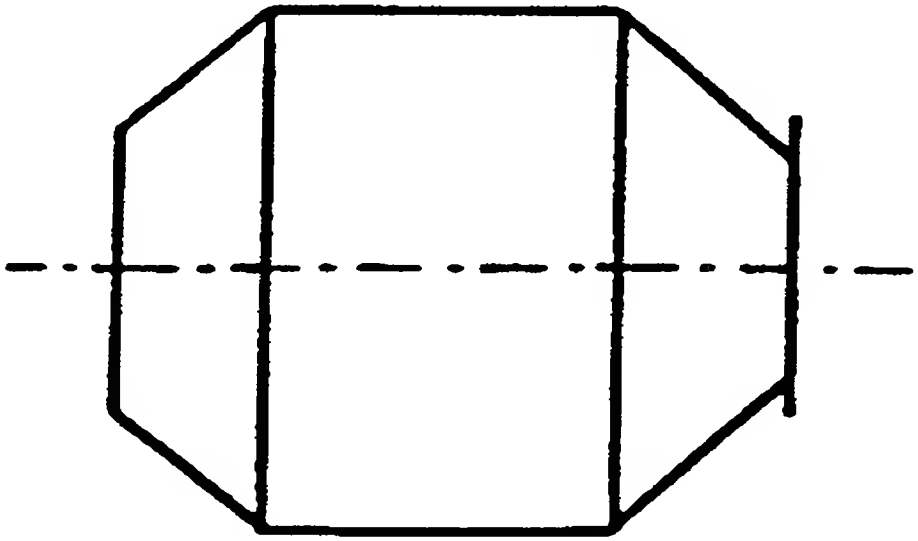
Figure 2: Drum (non-tilting) mixer



Pan mixer: consists of a circular pan rotating about its axis, with one or two stars of paddles rotating about a vertical axis not coincident with the axis of the pan (see Figure 3).

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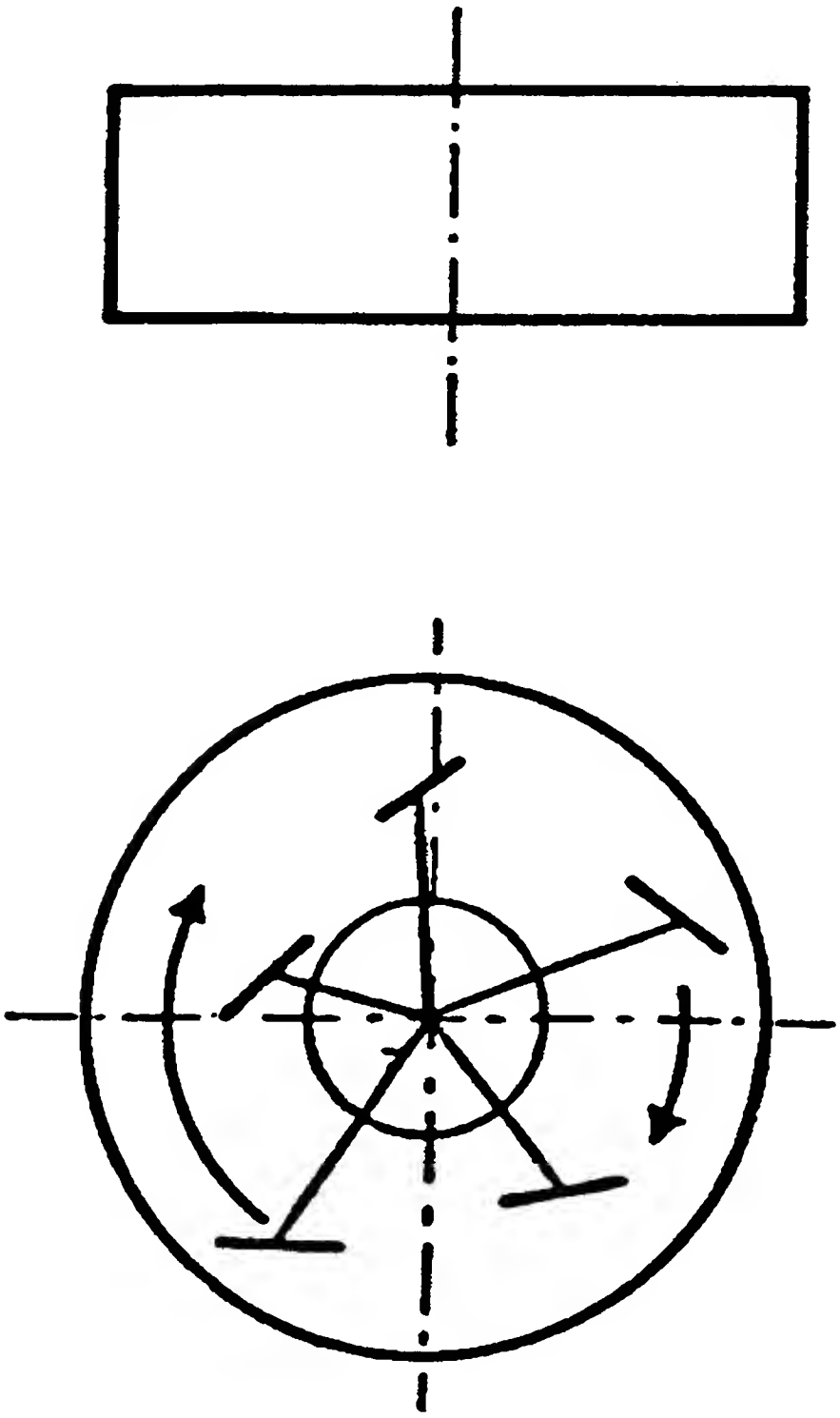
Figure 3: Pan mixer



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The capacity of tilting and drum mixers is usually large (up to 6 m^3), whereas the pan mixer is smaller (up to 2 m^3) and is particularly efficient for stiff and cohesive mixes often used in precast concrete industry.

Bowl-and-stirrer Mixer (cake mixer): works according to the same principle as the pan mixer and is sometimes used for mixing of mortar.

3.2 Uniformity of mixing

The efficiency of the mixer can be determined by the variability of the mix discharged into a number of samples without interrupting the flow of concrete.

The values given in Table 2 are the highest acceptable values for a 'satisfactory' mixer.

Table 2: Variability of concrete in a 'satisfactory' mixer

Compressive strength	4 - 6%
Percentage of coarse aggregate	6 - 8%
Percentage of fine aggregate or cement	5 - 8%

3.3 Mixing time

The mixing time varies with the type of mixer and, strictly speaking, it is not the mixing time but the number of revolutions of the mixer that is the criterion of adequate mixing. Generally about 50 revolutions are sufficient.

4. HANDLING, TRANSPORTING, PLACING

Each step in handling, transporting and placing concrete should be carefully controlled to maintain uniformity within the batch and from batch to batch so that the completed work is consistent throughout. It is essential to avoid separation of the coarse aggregate from the mortar or of water from the other ingredients.

Concrete is handled and transported by many methods. These include the use of chutes, buggies operated over runways, buckets handled by cranes or cabinways, small railroad cars, trucks, pumping through pipelines, and pneumatically forcing the concrete or dry concrete materials through hoses.

Preparation prior to placing includes compacting, trimming and moistening the subgrade; erecting the forms; and setting the reinforcing steel. A moist subgrade is especially important in hot weather to prevent extraction of water from the concrete.

Forms should be clean, tight, adequately braced, and constructed of materials that will impart the desired texture to the finished concrete. Sawdust, nails and other debris should be removed before concrete is placed. Wood forms should be moistened before placing concrete; otherwise they will absorb water from the concrete and swell. Forms also should be treated with a releasing agent such as oil or lacquer to facilitate their removal. For architectural concrete, lacquer or emulsified stearates are used since they are non-staining.

Reinforcing steel should be clean and free of loose rust or mill scale at the time concrete is placed. Any coatings of hardened mortar should be removed from the steel.

5. COMPACTING

The process of compacting concrete consists essentially of the elimination of air voids.

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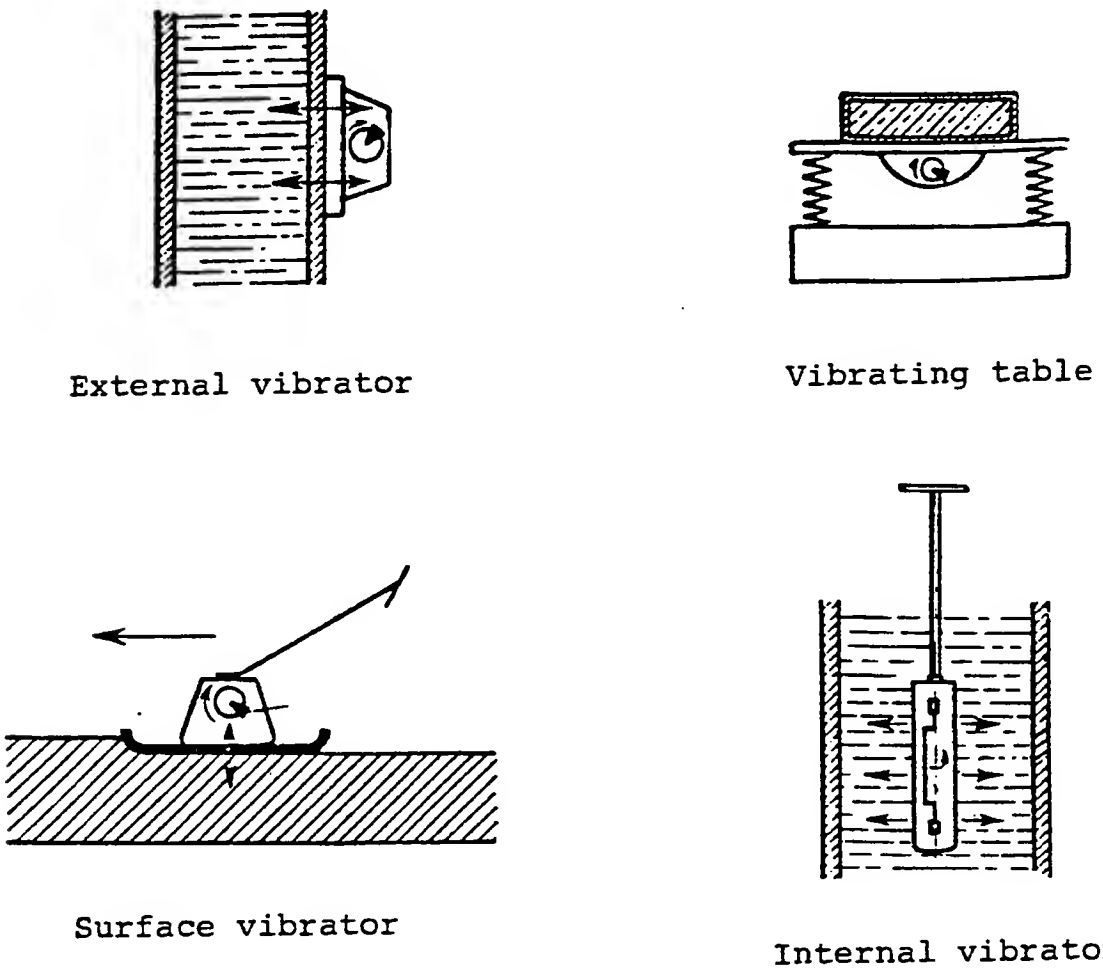
The oldest means of achieving this is by ramming or punning the surface of concrete. The most common method for compacting is by vibration. The use of vibration makes it possible to work with drier mixes, so that concrete with a lower water/cement ratio and a lower cement content can be manufactured. Whether or not vibrators can be employed is determined by the consistency of the mix. Mixes which are very wet should not be vibrated as separation may result. Each type of vibrator requires a different consistency.

Various types of vibrators have been developed (see also Figure 4):

Figure 4: Various types of vibrators



5.1 Internal vibrator

Of the several types of vibrators this is perhaps the most common one. It consists essentially of a poker, housing an eccentric shaft driven through a flexible drive from a motor. The poker is immersed in concrete and thus applies approximately harmonic forces to it; hence, the alternative names of poker- or immersion vibrator.

The frequency of vibration varies up to 12'000 cycles of vibration per minute.

The poker is easily moved from place to place, and is applied at 50 to 70 cm centers for 5 to 30 sec., depending on the consistency of the mix.

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5.2 External vibrator

This type of vibrator is rigidly clamped to the formwork resting on an elastic support, so that both the form and the concrete are vibrated. As a result, a considerable proportion of the work done is used in vibrating the formwork, which also has to be strong and tight so as to prevent distortion and leakage of grout.

The principle of an external vibrator is the same as that of an internal one, but the frequency is usually between 3000 and 6000 cycles.

5.3 Vibrating table

This can be considered as a case of formwork clamped to the vibrator instead of the other way round, but the principle of vibrating the concrete and formwork together is unaltered.

The source of vibration, too, is similar. Generally a rapidly rotating eccentric weight makes the table vibrate with a circular motion. With two shafts rotating in opposite directions the horizontal component of vibration can be neutralized so that the table is subjected to a simple harmonic motion in the vertical direction only.

5.4 Surface vibrator

A surface vibrator applies vibration through a flat plate direct to the top surface of the concrete. In this manner the concrete is restrained in all directions so that the tendency to segregate is limited; for this reason a more intense vibration can be used.

6. CURING OF CONCRETE

Properties of concrete such as resistance to freezing and thawing, strength, watertightness, wear resistance, and volume stability improve with age as long as conditions are favorable for continued hydration of the cement. The improvement is rapid at early ages but continues more slowly for an indefinite period. Two conditions for such improvement in quality are required:

- the presence of moisture
- a favorable temperature

Excessive evaporation of water from newly placed concrete can significantly retard the cement hydration process at an early age. Loss of water also causes concrete to shrink, thus creating tensile stresses at the drying surface. It these stresses develop before the concrete has attained adequate strength, surface cracking may result. All exposed surfaces, including exposed edges and joints, must be protected against moisture evaporation.

Hydration proceeds at a much slower rate when the concrete temperature is low; from a practical standpoint there is little chemical action between cement and water when the concrete temperature is near or below freezing. It follows that concrete should be protected so that moisture is not lost during the early hardening period and the concrete temperature is kept favorable for hydration.

6.1 Length of curing period

Since all the desirable properties of concrete are improved by curing, the curing period should be as long as practicable in all cases.

The length of time that concrete should be protected against loss of moisture is dependent upon the type of cement, mix proportions, required strength, size and shape of the concrete mass, weather and future exposure conditions. This period may be a month or longer for lean concrete mixtures used in structures such as dams; conversely, it may be only a few

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days for richer mixes, especially if Type III of high-early-strength cement is used. Steam-curing periods are normally much shorter.

For must structural uses, the curing period for cast-in-place concrete is usually 3 days to 2 weeks, depending on such conditions as temperature, cement type, mix proportions, etc. More extended curing periods are desirable for bridge decks and other slabs exposed to weather and chemical attack.

6.2 Curing methods

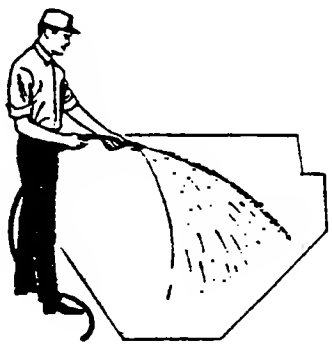
There are different methods to cure concrete:

1) Methods that supply additional moisture to the surface of the concrete during the early hardening period. These include sprinkling and using wet coverings. Such methods afford some cooling through evaporation, which is beneficial in hot weather.
2) Methods that prevent loss of moisture from the concrete by sealing the surface. This may be done by means of waterproof paper, plastic sheets, liquid membrane-forming compounds, and forms left in place.
3) Methods that accelerate strength gain by supplying heat and moisture to the concrete. This is usually accomplished with live steam or heating coils.

Sprinkling (see Figure 5)

Continuous sprinkling with water is an excellent method of curing . If sprinkling is done at intervals, care must be taken to prevent the concrete from drying between application of water. A disadvantage of sprinkling may be its cost. The method requires an adequate supply of water and careful supervision.

Figure 5: Sprinkling



Wet coverings

Burlap, cotton mats, or other moisture-retaining fabrics are used for curing. Treated burlaps that reflect light and are resistant to rot and fire are available. Coverings should be placed as soon as the concrete has hardened sufficiently to prevent surface damage. Care should be taken to cover the entire surface, including the edges of slabs such as pavements and sidewalks. The coverings should be kept continuously moist so that a film of water remains on the concrete surface throughout the curing period.

Waterproof paper (see Figure 6)

Waterproof curing paper is an efficient means of curing horizontal surfaces and structural concrete of relatively simple shapes. One important advantage of this method is that periodic additions of water are not required.

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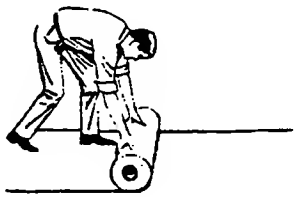
Figure 6: Waterproof paper

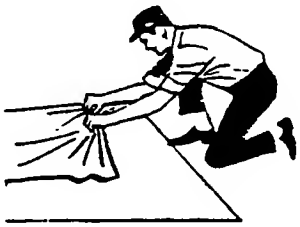


Plastic sheets (see Figure 7)

Certain plastic sheet materials (polyethylene films) are used to cure concrete. They are lightweight, effective moisture barriers and easily applied to simple as well as complex shapes.

In some cases, the use of thin plastic sheets for curing may discolor the hardened concrete. This may be especially true when the concrete surface has been steel-troweled to a hard finish. When such discoloration is objectionable, some other curing method is advisable.

Figure 7: Plastic sheets



6.3 Curing Compounds

Liquid membrane-forming curing compounds retard or prevent evaporation of moisture from the concrete. They are suitable not only for curing fresh concrete, but also for further curing of concrete after removal of forms or after initial moist curing.

The concrete surface should be moist when the coating is applied. Normally only one smooth, even coat is applied, but two coat may be necessary to ensure complete coverage. A second coat, when used, should be applied at right angles to the first.

Curing compounds used in hot weather should be white colored.

Curing compounds are applied by hand-operated or power-driven spray equipment (see Figure8) immediately after the disappearance of the water sheen and the final finishing of the concrete.

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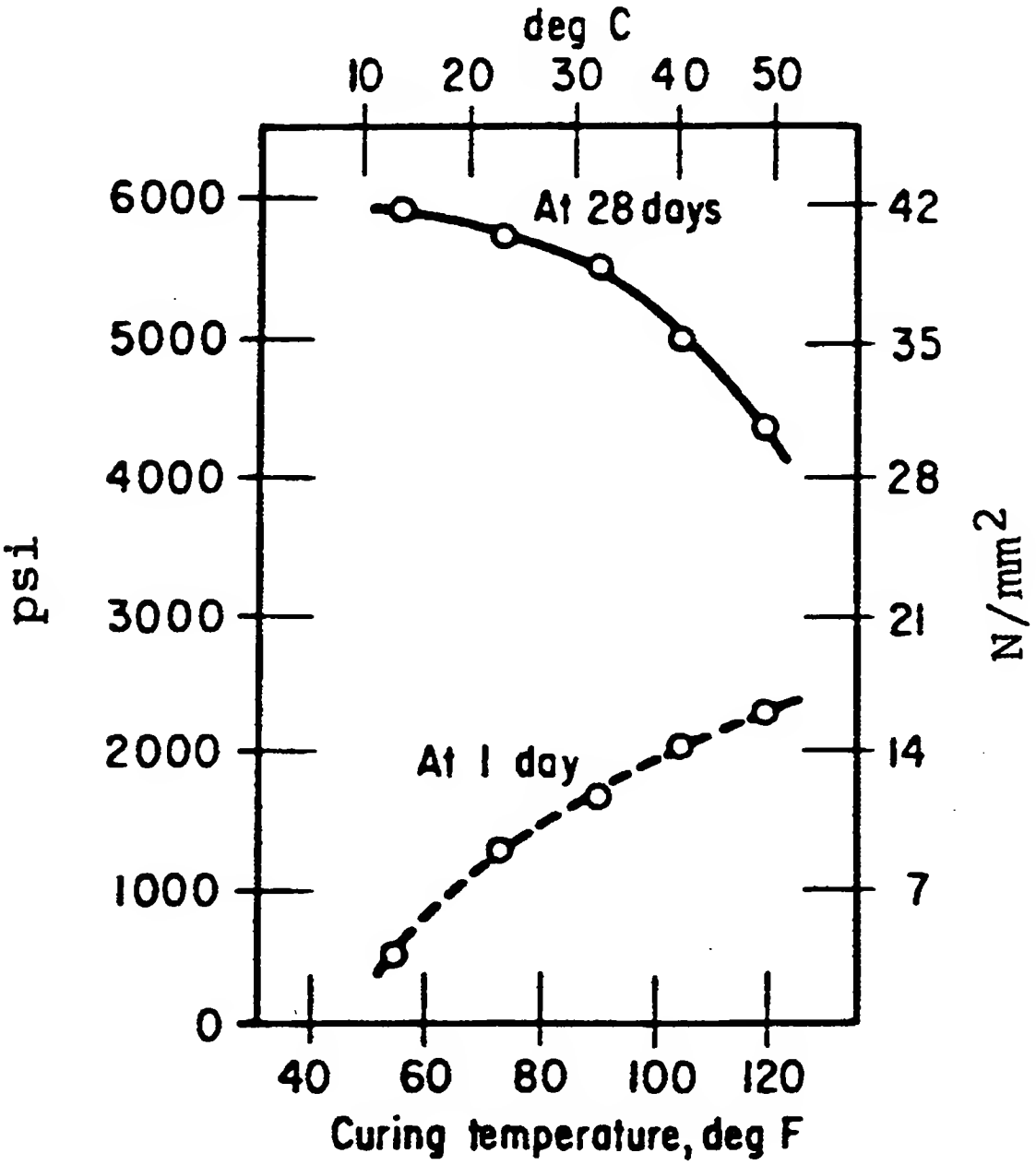
Figure 8: Curing compounds



6.4 Steam curing

Of the methods in Group 3, accelerating the strength development, the most common is steam curing. It can be used to advantage where early strength gain in concrete is important or where additional heat is required to accomplish hydration, as in cold-weather concreting.

Two methods of steam curing for early strength gain are used today:

- curing in live steam and atmospheric pressure (for enclosed cast-in-place structures and manufactured precast concrete units)
- curing in high-pressure steam autoclaves (for small manufactured units).

A steam-curing cycle consists of:

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1) an initial delay prior to steaming
2) a period for increasing temperature
3) a period for holding the maximum temperature constant
4) a period for decreasing temperature

A typical atmospheric steam-curing cycle is shown in Fig. 11, chapter 'Concrete Categories', paragraph 3.6.

Steam curing at atmospheric pressure is generally done in a steam chamber or other enclosure to minimize moisture and heat losses.

High-pressure steam curing in autoclaves takes advantage of temperatures in the range of 160 to 190°C and corresponding pressures. Hydration is greatly accelerated and the elevated temperatures and pressures may produce additional beneficial chemical reactions between the aggregates and/or cementitious materials that do not occur under normal steam curing.

7. HOT WEATHER CONCRETING

Hot weather for concreting is defined as any combination of high air temperature, low relative humidity and wind velocity, tending to impair the quality of fresh or hardened concrete or otherwise resulting in abnormal properties. Hot weather can adversely affect the properties and serviceability of concrete.

Concrete mixed, placed and cured at elevated temperatures normally develops higher early strength than concrete produced at normal temperatures, but the 28 day or later strength is generally lower (Figure 9).

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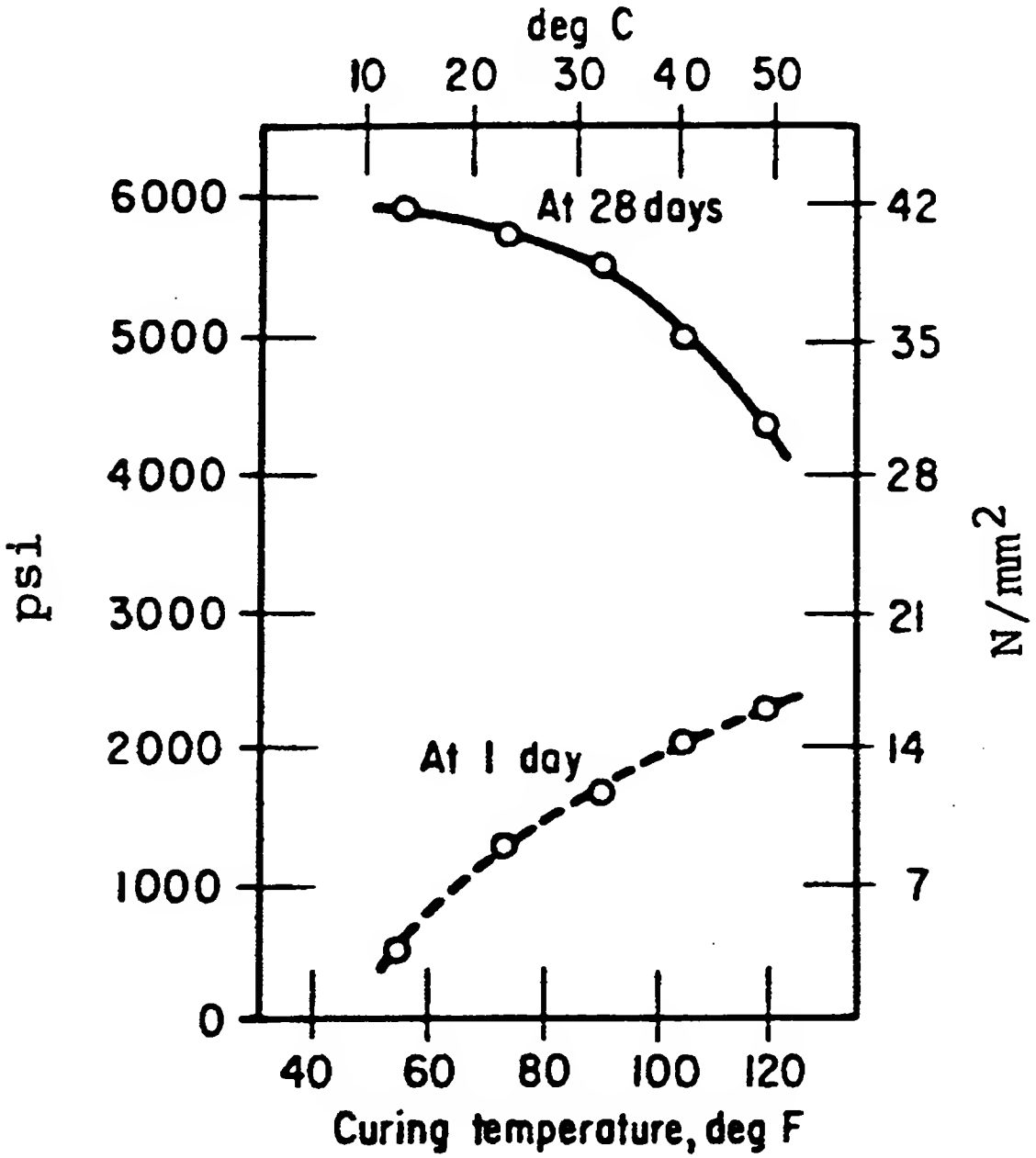
Figure 9: Influence of curing temperatures on strength at day and 28 days



Plastic shrinkage cracking is usually associated with hot weather concreting. High concrete temperature, high air temperature, high wind velocity and low humidity, or combinations thereof, cause rapid evaporation which significantly increases the probability that plastic shrinkage cracking will occur.

As shown in Figure 10, the amount of mixing water required to make a concrete of a certain consistency, increases considerably as the temperature of fresh concrete increases.

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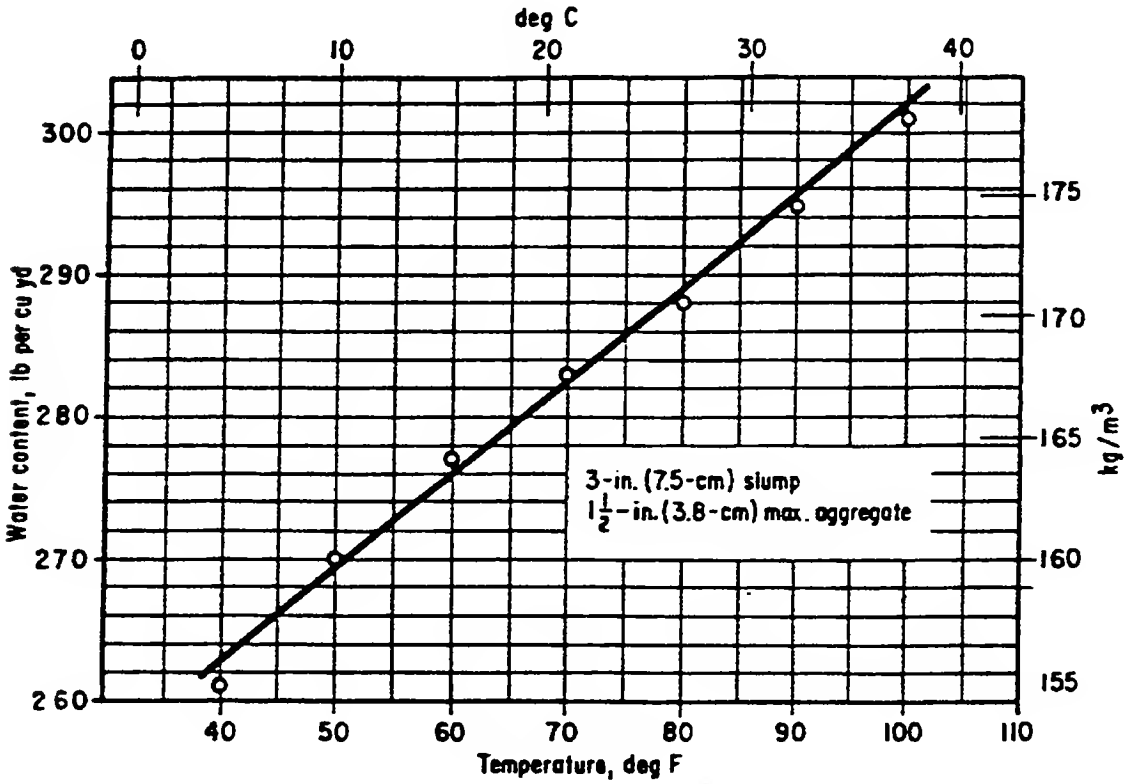
Figure 10: Water requirement of a concrete mix increases with an increase in temperature



For the common, non-massive types of structure a maximum concrete temperature of 32°C is recommended by ACI as a reasonable upper limit. For the massive types of construction a temperature of 16°C or even lower would be desirable.

The most practical method of maintaining low concrete temperatures is to control the temperature of the materials.

The equation given below shows that, for concrete of conventional properties, a reduction of the concrete temperature by 1°C requires reducing either the cement temperature by about 8°C or the water temperature by about 4°C or the aggregate temperature by about 2°C.

Equation:

$$T = \frac{0.22\,(T_a W_a + T_c W_c) + (T_w W_w + T_{wa} W_{wa})}{0.22(W_a + W_c) + W_w + W_{wa}}$$

T	=	temperature of freshly mixed concrete (deg. C)
T_a, T_c, T_w, T_{wa}	=	temperature of aggregate, cement, mixing water and water on aggregate respectively (deg. C)
W_a, W_c, W_w, W_{wa}	=	weight of aggregate, cement, added mixing water and water on aggregate, respectively (kg)

Of the materials contained in concrete, water is the easiest to cool. It can be cooled by refrigeration or by adding ice which is used as part of the mixing water, provided that it is completely melted by the time mixing is completed.

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Cement temperature has only a minor effect on the temperature of the freshly mixed concrete because of the low specific heat of cement and the relatively small amount of cement contained in the mix. Fresh hot cement from the plant can cause difficulties.

Aggregates have a pronounced effect on the fresh concrete temperature because they represent 60 to 80% of the total weight of concrete. Aggregate stockpiles should be shaded from the sun and kept moist by sprinkling. Before concrete is placed, the forms, the reinforcing steel, and the subgrade should be cooled by sprinkling as well.

Transporting and placing of concrete should be done as quickly as practicable. During extremely hot periods improved results may be obtained by restricting the placing of concrete to the early morning or evening hours.

In order to prevent concrete from drying out, curing should be started as soon as the concrete surface is finished. In special cases of concreting during hot weather a retarding admixture may be used to delay the setting time.

8. COLD WEATHER CONCRETING

Concreting in cold weather requires special precautions. Concrete sets slowly in cold weather and development of strength is delayed. At low temperatures (at +5°C or lower during placing and the early curing period), one or more of these recognized protective measures should be used:

- Heating the area where concrete is placed
  - The presence of ice and the possibility of ice formation during concreting must be avoided. Temperature of the concrete forms should be raised to above freezing, as well as that of adjacent concrete and subgrade, through the use of heated enclosures.
- Heating the water and concrete materials
  - In cold weather, freshly placed concrete should be at least 10°C and not more than 32°C when poured in the forms. In addition to heating water, it may be necessary to heat aggregates.
- Use of chemical accelerators
  - During low temperatures, the use of chemical accelerators can speed up the set of concrete. Calcium chloride is the most commonly used and may be used up to 2% by weight of cement.
- Calcium chloride or admixtures containing soluble chlorides must not be used:
  - in reinforced and prestressed concrete
  - in concrete containing embedded aluminium
  - in lightweight insulating concrete place over metal decks
  - in concrete that will be in contact with soils, or water containing sulfates
- Maintaining concrete temperatures
  - Concrete slabs lose heat and moisture rapidly in cold weather. They need protection against wind and cold. A heated enclosure or insulation should be provided to keep concrete temperature above 10°C. The following Time-Temperature Chart shows minimum periods in which concrete temperatures should be maintained:

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	20°C	10°C
plain concrete	3 days	7 days
plain concrete with calcium chloride	2 day	3 days

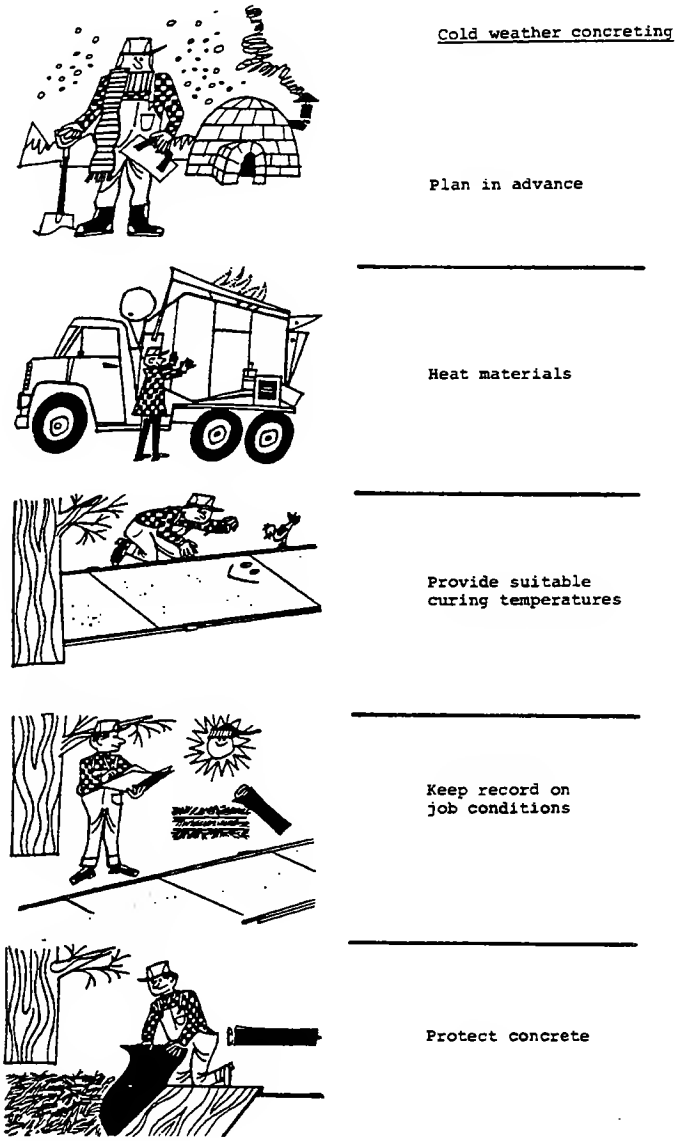
- Note: After the periods shown above, concrete temperature should be maintained above 5°C for at least four days. Concrete should not be allowed to dry out.

♦ Special provisions for curing

- Cold weather air acts like a sponge to draw moisture from concrete in both its fresh and hardened state. This rapid drying-out process must be avoided. Curing and protection from start to finish should be continuous and uninterrupted until the concrete attains its designed strength. At the end of the curing period (see Time-Temperature Chart above), protection should be removed in such a way that the temperature of the concrete will not drop faster than 5°C in 24 hours.

The following figure 11 illustrates the precautions that should be taken for cold weather concreting.

Figure 11:



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9. READY-MIXED CONCRETE

If, instead of being batched and mixed on the job site, concrete is delivered ready for placing from a central plant, it is referred to as **ready-mixed concrete**.

This type of concrete is used extensively in many countries as it offers numerous advantages as compared to the job-site-mixed concrete (Table 3).

Table 3: Production of ready-mixed concrete in W. Europe (from ERMCO Annual Report 1976/77 and 1995/96)

Country	Cement consumption in ready-mixed concrete in % of total cement consumption		Ready-mixed concrete production in mio m^3	
	1976/77	1995/96	1976/77	1995/96
Austria	19.5	44.0	4.3	8.6
Belgium	30.9	39.0	6.34	9.1
W. Germany	44.6	55.0	49.8	68.3
Finland	57.7	40.0	3.4	1.6
France	26.3	40.6	25.2	29.7
Great Britain	42.1	50.0	24.8	22.3
Italy	23.1	43.8	31.7	54.0
Spain	19.0	36.6	15.6	36.2
Holland	36.4	52.0	6.8	8.0
Sweden	48.4	61.7	5.2	2.4
Switzerland	40.7	57.0	5.6	9.0
USA	60.0	71.5	130.0	192.0

Ready-mix concrete plants use precision scales to weigh the ingredients as the producer is responsible for delivering concrete of the required quality.

There are three types of mixing:

- Transit mixed concrete is mixed completely in a truck mixer. The batching of ingredients is carried out in a dry-batch plant.
- Central mixed concrete is batched and mixed completely in a stationary batching and mixing plant (premix-plant) and is delivered in a special dump truck (tipper) without mixing or agitating, or in a truck mixer at agitating speed.
- Shrink-mixed concrete is mixed partially in a stationary mixer and then completed in a truck mixer.

Because of the many advantages offered by ready-mixed concrete, this industry has shown a phenomenal growth in recent years. These advantages can be summarized as follows:

- Use of modern precision batching equipment assures an accurately proportioned mix and allows the use and modification of different mix designs.
- Thorough mixing of each batch helps to produce uniform concrete.
- More efficient operation on the job site (no space problems with respect to storage of ingredients, etc.).
- Delivery at the time required.

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- Delivery in the exact quantity desired, thus eliminating waste.
- Equipment on the building site not necessary.
- Possibility to use different chemical admixtures.
- Possibility to use special concretes.
- Clear and easy base for delivery contracts; no problems with estimates of concrete cost.

The basis for the purchase of ready-mixed concrete is the concrete volume (m^3, cubic yard) and required concrete quality. To attain the desired concrete quality, different types of concrete mixes (mix design) can be used (see also paper on "Concrete Mix Design"):

- Designed mix (performance mix): concrete is designed by the ready-mix producer and should fulfill all requirements as stated by the purchaser.
- Codified mix: mix proportions are established in a National Code and are usually suitable for a restricted range of concrete strength classes and applications (for example in CEB-Code only for mixes with strength ≤ 25 N/mm^2).
- Prescribed mix: the purchaser is responsible for designing the concrete mix and specifies the mix proportions and the materials to be used by the ready-mix producer.

In recent years the ready-mixed concrete industry has gained considerable influence on the development of concrete standards and specifications. Several new developments in the concrete technology have been introduced by the ready-mixed concrete industry, e.g.

- flowing concrete
- ready-mixed mortar
- developments of new accelerated testing methods, etc.

10. PUMPED CONCRETE

Pumped concrete may be defined as concrete conveyed by pressure through either a rigid pipe or flexible hose and discharged directly into the desired area.

Pressure is applied by (see Figures 12 to 14):

- Piston pumps
- Pneumatic pumps (compressed air)
- Squeeze pressure pumps

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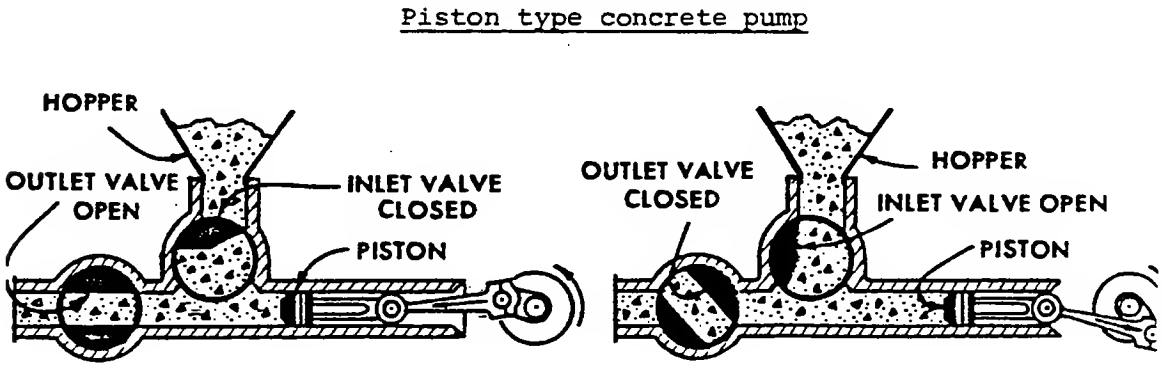
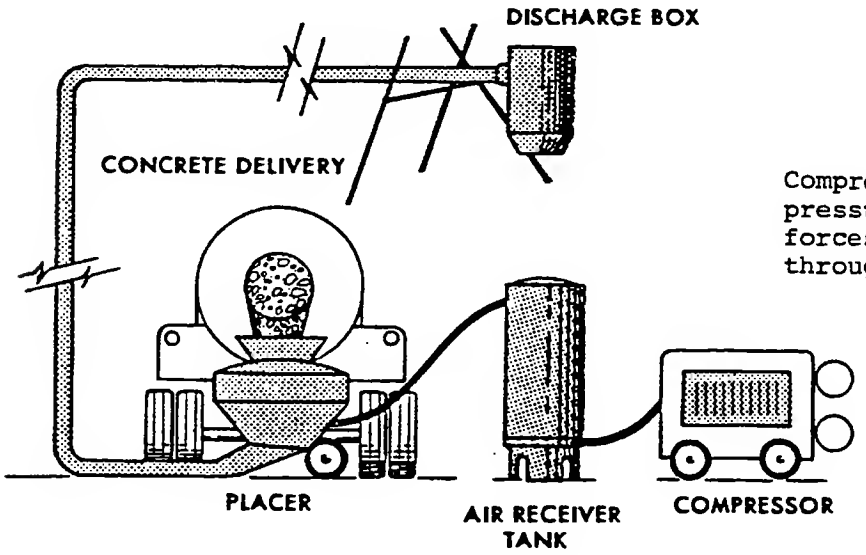
Figure 12: Schematic drawings of the various pump types - piston type concrete pump



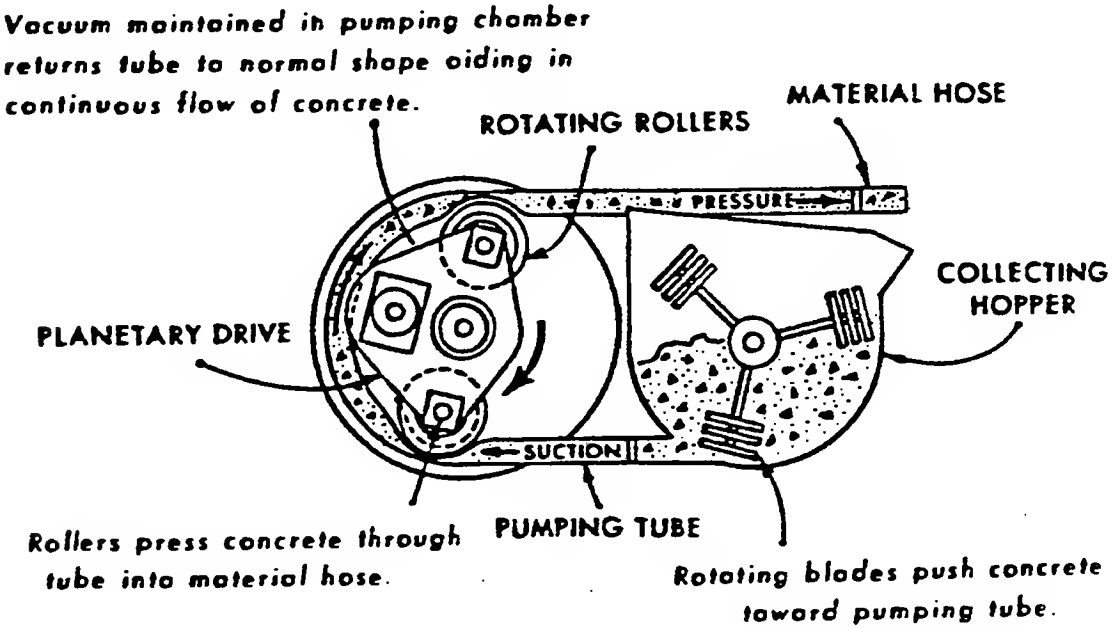
Inlet valve opens while outlet valve is closed and concrete is drawn into cylinder by gravity and piston suction. As piston moves forward inlet valve closes, outlet valve opens, and concrete is pushed into pump line.

Figure 13: Pneumatic type concrete pump



Compressor builds up air pressure in tank, which forces concrete in placer through the line.

Figure 14: Squeeze Pressure Type Concrete Pump



Pumping may be used for most concrete constructions, but it is especially useful where space for construction equipment is limited.

Very often the ready-mixed concrete companies include a pumping division. The portion of pumped concrete as compared to the total ready-mixed concrete production in various countries is shown in Table 4.

Table 4: Pumped concrete in proportion to the total ready-mixed concrete production (1996)

(Ready-mix concrete companies affiliated with the 'Holderbank' Group)

Country	No. of Plants	Pumped concrete in % of the total production
Belgium	55	30
France	116	7
Switzerland	18	15
W. Germany	38	49
Greece	3	74
Colombia	29	37
Costa Rica	13	33
Mexico	62	32
Brazil	40	30
Ecuador	7	70
South Africa	38	16

The concrete pumps can be:

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- Truck-mounted or
- Stationary

Ease of operation, safety and economy is required of a good pump. Today the most widely used diameter of a pipe for pumping is 125 mm; the usual maximum grain size is 32 mm, the minimum cement content about 300 kg and the consistency should be plastic (slump 7 to 12 cm).

Pumping jobs over heights of more than 200 m and with volumes of up to 110 m^3/hour are quite common in today's concrete practice.

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11. SPECIAL CONCRETING PROCESSES

The various special concreting processes will not be discussed in detail. However, some of them are briefly explained in the following:

- Vacuum-processed concrete: concrete from which, after compaction and before hardening, water is extracted by a vacuum process. The w/c ratio is thus reduced so that concrete with higher strength, higher density, lower permeability and better durability can be obtained.
- Preplaced-aggregate concrete: concrete produced by placing coarse aggregate into a mould and later injecting a cement-sand grout, usually with admixtures to fill the voids.
- Dry-packed concrete: concrete mixture sufficiently dry to be consolidated by heavy ramming.
- Spun concrete: concrete compacted by centrifugal action, e.g. in the manufacture of pipes (centrifugal process, roller-suspension process).
- Tremie concrete: concrete placed under water through a pipe or tube.
- Shotcrete: mortar or concrete conveyed through a hose and projected at high velocity onto a surface; also known as air-blown mortar, pneumatically applied mortar or concrete, sprayed mortar and gunned concrete (gunite).

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Chapter 11

Concrete Categories

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CONCRETE CATEGORIES

By Horst Wolter, Product Development and Application, HMC

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1. CLASSIFICATION OF CONCRETE

Concrete, for example, can be classified according to the production method or to the type. The following table gives an overview about the main categories:

Production methods and types of concrete

Production methods	Concrete types
Ready-mixed concrete	Plain concrete
Site mixed concrete	Reinforced concrete
Shotcrete	Pre-stressed concrete
Roller compacted concrete	Lightweight concrete
Pumped concrete	High performance concrete
Slipforming	Fibre reinforced concrete
Underwater concrete	Mass concrete
Vacuum concrete	Mortars
Steam curing concrete	Repair mortar
Autoclaved concrete	Flowing fill
	Heat resistant concrete
	Concrete for aggressive environment
	Coloured concrete

2. PLAIN CONCRETE

Plain concrete containing no steel reinforcing bars or wire, or containing not more than two tenth of percent of reinforcing to reduce shrinkage and temperature cracking.

3. REINFORCED CONCRETE

3.1 Introduction

Concrete is a brittle material and has therefore as all such materials a high compressive but a poor tensile strength. The compressive strength of concrete is about 10 times greater than the tensile strength. When, however, a reinforcement is used, the tensile strength is equal or greater than its compressive strength, that means, the steel is carrying the tensile stress and the concrete takes up the compressive strain; the most concrete structures are reinforced.

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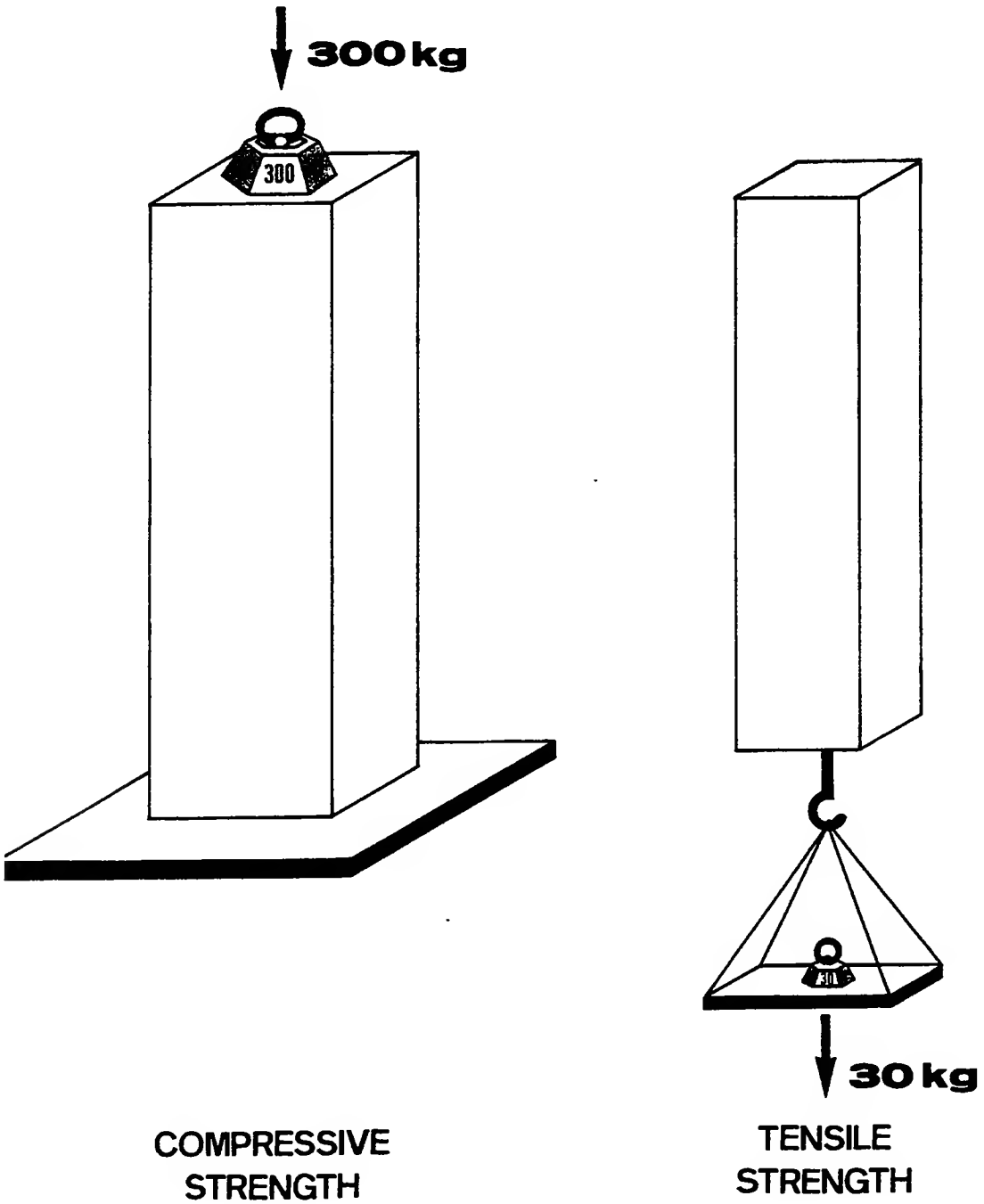
Fig. 1: Compressive and tensile strength of concrete



Reinforcement is the term used to describe the steel bars and small or large welded wire fabric positioned in concrete to increase its tensile stress. Many materials have been tried as reinforcement in concrete.

Steel is universal accepted and used. Reasons for practical utilisation of steel as reinforcement in concrete are as follows:

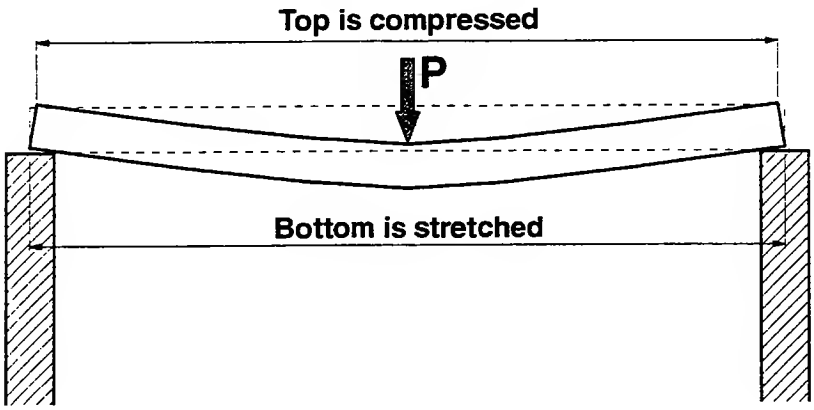
- Steel has a high tensile strength
- Very similar coefficient of thermal expansion of both materials (concrete and steel)
- Concrete protects steel against corrosion
- Satisfactory bond between both concrete and steel

3.2 How Reinforcement Works?

Concrete can be subjected to a number of tensile forces including a straight tensile pull, bending, and forces resulting from temperature and moisture changes. Concrete is often subjected to a straight tensile pull as in round structures such as water tanks and farm silos.

Pressures within the tank tend to push the two halves apart. Reinforcing steel in the wall holds the tank halves together. The steel must be around the entire tank wall since the outward pressures act in all directions. For round, structures, the reinforcing steel is usually placed near the centre of the wall cross section. In concrete beams only part of the beam contains tensile forces. Therefore, the location of the reinforcement is just as critical as the amount used to resist the bending loads; the reinforcement is placed near the lower side.

Fig. 2: Location of compressive and tensional forces in a simple beam



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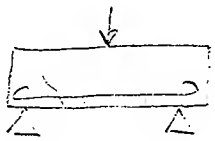
3.3 Placement of Reinforcement

The size, location and spacing of the reinforcement are designed in advance by engineers and are an important part of the design of the whole reinforced concrete structure. All reinforcement should be placed so that it is protected by an adequate coverage of concrete.

Concrete cover of reinforcement, as specified in the standards are between 10 to 50 mm. (for different types of structure and environmental conditions). In recent years, great strides have been made in reinforced concrete technology. Reinforcing bars are constantly being improved. The deformed bars used today bond to the concrete much more firmly than those used years ago. This improvement was achieved by changing the shape of the deformity or lugs. A recent development is high-strength steel reinforcing bars. When high-strength steel is used, the amount of reinforcing steel can be reduced.



4. PRESTRESSED CONCRETE

4.1 Overview

One of the disadvantage of ordinary reinforced concrete is that a reinforced concrete member develops cracks on the tension side before the reinforcement is stressed to its design load. The maximum possible elongation of concrete is about 0.01 to 0.02 %, whereas the elongation of the reinforcement when stressed to 150 MPa is about 0.06%. The crack widths are proportional to the steel stress and, in the case of plain rods, to the diameter of the reinforcing elements.

In **prestressed concrete**, large and carefully located compressive forces are introduced into high-strength concrete for increased flexural resistance or load-carrying capacity. A permanent state of induced compression is thus established and this transforms concrete into an elastic material which will carry forces that would otherwise cause critical tension or cracking.

4.2 Pre-tensioning Method

Tendons of high-tensile wire are tensioned between end grips or anchorages before the concrete is cast. The tension is usually applied by a single-strand hydraulic jack. After the concrete has hardened, the tension is released and prestress is transferred by bond from the tendons to the concrete. The method is very suitable for mass production with good factory control.

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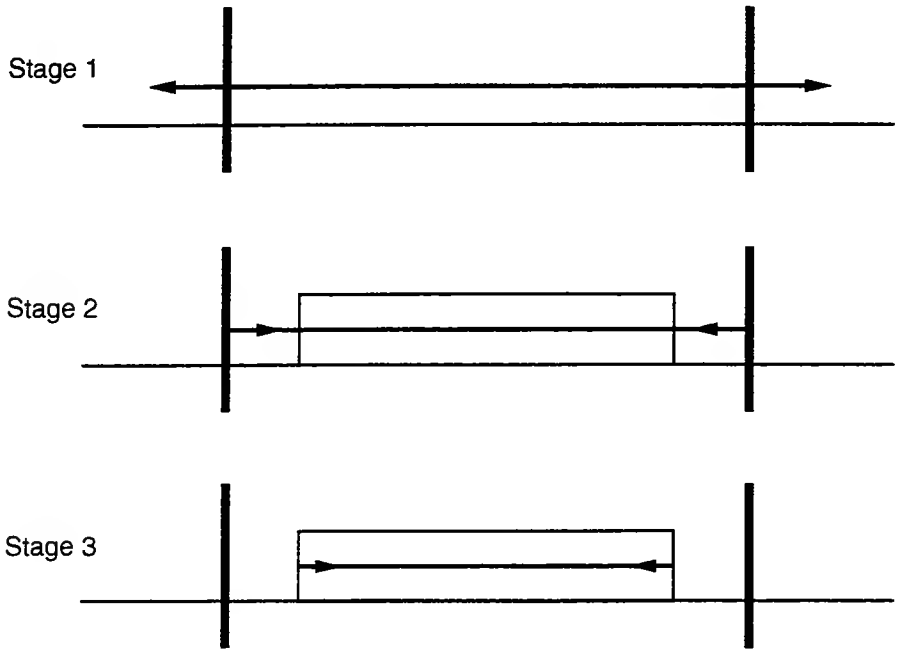
Fig. 3: Pre-tensioning of concrete



How it works:

- Steel is first tensioned between fixed abutments
- Concrete is then cast in moulds around the steel
- Steel is released from the abutments as soon as the concrete achieves sufficient strength
- The force is transferred through bond of the concrete

This method is mainly suitable for factory production where larger numbers of similar or identical units are required.

4.3 Post-tensioning Method

In the post-tensioning method the concrete is usually cast with sheathed or pneumatic-core formed ducts and is allowed to harden. It is then prestressed by tendons which are located usually within the ducts or otherwise outside. When the tendons are within the ducts, the cables are fixed by pressing grout into the ducts and thus anchoring the steel to the concrete.

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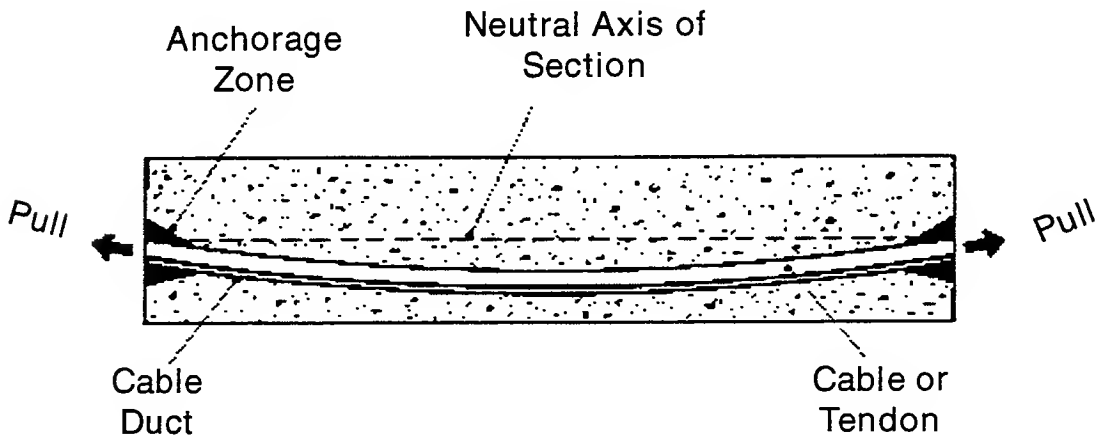
Fig. 4: Post-tensioning of concrete



How it works:

- Tendons are placed inside the structure, usually surrounded by metal sheathing to ensure free movements with relation to the structure, after the concrete has been cast.
- As soon as the concrete attains sufficient strength, tendons are stressed and their ends are anchored against the concrete to transfer the pre-stressing forces onto the structure.

Construction procedure:

- Cast beam with duct inside
- Thread pre-stressing cable
- Tension cable with hydraulic jack(s)
- Lock cable at anchorage and crop cable
- Grout cable duct
- Concrete over anchorage block

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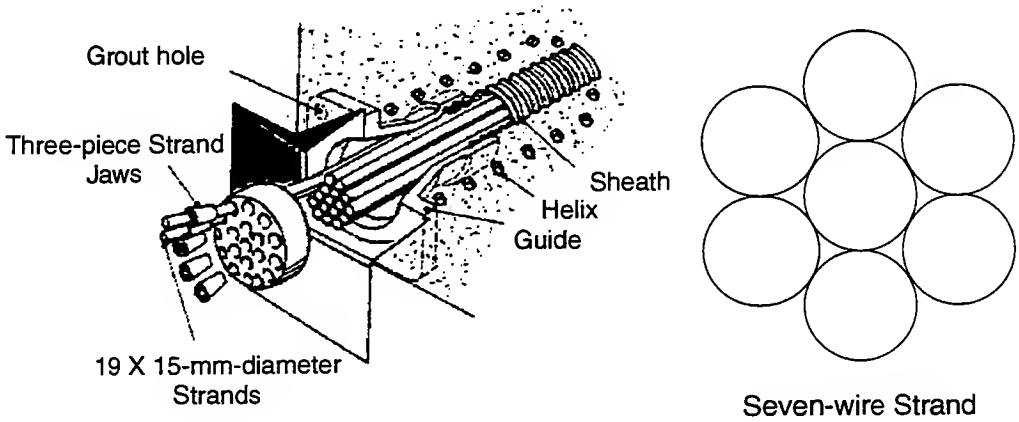
Fig. 5: Equipment: Post-tensioning



4.4 Application and Uses of Pre-stressed Concrete

Typical prestressed concrete products are:

- Bridge girder beams and segments
- Complete frames for precast building construction
- Dams, pipes, tanks
- Floor beams, reinforcing elements, joist and box sections
- Floor, road and aerodrome slabs
- Bearing piles
- Rail sleepers
- Telephone and electric power-transmission poles

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5. PRECAST CONCRETE

5.1 Purpose of Precast Concrete

It has always been common practice to place concrete directly after mixing into its permanent, final position in the form of a structure, paving or other construction. By this classical way of concreting, the so called „cast-in-place" or „cast-in-site" concrete is obtained. Today, however, attempts are made to rationalise building-construction, due to shortage of man-power and rise of wages, as well as to increasing requirements on quality and uniformity of concrete. Therefore, several processes are transferred from site to stationary installations or special plants. These processes include the preparation of aggregate, the pure batching of concrete ingredients and the mixing of fresh concrete.

A further possibility of rationalising is the precast concrete - the delivery of hardened concrete construction parts to the construction site.

Formwork and reinforcement are more and more standardised and whole systems are manufactured in mass production in a factory.

Precast concrete means:

Concrete that is cast in a mould in a factory or on site or elsewhere. When sufficiently strong, the precast elements are delivered to the construction site, where they are erected in their final (service) positions.

Precast concrete is produced, if a stationary plant supplies various building-sites or if temporary production installations were placed on a large building-site. In other words, if concrete up to the hardened stage is manufactured at a place away from the final position in the building, we are dealing with a precast concrete. Precast concrete may be plain concrete or reinforced or prestressed.

5.2 Advantages of Precast Concrete

The main benefits of precast concrete are:

- Speeding the on-site processes of construction, replacement of manual masonry work by mechanical assembling;
- Continuity of processes regardless of weather and seasonal conditions;
- Better utilisation of skills, through centralisation and intensification of activities, and use of less skilled labour through training and specialisation;
- A high degree of sophistication concerning concrete composition and material utilisation and possibilities of closer control of all manufacturing activities and qualities of materials and products;
- Maximum accuracy of mould, re-use of moulds and equipment and the best use of techniques of prestressing;
- Most economical method of placing and compacting.

Of course, apart from these advantages there are also some disadvantages, e.g. difficulties arising from the transport of large elements. The individual design of buildings is limited and high investment costs of stationary installations are required.

Therefore, it has to be decided, in each case, according to the type of building and elements, and based on economical and aesthetic aspects, whether cast-in-place or precast concrete should be chosen.

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The application of precast concrete has increased considerably in proportion to the entire concrete consumption over the past decades; but not as much as that of ready-mixed concrete.

5.3 Precast Concrete Products

Since many decades concrete products have been manufactured in mass production in small workshops, medium size or large factories, e.g. blocks, bricks, roofing tiles, steps, paving slabs, kerbs, piles, posts, well rings, manhole covers, sewer and drain pipes, cable and heating ducts, floor tiles, architectural cast stone and other ornamental concrete. Centrifuged reinforced or prestressed pipes are manufactured with diameters of up to 2 m.

To produce special effects on the surface of the concrete elements, special aggregates are used. Thus, Terrazzo is coloured marble or similar material embedded in cement paste and polished to a smooth finish as decor for walls and floors.

Granolithic concrete is a hard-wearing surface on floors, the aggregate being crushed granite or other hard stone.

5.4 Size and Handling of Precast Elements

A variety of types and sizes of precast structural units is manufactured for various types of buildings. Their mode of transportation and assembling varies as well. There are posts, beams, slabs of up to 30 m length, bridge-segments weighing up to 90 t walls and even three-dimensional cells.

Precast elements can be manufactured with decorative finishes or even water, sewage, heating, air-conditioning or electrical installations. Thus, all-equipped and finished rooms (bath-room units, kitchens, elevator-shafts etc.) can be supplied to the building site.

The manufacturing process is basically the same as for cast-in-place concrete: Mix design including possible additions, dosing, mixing, placing in moulds, compaction, possibly prestressing of reinforcement, curing, demoulding, possibly further curing. With stationary methods of production, all processes are carried out at the place of casting, before the products are removed. With conveyor methods, moulds and products are moved after each operation.

A special feature of precast concrete fabrication is the necessity for the placed concrete to pass quickly and without damage - to the next stage of production. The faster this can be achieved, the shorter the production cycles and the more efficient the utilisation of costly moulds and production installations.

5.5 Requirements to Precast Concrete

Note:
Precasting puts heavy demands on concrete properties

Precast elements are subjected to excessive stresses due to repeated transportation and storage. Stresses during demoulding, transport and assembling can be higher than later-on during loading at service.

Depending on the application in question, „green" strength, demoulding time, the ability to withstand transportation and stacking or prestressing can be decisive. These various criteria cannot be brought to a common denominator, but empirical correlation to compressive strength do exist. So, in practice, the problem is usually solved by specifying mandatory threshold ratings for compressive strength.

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Note:
The initial strength development is of vital importance for precast concrete.

Acceleration of hardening by optimising the concrete mix is subject to limitations, but it is possible in many cases to achieve sufficient early strength by using rapid-hardening cements. With a suitable brand of cement, demoulding times can be reduced by up to 16 hours, whereas prestressing can already be applied after 48 hours.

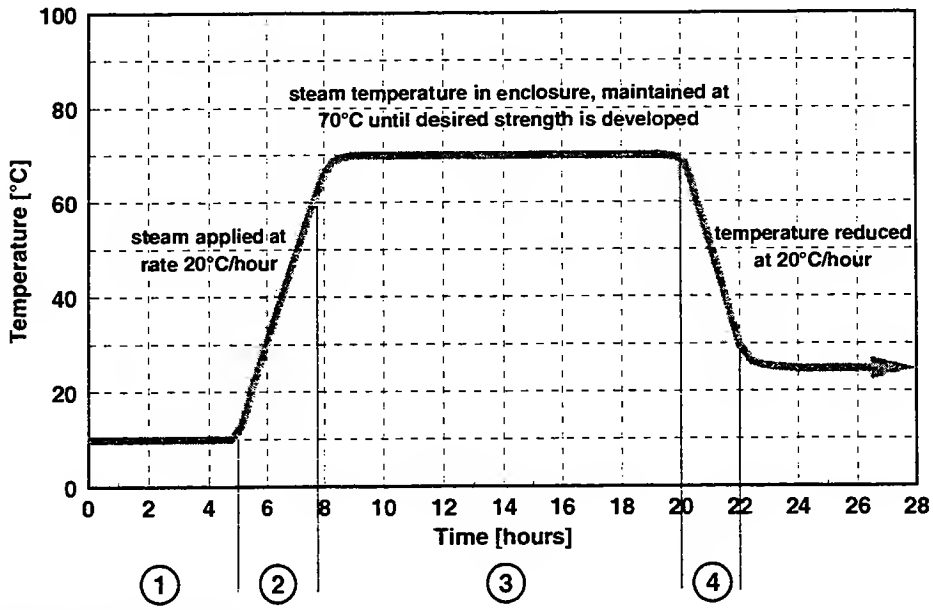
In some cases it is justified, from a technical and economical point of view, to accelerate hardening by special curing.

5.6 Curing

Various methods to accelerate hardening of precast concrete are used. However, many of these methods are cost intensive. Heat treatment in various forms is predominant. Heat can either be applied externally on the moulds, or led directly into the concrete or generated inside the concrete. The latter is accomplished by establishing an electric circuit through the fresh concrete.

External heat can be applied by vapour, hot water, oil bath or infrared radiation. Most common is the **curing in live steam** at atmospheric pressure.

Fig. 6: Time-Temperature-Diagram of a typical live steam concrete curing



① initial delay 2 to 5 hours
② temperature increase period 2 to 3 hours
③ constant temperatureperiod 6 to 12 hours
④ temperature decrease period 2 hours

The steam-curing must have an optimum cycle adapted to the element to be produced. This cycles consists of:

- An initial delay prior to steaming (at least 2 hours)
- Increasing temperature (by not more than 20 °C per hour)

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- Maintaining a constant maximum temperature (60 to 80 °C)
- Decreasing temperature (by not more than 20 °C per hour)

By **high pressure steam-curing** in autoclaves at 160 to 190 °C and corresponding pressures of about 5 to 12 bars, the 28-day strength is reached within 24 hours. This hydrothermal treatment produces reactions between silica of the aggregates and lime of the cement that do not occur with normal or steam curing at atmospheric pressure. A chemical bond between cement and aggregate improves, of course, the concrete properties. Steam-curing in an autoclave is not any more expensive than steam-curing at atmospheric pressure. The investment costs, however, are significantly higher. In an autoclave, the sizes of the produced elements are limited. Therefore, the hydrothermal process is only rarely applied for products of ordinary concrete. It is, however, used for the manufacture of lime-siliceous wall blocks and gas concrete.

Generally, all Portland cements are suitable for precasting with or without heat-treatment. The relative increase of initial strength by heat is usually higher for cements producing low heat (e.g. slag cements). High alumina cements are not suited for heat treatment.

6. LIGHTWEIGHT CONCRETE

6.1 Purpose of Lightweight Concrete

Lightweight concrete instead of normal weight concrete is mostly used, if an especially good thermal insulation of walls and roof is required. Due to the shortage of energy, this insulating concrete continually gains in importance.

Lightweight concrete is also used to reduce the weight of constructions, e.g. bridges. This load bearing concrete is called structural lightweight concrete.

Both of these purposes can be attained by one and the same material - by a structural and insulating concrete. This type of application replaces the use of a combination of different materials with different properties such as:

- Load-bearing skeletons with insulating panels, and
- Thermal insulating coats on ordinary concrete or
- Sandwich panels.

According to its purpose, different bulk weights of the concrete are chosen:

Table: Bulk density of lightweight concrete

Concrete application	Bulk density, kg/m^3
Insulating concrete	300 - 800
Structural concrete	1400 - 1900
Combined application	800 - 1400

For the latter two purposes reinforced or even prestressed concrete can be used. By decreasing the bulk weight, the thermal insulating capacity increases, strength, however, diminishes.

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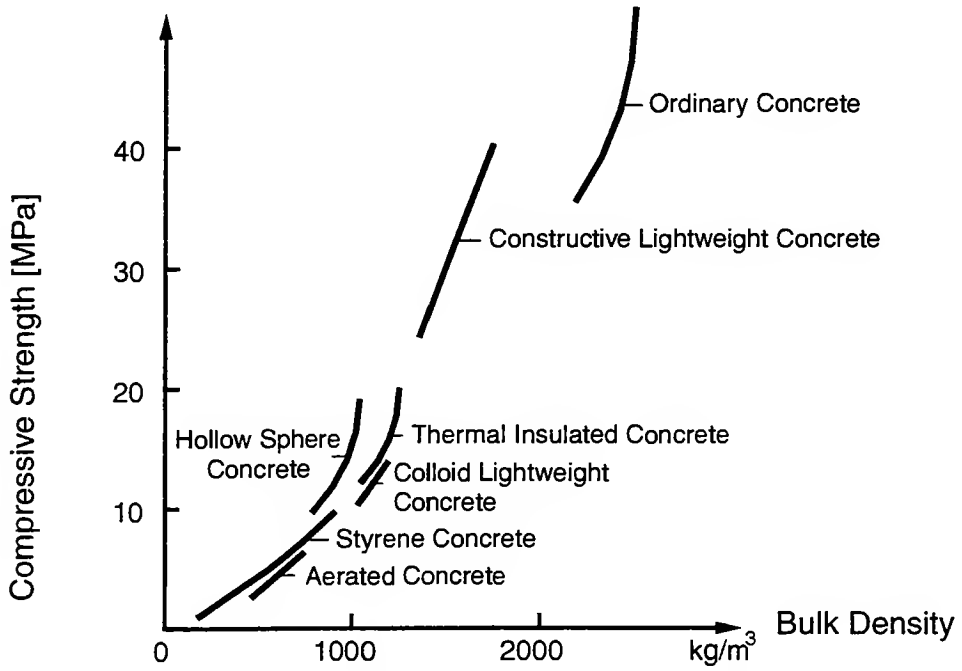
Fig. 7: Relation of strength and bulk density of different concrete types



Physical properties lightweight aggregate

Aggregate type	Particle shape, surface texture	Density [kg/m³]	Bulk density [kg/m³]	24 h-water absorp. [wt-%]	Concrete Compr. strength [MPa]	Concrete Unit weight [kg/m³]
Aggregates for high strength lightweight concrete, > 15 MPa						
Expanded clay	Rounded and slight rounded part.	Coarse: 600 to 1600 Fine: 1300 to 1800	300 to 600	5 to 30	10 to 60	1000 to 1700
Exp. shale and slate	Angular slightly rounded, smooth surface	Coarse: 800 to 1400 Fine: 1600 to 1900	400 to 1200	5 to 15	20 to 50	1300 to 1600
Fly ash	Similar to exp. clay	1300 to 2100	600 to 1100	20	30 to 60	15000 to 1600
Foamed blast furnace slag	rough and open pored	1000 to 2200	400 to 1100	10 to 15	10 to 45	1800 to 2000
Sintered colliery waste	Angular open pored	1000 to 1900	500 to 1000	15	10 to 40	1400 to 1600
Aggregates for medium strength lightweight concrete, 3.5 - 15 MPa						
Pumice	Rounded, rather smooth surface	550 to 1650	350 to 650	50	5 to 15	1200 to 1600

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Aggregates for high strength lightweight concrete, 0.5 - 3.5 MPa						
Perlite	Rounded shape, rough surface	100 to 400	40 to 200	---	1.2 to 3.0	400 to 500
Vermiculite	Cubical	100 to 400	60 to 200	---	1.2 to 3.0	300 to 700

6.2 Types of Lightweight Concrete

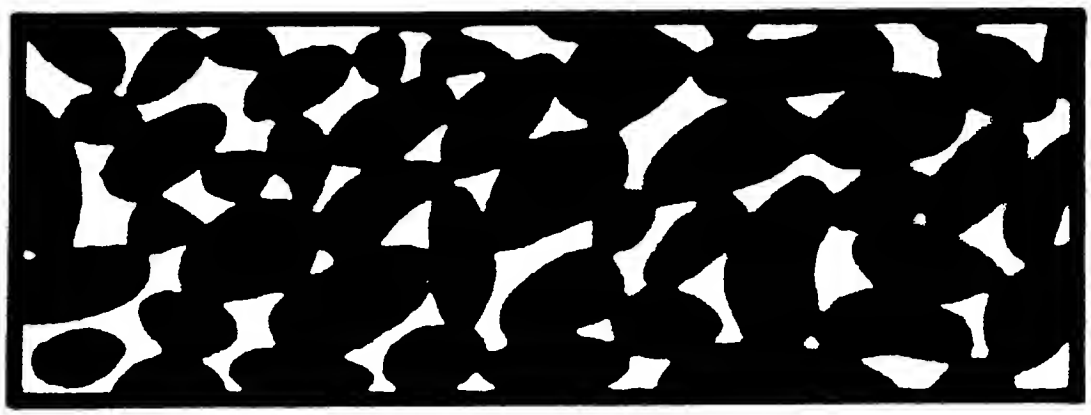
The best and least expensive thermal insulation is achieved with occluded air which does not contribute to heat convection; just like animal fur and human hair, where not the hair itself is the insulating medium but the air trapped in between. All types of lightweight concrete contain air voids, but the manner of incorporation varies.

6.2.1 „No Fines Lightweight Concrete"

The simplest way to obtain a lightweight concrete is one that would be the greatest mistake for ordinary concrete:

Only coarse aggregates, i.e. only one particle size fraction with 35 - 50 % voids, is used. Just enough cement paste to bind the grains of the aggregates but not to completely fill the voids, is added.

Fig. 8: View of 'no fineness aggregate' lightweight concrete (water draining concrete)



6.2.2 Concrete with Lightweight Aggregates

The voids are inside the aggregates

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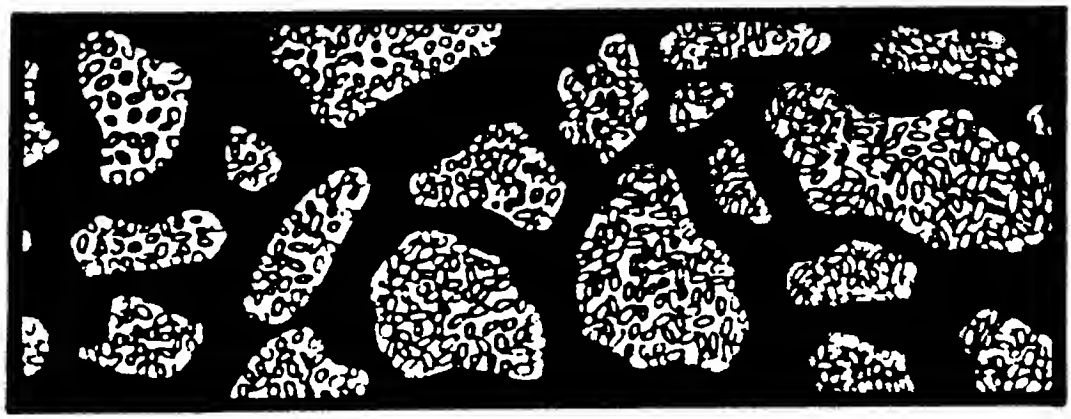
Fig. 9: View of lightweight concrete with lightweight aggregates



6.2.3 Cellular (aerated) Concrete

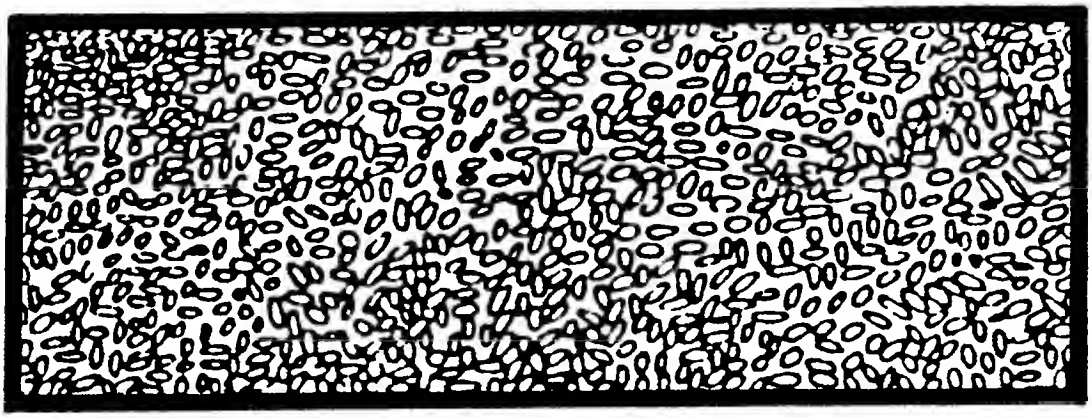
- The pores are within the cement paste or mortar.
- They are produced during mixing by foaming agents. The product is called aerated or foamed concrete.
- Another method is the generation of gas after casting by a chemical reaction within the unhardened mix. This type of concrete is called gas concrete.

„No fines concrete" can also be produced with lightweight aggregate. On the other hand, lightweight aggregate can be combined with aerated mortar. In both cases, lower bulk weights are obtained than in the ordinary lightweight aggregate concrete (800 to 1600 kg/m^3 with compressive strengths from 1.5 to 15 MPa).

When lightweight concrete is used, the differences in moisture content, desiccation and shrinkage in relation to ordinary concrete must be considered. According to the pore structure, the absorption capacity and frost resistance of the lightweight concrete may be higher or lower than those of ordinary concrete. In lightweight concrete the steel reinforcement is less protected against corrosion and therefore very often has to be covered by a protective coating.

Poudre aluminium qui réagit avec OH$^-$ => H_2

Fig. 10: View of cellular concrete



6.3 Concrete with Lightweight Aggregates

Porous aggregates are sometimes found as natural rocks, mostly of volcanic origin; they can also be produced artificially from natural raw materials, industrial by-products or wastes. Even organic substances in the form of natural products, waste-products, refined or synthesised fillers are used in lightweight concrete.

Artificial lightweight aggregates have different trade names, even for similar products.

These aggregates are usually crushed and screened as well .The bulk weight of finer particle size fractions generally is higher than that of coarser fractions. Often, coarse lightweight aggregate is used together with normal sand.

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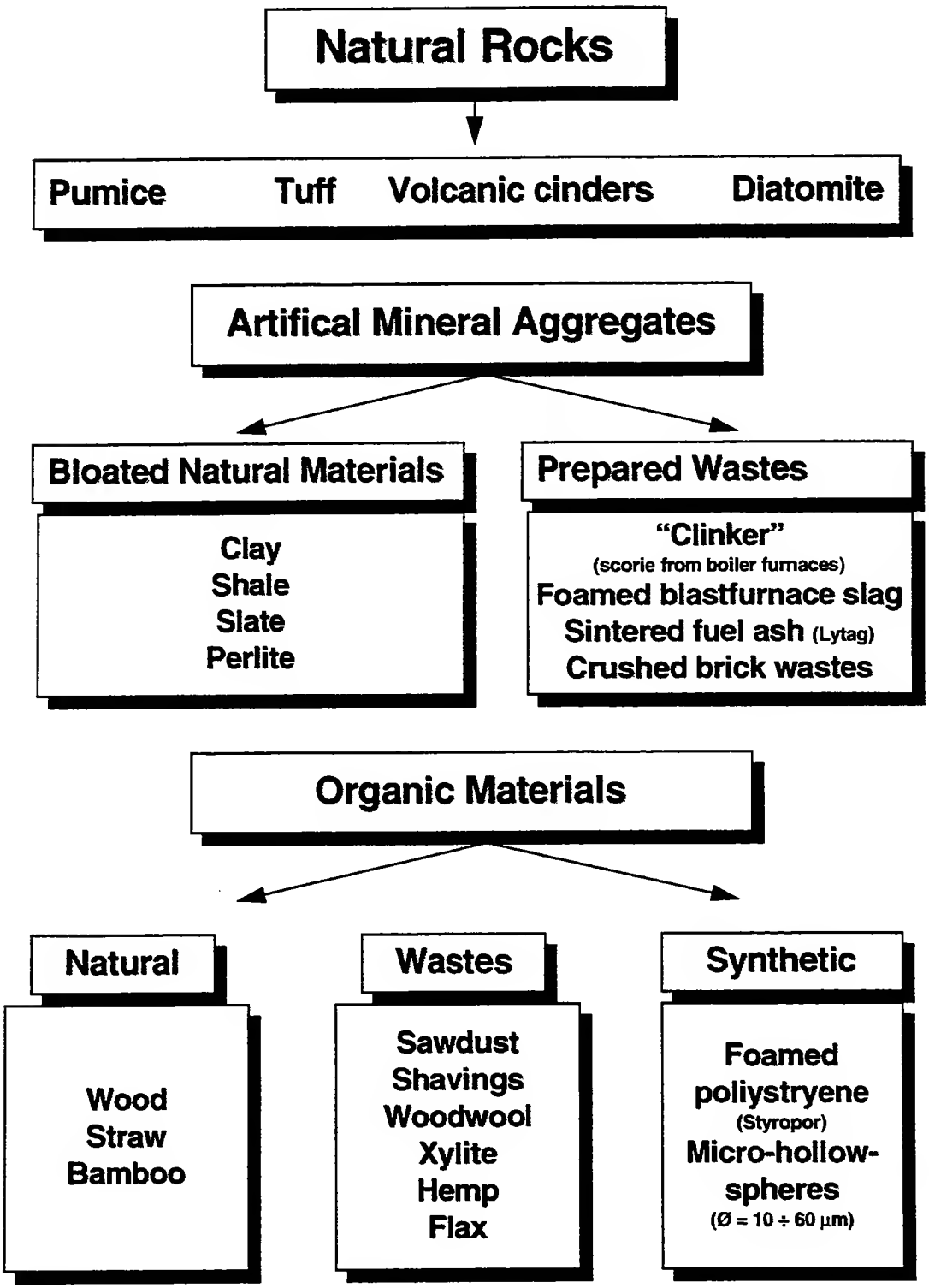
Fig. 11: Lightweight aggregates



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Concrete manufacture with lightweight aggregate is principally subject to the same rules as ordinary aggregates; however, for mix designs, placing and compaction some particularities must be observed. Especially the absorption of mixing water by porous aggregates and the fast stiffening of concrete must be taken into account when dosing water. Therefore, it is recommended to saturate the aggregates with water before dosage. Very light aggregates tend to float on top of the concrete mix, resulting in segregation. This can be prevented by the addition of admixtures.

Due to these particularities in mixing and placing, lightweight aggregates are preferably used in precast concrete rather than in cast-in-place. It is applied in the manufacture of bricks, wall panels, floor and roof slabs.

It should be mentioned that sometimes natural products, such a straw, hemp, flax, bamboo, wood shavings and sawdust are used in lightweight concrete manufacture. Insulating panels and blocks (Durisol) are industrially manufactured with wood-wool, sawdust and shavings together with cement.

In extremely light concretes small spheres of expanded Polystyrene („Styropor") replace the mineral lightweight aggregates. The bulk weight of Styropor can be as low as 15 kg/m^3.

6.4 Aerated (foamed) Concrete

The simplest way to obtain a high pore content is to use concrete mixes with a high water content. Through quick hardening by vapour and desiccation, voids are created. A higher pore volume (up to 30%) is attained by foaming agents (hydrolysed proteins or resin soaps). Either a separately produced foam is injected into the concrete mix by pressure, or the foam is produced in the quick rotating concrete mixer by the direct addition of a foaming agent. The concrete mix must contain only fine aggregates; generally, a cement mortar with sand up to 3 mm. This procedure can easily be carried out on site and is used e.g. for roof insulation. It is, however, not widely applied as it is very difficult to regulate the pore content and the mechanical properties. The relations between bulk weight and compressive strength is recorded in the following figure:

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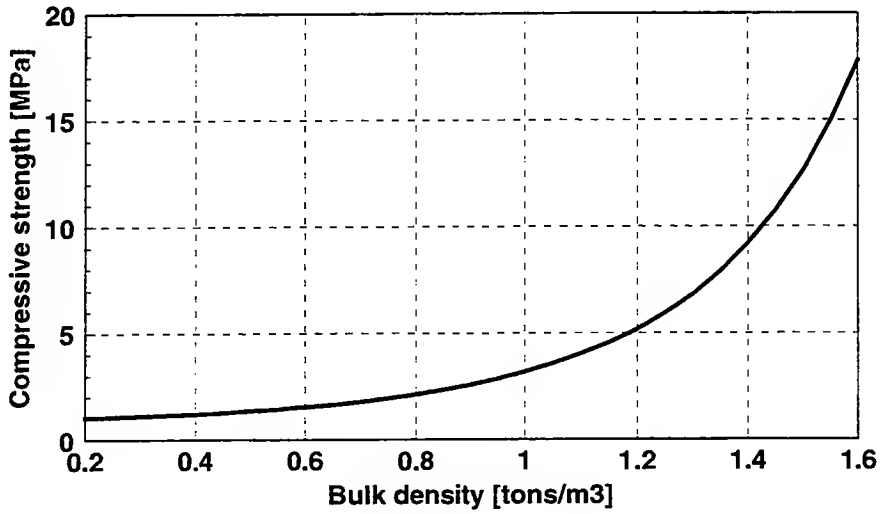
Fig. 12: Relation of strength to density to foamed concrete



Aerated concrete (up to 30 vol.-% pore content) has to be distinguished from air-entraining concrete which, due to its 3 to 5 vol.-% pore content, is used for structural purposes demanding high frost/thaw resistance.

6.5 Gas Concrete

The voids are developed within the unhardened mix, usually by hydrogen generated by the action of lime on fine Aluminium powder used as an admixture (0.2% of cement weight). Only occasionally used is oxygen from peroxides or acetylene from carbide.

Cement and sand (type „Siporex") or quick lime with various siliceous components (type „Ytong" with ground shale or fly ash) are applied as basic materials. In some procedures a combination of cement and lime in various proportions is used („Durox", „Calsilox", „Hebel"). In order for the mass to be bloated by gas, the components must be very fine (to compare with the rising of bread). Therefore, in gas concrete a mortar is used and not a concrete mix with coarse aggregates. It is important that the generating of gas is synchronised with the stiffening of the mass, so that the gas cannot escape and the cake expands without collapsing when gas generation has stopped. Before hardening, the demoulded blocks are cut with wires into blocks or slabs of desired sizes.

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Curing generally (always when lime is applied) takes place in autoclaves. By a chemical bond during the hydrothermal process, the aggregates are incorporated into the binding material. This fact, together with the regular pore structure create a good relation between strength and bulk density, for example:

Note:
At bulk densities of 800 kg/m^3 equal compressive strengths as for lightweight concretes with bulk densities of more than 1200 kg/m^3 are reached.

With gas concrete mainly wall blocks, reinforced roof slabs and wall panels, even with a precast surface coating, are produced.

7. HIGH PERFORMANCE CONCRETE (HPC)

This is a term usually used to describe concretes made with carefully selected high quality ingredients, optimised mixture designs, and which are batched, mixed, placed, compacted and cured to the highest industry standards. Typically such concretes will have a low water/cementing materials ratio of 0.40 or less, and generally much less. Most HPCs have water/cementing materials ratios in the range of 0.25 to 0.35 with 0.30 generally die optimum value. The achievement of such low water/cementing materials ratios depends on the use of large quantities of superplasticizers, the use of which leads to high workability, another common characteristic of mixes.

Because of the formulations used, HPC almost always has a higher strength than normal concrete. However, strength is not always the prime required property, and HPC is valuable where any of the following properties are required:

- High strength
- Very-high strength
- High early strength
- Abrasion resistance
- Low permeability
- Low gas and water absorption
- Low diffusion coefficient
- High resistivity
- Resistance to chemical attack
- High modulus of elasticity
- High resistance to freezing and thawing damage
- Volume stability
- Inhibition of bacterial or mould growth

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In 1993 the ACI Committee on High-Performance Concrete proposed the following Definition of High Performance Concrete:

High Performance Concrete:

Concrete meeting special performance requirements which cannot always be achieved routinely using only conventional constituents and normal mixing, placing and curing practices.

These requirements may involve enhancements of the following:

- Ease of placement and compaction without segregation
- Long-term mechanical properties
- Early-age strength
- Toughness
- Volume stability
- Service life in severe environments

7.1 High Strength Concrete

The definition of high strength changes from time to time. According, to the American Concrete Institute, high strength is defined as that over 41 MPa compressive strength. This value was adopted by ACI in 1984 but is not yet hard and fast, because ACI recognises that the definition of high-strength varies on a geographical basis. Prof. J. Francis Young of the University of Illinois at Champaign-Urbana has developed a strength classification system that, though not yet adopted by a recognised authority, is a helpful tool for describing high-strength concretes (see following table).

Strength classification of concrete

Parameter	Conventional concrete	High-strength concrete	Very high strength concrete	Ultra high-strength concrete
Strength, MPa	<50	50 - 100	100 - 150	>150
Water/cement ratio	>0.45	0.30 - 0.45	0.25 - 0.35	<0.25
Chemical admixtures	not necessary	WRA/RWR *)	HRWR **)	HRWR **)
Mineral admixtures	not necessary	Fly ash	Silica fume	Silica fume
Permeab. coeff. (water), cm/s	$>10^{-10}$	10^{-11}	10^{-12}	$<10^{-13}$
Freeze-thaw protection	needs air entrainment	needs air entrainment	needs air entrainment	no freezable water

*) **WRA = Water Reducing Admixture;** **) **HRWR = High Range Water Reducer**

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Compositions of typical HPC mixtures can be seen in the following table:

Typical HPC mixtures

Component	Mix 1	Mix 2	Mix 3
Water, kg/m^3	195	135	130
Cement, kg/m^3	505	500	513
Fly ash, kg/m^3	60	---	---
Silica fume, kg/m^3	---	30	43
Coarse aggregate, kg/m^3	1030	1100	1080
Fine aggregate, kg/m^3	630	700	685
Water reducer, ml/m^3	975	---	---
Retarding agent, L/m^3	---	1.8	--
Superplasticizer, L/m^3		14	15.7
w/c ratio	0.35	0.27	0.25
Strength at 28 d, MPa	65	93	119
Strength at 91 d, MPa	79	107	145

8. SHOTCRETE

8.1 INTRODUCTION

Shotcrete is the name given to mortar or concrete conveyed through a hose and pneumatically projected at high velocity onto a backup surface. The force of the jet impacting on the surface compacts the material so that it can support itself without sagging or sloughing, even on a vertical face or overhead. Other names are also used for some types of shotcrete, e.g. *gunite,* but only *sprayed concrete* is sufficiently general and is indeed the preferred term in the European Union terminology.

The properties of shotcrete are no different from the properties of conventionally placed mortar or concrete of similar proportions: it is the method of placing that bestows on shotcrete significant advantages in many applications. At the same time, considerable skill and experience are required in the application of shotcrete so that its quality depends to a large extent on the performance of the operators involved, especially in control of the actual placing by the nozzle.

Because shotcrete is pneumatically projected on a backup surface and then gradually built up, only one side of formwork or a substrate is needed. This represents economy, especially when account is taken of the absence of form ties, etc. On the other hand, the cement content of shotcrete is high. Also, the necessary equipment and mode of placing are more expensive than in the case of conventional concrete. For these reasons, shotcrete is used primarily in certain types of construction: thin, lightly reinforced sections, such as roofs, especially shell or folded plate, tunnel linings, and prestressed tanks. Shotcrete is also used in repair of deteriorated concrete, in stabilising rock slopes, in encasing steel for fireproofing, and as a thin overlay on concrete, masonry or steel. If shotcrete is applied to a surface covered by running water, an accelerator producing flash set, such as washing soda, is used. This adversely affects strength but makes repair work possible. Generally, shotcrete is applied in a thickness up to 100 mm.

There are two basic processes by which shotcrete is applied. In the *dry mix process* (which is the more common of the two, in many parts of the world) cement and damp aggregate are intimately mixed and fed into a mechanical feeder or gun. The mixture is then transferred by a feed wheel or distributor (at a known rate) into a stream of compressed air

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in a hose, and carried up to the delivery nozzle. The nozzle is fitted inside with a perforated manifold through which water is introduced under pressure and intimately mixed with the other ingredients. The mixture is then projected at high velocity onto the surface to be shotcreted.

8.2 DRY MIX AND WET MIX SYSTEMS

8.2.1 Dry Mix

Two distinct methods of spraying concrete exist, namely the dry mix and wet mix systems. In the dry mix system, a mixture of cement, sand and large aggregate, if used, but with no added water, is fed into a special mechanical feeder - the gun - and metered into a high pressure delivery hose. The material is then carried on a stream of compressed air through the hose to a special nozzle which is equipped with a water injection system operated by a valve which is under the direct control of the nozzleman. The function of the nozzle is to convert the incoming stream of dry mixed material into concrete of the correct consistency and project it at a specified point, some distance away, where it will impact on a surface and stick there.

The original dry mix concrete spraying system has been developed over the years to a high degree of efficiency but several other types of gun are now in use. Their mechanisms are different but the basic principle is the same. This is to feed dry mix into a chamber where high pressure air blasts it down a hose.

8.2.2 Wet Mix

In the wet mix concrete spraying system all the materials, including water, are thoroughly mixed before being introduced into a pump. This can be a simple, non-mechanical device such as a CEM pump or any one of a number of mechanical pumps, usually of the smaller variety, currently available. By such means the concrete is pumped or squeezed along the delivery hoses. It is, of course, travelling at a relatively low velocity but the nozzle converts the pumped concrete into shotcrete by the introduction of compressed air - not water as in the dry mix system - which blows the concrete at high velocity onto the surface to be coated.

8.2.3 System Advantages

Each system possesses certain advantages. The wet mix system would appear to diminish the reliance on operator technique at the nozzle as the water is added at the mixer as in ordinary concreting practice. However, very strict control is required to ensure that the mix possesses the required characteristics pumpable but not too pumpable! Certainly the wet mix system produces less dust and rebound and therefore finds favour in tunnels where its high output and ability to produce a more recognisable large aggregate concrete are of advantage. But it would appear that the lead that has been built up by the dry mix system will ensure that its popularity is maintained.

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8.3 Shotcrete and Rebound

Rebound consists mainly of sand and aggregate - with a small cement content - which does not adhere to the surface of application, the reinforcement or the fresh concrete layer itself, but which bounces back out of the placing area or otherwise becomes detached from the surface. Rebound percentages can be anything from 5% to 50% depending on various factors. It is expensive and wasteful and rather unpleasant for the nozzleman. Moreover, there is always the danger that instead of falling clear the rebound may become lodged particularly in the corners of the work and behind the reinforcement. Should rebound remain trapped in the finished work this results in zones of weakness.

8.4 Strength and other Properties

Properly applied shotcrete is a versatile structural material, possessing great durability, and is capable of excellent bond with concrete, masonry, steel and other materials. A mix designed for placing by traditional methods can show up to 30% increase in strength if applied as gunite and one rarely hears of any shotcrete strengths below 30 MPa while strengths in excess of 70 MPa have frequently been obtained. Such strengths are obtained partly by virtue of the low water/cement ratio which falls within the range 0.35 to 0.40 by weight, which is lower than most values for conventional concrete mixes and partly by the packing action as each particle is impacted into the material already in place.

This same action, in part, explains the high bond strength of shotcrete to other material faces which should be presented in a clean, roughened and damp state. If, for example, concrete was sprayed onto such a face, it should be impossible to pull the hardened shotcrete away without pulling the face of the original concrete away with it. The compressive strength and bond strength of shotcrete are two of its more important characteristics.

Other properties include flexural-tensile strength at 4 to 5 MPa drying shrinkage in the range 0.05 to 0.10% density generally 2240 kg/m^3 to 2320 kg/m^3 and superior resistance to acid attack and abrasion. Shotcrete has performed well in permeability tests and one test at least has indicated that a 50 mm thick gunite specimen would show no water percolation at all up to 700 kN/m^2. A very valuable property of shotcrete from the practical point of view is its ability to support itself in thickness of up to 50 mm at a time in the overhead position and, of course, to a much greater thickness on a vertical surface without the use of a face shutter. Thus, thickness can be reduced with consequent savings and alternative methods of approaching certain types of work considered.

8.5 Uses of Shotcrete

Ever since its introduction, shotcrete has been used for repair works. Structural repairs are often needed following fire damage, impact or abrasion including erosion of marine structures, or due to poor concrete in the original construction, incorrect positioning of reinforcement causing lack of cover, design faults, chloride attack or simply old age.

Cosmetic or superficial repairs are sometimes required because of poor finishing of the original construction, honeycombing of the concrete, incorrect alignment due, for instance, to shutter slips, or wear and tear, while refractory repairs often form part of maintenance programmes in industrial installations to the linings of furnaces, reactors or chimney stacks which are subject to chemical attack and very high temperatures.

A shotcrete repair carried out correctly will produce a high strength material with a complete bond to the original structure provided by a compact and portable plant which can be operated at a convenient point some way away from the area to be repaired.

It should be stressed that the preparation of the works is of paramount importance. Failure can normally be traced back to insufficient cutting out of unsound concrete or insufficient

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cleaning of existing steelworks both of which will lead to a failure of the bond between the shotcrete and the parent surface or reinforcement.

Many brick and masonry arch structures such as bridges and viaducts were constructed 100 years ago or even more. It is a tribute to their designers and builders that they have lasted so long but time does leave its mark and the loading imposed upon them today are often far in excess of their design loadings. Such structures can be strengthened by spraying a new concrete arch underneath the original arch with minimal access requirements and no shuttering.

In original construction, perhaps the best known use of shotcrete is in the building of swimming pool shells where the flexibility of the medium is reflected in freedom of design. The techniques developed for building pools have more recently been adapted for the construction of skateboard parks with some success.

Some excellent examples of imaginatively designed thin section shell roofs also exist and pre-shot gunite elements have been produced ranging from thin sewer-lining segments to large curved capping pieces for North Sea Oil structures.

The construction of houses and even small schools is now being developed using reinforced shotcrete each side of a lightweight insulating layer.

As a protective coating, shotcrete has many uses. When an excavation is opened up, weathering of the newly exposed face quickly takes place. On an embankment surface deterioration can be a long term problem while in a steep sided cutting the falling of even small pieces of rock can cause serious danger to those working below. A relatively thin coating of lightly reinforced shotcrete can seal the surface and protect it particularly if the treatment is carried out at an early stage. Many steel pipelines, particularly for North Sea gas and oil are coated with shotcrete for protection and negative buoyancy where they cross rivers on their journey across the country. The portable nature of the plant makes it ideal to carry out such work on the riverbank and circular encasements are completed without shuttering.

It is said that if we have space for a man and a hose, we can place concrete no matter what the problems of access. Small amounts of concrete are often needed to fill pockets, for example, after completion of surrounding structures and this can be done without recourse to buckets and ropes and without the need for shutters or letterboxes. With the dry mix system, concrete can be blown through hoses in excess of 100 metres horizontally or vertically.

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8.6 Summary: Shotcrete

The production of shotcrete relies to a great extent on the skill of the operators and there is a greater inherent variability than with in-situ concrete. The design should allow for this.

As far as quality control is concerned, the short answer is to find a good contractor and a good specification and make sure that the contractor keeps to the specification.

The process has been with us a long time. The machinery and techniques have evolved, the product exhibits certain very interesting and useful characteristics and, with proper use and control can be made to serve the contractor, the engineer, the client and ultimately the industry of which we are all a part.

9. ROLLER COMPACTED CONCRETE (RCC)

9.1 INTRODUCTION

Roller Compacted Concrete (RCC), rather than a new material is a new technique for building in concrete, which has proved very suitable for dam and road construction. Its relevant to the cement industry must be made clear from the beginning: It opens up excellent prospects to compete advantageously with traditional non-cementitious 'foes': earth and rockfill dams and asphalt pavements.

By the end of the '60s it was ascertained in the USA that less than 10% of the newly built dams were concrete dams. The implication for their future was clear: Either they were constructed more rapidly and economically or they ran the risk of becoming a species under threat of extinction. This challenge generated a more systematic search for a solution, the result of which is RCC.

9.2 WHAT IS RCC AND WHAT ARE ITS ADVANTAGES ?

Basically, the technique of building dams with RCC consists of placing concrete - of dry consistency - in layers 0.30 to1.00 m thick, which are subsequently compacted by several passes of heavy vibrating rollers. This process can be carried out without interruption along the entire length of the dam and, thus, the traditional system of erecting dams in blocks, separated by joints, is substituted by one in which the elevation of the structure is achieved uniformly. This results in a simpler RCC construction layout, with associated reductions in materials and labour costs. Even more important is that the rate at which concrete is placed and compacted is much higher than for conventional concrete dams, coming close to those achieved for earth-fill dams. As an example, in Upper Stillwater Dam (USA), RCC production peaks of 8400 m^3/day were attained. Building dams in RCC brings about huge savings to the civil engineering construction, the cost of which may become only one third of that necessary in the case of the traditional concrete dams techniques. Even more important, a project in RCC may result more economical that the alternative earth-fill dam.

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9.3 DIFFERENT RCC CONCEPTS

As a consequence of its evolution, three different concepts of RCC have emerged:

- **Lean RCC**, developed by the US Army Corps of Engineers
- **Moderate in Paste RCC** or 'Rolled concrete dam (RCD)' according to Japanese practice and designation
- **Rich in Paste RCC** developed in the UK and adopted by the US Bureau of Reclamation

9.4 SUITABLE MATERIALS FOR RCC

9.4.1 Binders

Binders used in the construction of RCC dams are mainly based on Portland cement, be it pure or blended with additions. When pure Portland cement is used as the only binder, it is preferred to use cements of low or moderate heat of hydration (ASTM Types IV and II).

There is a preference for blended binders containing fly ash due to its beneficial effect on workability, the fly ash content being high, generally over 40% of the total The preferred addition for RCC is the low Ca0 fly ash (ASTM Type F); it can be mentioned that, for this application, fly ash of substandard quality can be used. To a smaller degree, blast furnace slag and natural pozzolans have also been used.

In different projects the fly ash has been incorporated into the mix in various ways: as part of the cement, as a separate concrete ingredient or a combination of both (this solution has been used in Santa Eugenia Dam in Spain).

A very important property of a binder to be used in RCC dam construction is extremely low heat of hydration. For instance, for a dam project in Colombia, a binder having a heat of hydration 40 cal/g at 7 days was specified, a limit which is considerably stricter than the 60 cal/g specified as a maximum for ASTM Type IV, Low Heat cement. This explains the convenience of adding large amounts of fly ash into the binder to keep the heat generation low.

9.4.2 Aggregates

Aggregates that fulfil the typical quality requirements for conventional concrete will be adequate for RCC. Of course the deleterious effects of the Alkali-Aggregate reaction should be avoided, its likelihood being low when moderate to high pozzolanic materials are included in the binder.

A special requirement is that the strength of the coarse aggregate must be sufficient to withstand the heavy compaction pressures exerted on the material by the rollers.

Regarding the grading the following applies:

- The preferred maximum size seems to be around 80 mm, although larger aggregates (up to 150 mm) and smaller (about 40 mm) have also been used.
- The grading curve for RCC differs from that of conventional concretes, particularly in the fine fraction. In particular, for **Lean RCC** a high proportion of material below 0.075 mm (between 4 and 10%) is recommended.

Some experts claim that the uniformity of the grading is not as vital for RCC as it is for conventional concrete and that the aggregates could be stored directly as a single fraction or eventually as only two fractions (see cases of Monkville and Saco de Olinda dams in the following table). That table shows that, in general, the aggregates should be stored in between 3 to 5 different fractions depending on the maximum size.

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Some examples of aggregates for RCC for dams

Dam (country)	Max. size, mm	No. of fract.	Fraction size, mm
Ohkawa (Japan)	80	4	0-5, 5-20, 20-40, 40-80
Shin-Nakano (Japan)	50	4	0-5, 5-20, 20-40, 40-80
Shin-Nakano (Japan)	150	5	0-5, 5-20, 20-40, 40-80, 80-150
Castilblanco (Spain)	50	4	0-5, 5-12.5, 12.5-25, 25-50
Morales (Spain)	80	4	0-5, 5-20, 20-40, 40-80
Morales (Spain)	40	3	0-5, 5-20, 20-40
Erizana (Spain	100	4	10-6, 6-20, 20-50, 50-100
Monkville (USA	76	2	0-25, 25-76
VVillow Creek (USA)	76	3	0-20, 20-38, 38-76
Upper-Stillwater (USA)	38	3	0-4.8, 4.8-19, 19-38
Saco de Nova Olinda (Brazil)	76	2	0-30, 30-70

9.4.3 Water

The specifications for the quality of mixing and curing water for RCC are the same as for conventional concrete. Regarding quantity, the water content of RCC mixes ranges normally between 90 and 120 kg/m^3; sometimes these low values create problems if the aggregates are wet, since they may carry more water than is strictly necessary to prepare the mix (e.g. Pangue Project, Chile).

9.4.4 Chemical Admixtures

Due to the very dry consistency and high amount of fineness, the use of chemical admixtures has proved problematic for Lean RCC mixes.

For the other two types of RCC the chances of successful use of chemical admixtures are higher. For instance, air-entraining and water-reducing agents are introduced into all Japanese RCC mixtures. A water-reducing and retarding agent, at a relatively high dosage (0.9 to 1.3%), was used for Elk Creek Dam with good results.

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10. FIBRE REINFORCED CONCRETE

In recent years, numerous attempts have been made to reinforce concrete by incorporating randomly distributed fibres, when fibre-reinforced plastic materials have already widely been used. For such reinforcements four groups of fibres may be considered: steel, plastic, glass and (in former times) asbestos fibres. Other special fibres, such as carbon, are far too expensive for concrete.

For reinforcing materials the important factor is the Young's modulus (= initial tangent modulus) of the fibres. If it is smaller than that of concrete, the fibre does not help to increase the tensile strength of the concrete. The various materials referred to possess the following values of Young's modulus:

Table: Young's modulus of fibres

Fibre type	Young's modulus [GPa]
Concrete	30
Polypropylene fibres	≤5
PVA fibres	30
Glass fibres	70
Steel fibres	210
Asbestos fibres	150

10.1 Different Types of Fibres

The most commonly used man-made fibres have been *steel and polypropylene,* principally in concrete, and glass, principally in cement mortar for thin section applications. Properties of some of the commonly used fibres are given in the following table:

Physical properties of various types of fibres and matrices

Material	Spec. gravity, g/cm^3	Young's modulus, GPa	Tensile strength, MPa	Elong. at breaking point, %
Acrylic	1.10	2.1	210 - 420	25 - 45
Asbestos (Chrysotile)	2.55	150	200 - 1800	2 - 3
Carbon, high modulus	1.9	380	1800	0.5
Carbon, high strength	1.9	230	2600	1.0
Cellulose	1.5	10 - 40	500	---
Cotton	1.5	5	420 - 700	3 - 10
Glass (Cem-FIL filament)	2.7	80	1050 - 3870	1.5 - 3.5
Nylon	1.1	4.2	780 - 850	16 - 20
Polyester	1.4	8.5	750 - 880	11 - 13
Polyethylene, high modulus	0.96	15 - 40	300 - 700	3 - 10
Polypropylene	0.91	3 - 15	560 - 780	8
Rayon	1.50	7.3	420 - 630	10 - 25
Steel	7.86	200	280 - 420	3.5
OPC paste	2.0 - 2.2	10 - 20	2 - 6	0.01 - 0.05
OPC concrete	2.30	20 - 35	1 - 4	0.005 - 0.015

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10.1.1 Steel-fibre Reinforced Concrete

A number of steel-fibre types are available as reinforcement. Round steel fibres, the commonly used type, are produced by cutting round wires into short lengths. The typical diameters lie in the range of 0.25 to 0.75 mm. Steel fibres having a rectangular cross-section are produced by slitting the sheets about 0.25 mm thick. For improving the mechanical bond between the fibre and matrix, indented, crimped, machined and hook-ended fibres are normally produced. The aspect ratio (= length/diameter) of fibres which have been employed vary from about 30 to 250.

Fibres made from mild steel drawn wire with a diameter from 0.3 to 0.5 mm have been practically used. Round steel fibres are produced by cutting or chopping the wire; flat sheet fibre having a typical cross-section ranging from 0.15 to 0.40 mm in thickness and 0.25 to 0.90 mm in width are produced by slitting (shearing) flat sheets. Deformed fibres which are loosely bonded with water soluble glue in the form of a bundle are also available. Since individual fibres tend to cluster together, their uniform distribution in the matrix is often difficult. This may be avoided by adding fibre bundles which separate during the mixing process.

10.1.2 Polypropylene Fibre-reinforced (PFR) Cement-mortar and Concrete

Polypropylene is one of the cheapest and abundantly available polymers. Polypropylene-fibres are resistant to most chemicals and it would be the cementitious matrix which would deteriorate first under aggressive chemical attack. Its melting point is high (about 165 °C), so that a working temperature as high as 100 °C may be sustained for short periods without detriment to the fibre properties.

Polypropylene short fibres in small volume fractions between 0.5 to 1.0% have been commercially used in concrete to achieve considerable improvement in impact strength of the hardened concrete. The have low modulus of elasticity. Polypropylene fibres are available in two forms: monofilaments produced from spinnerets, and film fibres produced by extrusions. The film fibres are commonly used and are obtained from fibrillated film twisted into twine and chopped, usually into 25 - 50 mm lengths for use in concrete. The fibrillated film may also be opened to produce continuous networks for use in thin sheet manufacture. Fibrillated film may also be woven to produce flat meshes which may be used as thin cement sheet reinforcement.

Polypropylene fibres being hydrophobic can be easily mixed as they do not need lengthy contact during mixing and only need to be evenly dispersed in the mix. These are therefore added shortly before the end of mixing the normal constituents. Prolonged mixing may lead to undesirable shredding of fibres. There is no physico-chemical bond between fibre and the matrix, only a mechanical bond is formed as cement paste penetrates the mesh structure between individual fabrics of chopped length or continuous network.

10.1.2.1 Properties of fresh PFR concrete

- The compacting factor test has been reported to be most suitable. The inclusion of polypropylene fibres reduces the workability considerably, e.g., a normal concrete mix of medium workability (C.F. about 1.14) may reduce to a low workability mix (C.F. about 1.33) following the addition of 1% of chopped 35 mm polypropylene fibres. Polypropylene monofilaments can be used in small volume fractions of about 0.1 to 0.2% to alter rheological properties of the material, e.g., highly air-entrained concretes can be stabilised by fibres.

10.1.2.2 Properties of hardened PFR concrete

- The tensile strength of concrete is essentially unaltered by the presence of a small volume of short polypropylene fibres. Although the change in flexural strength of

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polypropylene reinforced-concrete is marginal; the post-cracking behaviour has shown its ability to continue to absorb energy as fibres-pullout. The energy absorbing capacity has been found to increase with the length of fibres, e.g. the 75 mm polypropylene fibres may result in an energy absorption comparable to that of the less efficient of steel fibres, and at a considerably lower cost.

10.1.2.3 Durability of PFR concrete

- Polypropylene may deteriorate under attack from ultra violet radiation or by thermal oxidation process. The cement matrix appears to prevent the former. To combat thermal oxidation, sophisticated stabilisers have been developed to delay degradation and enhance durability.

10.1.2.4 Applications of PFR mortar and concrete

- Cladding panels:
  Inclusion of polypropylene fibres instead of steel mesh reinforcement may allow reduction in panel thickness.
- *Shotcreting*:
  Surface coatings of polypropylene reinforced-mortar may be provided by shotcreting using normal equipment. The fibres of about 20 mm length enable smooth Transport of the dry mix through air hoses and nozzles. Water is then added at the gun orifice. Shotcreting can be advantageously used in wet environments where polypropylene fibres can eliminate the need for steel (corrodable) mesh on which spray of mortar is required.
- Polypropylene concrete:
  That type of concrete can be advantageously used in the energy dissipating blocks.

The potential market for polypropylene reinforced-cement is, principally, as a substitute for asbestos-cement roofing and cladding panels.

10.1.3 Glass-fibre Reinforced Concrete (GFR)

Glass fibres are made up from 200 to 400 individual filaments which are lightly bonded to make up a strand. These strands can be chopped into various lengths or combined to make cloth, mat or tape. Using the conventional mixing technique for normal concrete, it is not possible to mix more than about 2% (by volume) of fibres of up to a length of 25 mm.

The major application of glass fibre has been in reinforcing the cement or mortar matrices used in production of thin-sheet products. The commonly used varieties of glass-fibres are E-glass used in the reinforcement of plastics, and AR-glass. E-glass have inadequate resistance to alkalis present in Portland cements whereas AR-glass have improved alkali-resistant characteristics. Sometimes polymers are also added in the mixes to improve some physical properties such as moisture movement.

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The process of manufacture of glass-fibre cement products may involve spraying, premixing or incorporation of continuous roving. In the spray-suction process, the glass-fibre strand is chopped into lengths between 10 to 50 mm and blown in spray simultaneously with the mortar slurry on to a mould or flat bed followed by suction to remove excess water. On the other hand in the technique involving premixing short strands (about 25 mm in length) are mixed into mortar paste or slurry before further processing by casting into open moulds, pumping into closed moulds, etc. Care must be taken to avoid fibre tangling and matting together, and to minimise the fibre damage during mixing.

In the process incorporating continuous roving, the rovings are impregnated with cement slurry by passing them through a cement bath before they are wound on to an appropriate mandrel. Additional slurry and chopped fibres on to the mandrel and compaction can be achieved by the application of roller pressure combined with suction.

10.1.3.1 Properties of hardened GFR concrete

The behaviour of glass-fibre cement sheets under tensile force is typified by multiple cracking of the matrix. Longer fibres improve the ultimate failure stress. In wet environments, significant reduction in strength takes place. The material may become brittle on ageing.

One of the most important improvements in the property achieved by glass fibre is the spectacular improvement in impact strength. With the addition of for example 5% glass fibres, an improvement in the impact strength of up to 1500% can be registered as compared to plain concrete. With a 2% fibre content (up to 25 mm in length), the flexural strength is almost doubled. The second important improvement is in the resistance to thermal shock. Ductility also improves with an increase in strength and modulus of rupture.

The flexural strength of water stored and weathered specimens reduces with time and nearly equals that of the matrix alone. The reduction in energy absorption is similar to that in flexural strength. The **long-term durability** of glass fibre-reinforced cement can be improved by the addition of 15% polymer to the mortar matrix. The increase in matrix cost is balanced by the use of cheaper E-glass fibres.

10.1.3.2 Applications of GFR concrete

The glass fibre-reinforced cement finds its use in formwork systems, ducting, roofing elements, sewer lining, swimming pools, fire-stop partitioning, tanks and drainage elements, etc. Sometimes it is used in combination with polymer impregnated in situ concrete.

10.1.4 Asbestos Fibres

The naturally available inexpensive mineral fibre, asbestos, has been successfully combined with Portland cement paste to form a widely used product called asbestos cement. Asbestos fibres have thermal, mechanical and chemical resistance making them suitable for sheet products, pipes, tiles and corrugated roofing elements. Asbestos-cement products contain about 8 to 16% (by volume) of asbestos-fibres. The flexural strength of asbestos cement board is approximately two to four times that of unreinforced matrix. However, due to relatively short length (10 mm), the fibres have low impact strength. There are health hazards associated with the use of asbestos cement. In the near future, it is likely that cellulose, PP, PVA or glass fibre-reinforce concrete will replace asbestos completely.

10.1.5 Carbon Fibres

Carbon-fibres form the most recent and probably the most spectacular addition to the range of fibres available for commercial use. Carbon fibres come under the high E-type fibres. These are expensive. Their strength and stiffness characteristics have been found to be

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superior even to those of steel. But they are more vulnerable to damage than even glass fibres, and hence are generally treated with resin coating.

10.1.6 Organic Fibres

Organic fibres, such as polypropylene or natural fibres may be chemically more inert than either steel or glass fibres. They are also cheaper, especially if natural. The polypropylene-fibre concrete has been described earlier. A large volume of vegetable fibres (7%, 50 mm length) may be used to obtain a multiple cracking composite. The problem of mixing and uniform dispersion may be solved by adding a superplasticizer.

10.1.7 Vegetable Fibres, and General Remarks

The commonly used fibres are jute, coir and bamboo. They possess good tensile strength in their natural dry state. Their tensile strengths do not suffer significantly even after being immersed in 10% solution of sodium hydroxide for up to 28 days. However, long-term durability is doubtful.

In contrast to glass fibres, steel, asbestos and polypropylene fibres are chemically stable in a cement paste matrix. The high alkalinity of cement paste protects steel from being corroded. The corrosion of steel fibres can however become a problem when the matrix has cracked.

Irrespective of the type, size and shape of fibres to be used in a mix, the fundamental requirement of fibre-reinforced concrete is that all the individual fibres should be uniformly distributed throughout the matrix. The mix should have sufficient paste content to coat the fibres and aggregate, so that the ingredients can be placed and compacted in the final position without any segregation.

The mix proportions generally depend on the intended applications of the composite. The prime considerations are uniform dispersion of fibres, adequate workability for placing and compaction with the available equipment. The workability of fibre-reinforced concrete is influenced by maximum size of aggregate, volume fraction, geometry and aspect ratio of fibres.

As the size of aggregate increases, it becomes more difficult to achieve uniform fibre dispersion, since the fibres are bunched into mortar fraction which can move freely past the aggregate during compaction.

To obtain a better dispersion the coarse aggregate content is kept lower than a normal mix and the maximum size of aggregate is preferably limited to 10 mm. The mortar matrix (consisting of particles less than 4 mm) should be around 70%, and aggregate/cement ratio as low as 3 : 1. A fine-to-coarse aggregate ratio of 1 : 1 is often a good starting point for a mix trial. Water/cement ratio between 0.4 and 0.6, cement-content of 250 to 450 kg/m^3 are recommended for providing adequate paste content to coat large surface of fibres. Beyond a certain optimum content of fibres the workability of the composite decreases rapidly.

11. OTHER SPECIAL CONCRETES

11.1 Heavy-weight Concrete

Concrete with high density is required for instance for ballasting, for precast concrete used for construction in water sound proofing, and above all for radiation-shields of reactors, laboratories, protective rooms, etc. As absorption of radiation depends upon the concrete mass, the walls can be thinner when the concrete is heavier. Bulk densities from 2750 up to 5000 kg/m^3 can be reached.

Heavy concrete, of course, has to be as compact as possible, but high densities can only be obtained by using heavy aggregates. They are selected according to the purpose of their

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application and, in the case of radiation shields, also on the kind of radiation. Natural as well as artificial aggregates can be used; slag cements and high-alumina cements are preferred in some cases.

The following table gives an overview of fields of application for heavy concrete (medicine, research, technique industry):

Aggregates for preparation of radiation stable concrete

Source of radiation	Type of radiation	Applicable types of aggregate
Radio nuclides	Gamma-rays	**Aggregates with high density:** Baryte, slag from heavy metals, Ilmenite, Ferro-silicon, Magnetite, Ferro-phosphorous, Hematite, granulated steel
X-ray apparatus	X-rays	**Aggregates with high atomic number:** Baryte, slag from heavy metals
Cyclotron	Electron-rays	**Aggregates with crystal water:** Ilmenite, Serpentine
	Gamma-rays	**Aggregates with high absorption capacity:** Colemanite, Boron-ferrite, Bor-calcite, Bor-carbide
	Neutrons	
Atom reactor	Gamma-rays Neutrons	,
	Neutrons	

11.2 Heat Resistant or Refractory Concrete

Heat resistant or refractory concrete is usually meant by concrete consisting of a high-alumina cement and a special refractory aggregate. Properly chosen cement and aggregate will enable the concrete to withstand temperatures up to 18000 °C. Such concretes are also used for cement kiln linings.

11.3 White and Coloured Concrete

White and coloured concrete for architectural purposes can be produced with white cement, „warm-tone" cement or a cement to which a colouring material has been added. Similar effects can be obtained by adding a pigment when mixing the concrete (maximum 10%, mostly 3 to 5%, of cement weight). Usually mineral pigments will be used (iron oxides, chrome oxides, cobalt spinels) but carbon black is possible, too..

11.4 Sand-Lime Concrete (Lime siliceous blocks)

Sand-lime concrete - as the name indicates is made of sand and lime in an autoclave process.

During the hydrothermal curing under pressure in the autoclave, the quartz of the sand reacts with lime forming calcium silicate hydrate, quite similar to those formed by hydration of Portland cement. As lime contains more calcium than cement, a lower quantity of lime is sufficient to obtain a similar effect.

The calcium silicate hydrate is the strength bearing phase:

- **Hydrothermal reaction:**

$$CaO + SiO_2 + H_2O \rightarrow CaO \cdot SiO_2 \cdot H_2O$$

The raw materials are quartz sand (>60% SiO_2) and quick lime (> 80% active CaO); quick lime can be substituted by lime hydrate. The sand can be used unground or partly or entirely ground. Grinding can be done separately or together with the lime. To regulate the activity of

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the lime, retarders such as gypsum are added. The amount of water required for a definite consistency depends mainly upon the specific surface of the quick lime. The mix is immediately moulded and compacted. After a so-called aging time, steam-curing is performed in autoclaves at a pressure of 12 to 16 bars during 2 to 6 hours. Sand and lime quality as well as compacting and curing are responsible for the strength of the final product.

Sand-lime is mostly used in the form of bricks as wall-forming materials replacing clay or concrete bricks. Large precast elements are fabricated too; in this case, more attention is paid to the composition and the preparation of the raw mix.

The bulk weight of sand-lime bricks is 1500 to 2100 kg/m^3 the compressive strength is from 15 up to 60 MPa

11.5 Polymers in Concrete

11.5.1 General

Polymers (plastics) possessing special properties, which differ from those of inorganic materials, are also used in concrete but only for special applications.

11.5.1.1 Advantages of polymers are:

- Good chemical resistance
- Easily malleable and workable
- Air and water tight
- Low density

The physical properties can be varied in a wide range by the composition and admixtures. Due to certain unfavourable characteristics, however, the application, especially in buildings, is restricted.

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11.5.1.2 Drawbacks of polymers:

- High deformation under load (creep)
- Low heat resistance
- High price

Plastics as compared to cement of the same weight are about 100 times more expensive or 30 times more by volume.

In the following, the application as a binding material in concrete and mortar is treated. Whereas admixtures are used in quantities of a tenth per mil (0.1‰) up to a few percents of the cement weight, 5 to 30 % of the cement weight are added in this case, so that polymers form a significant part of the concrete matrix.

11.5.2 Polymer Cement Concrete (PCC)

Polymer Cement Concrete is manufactured by adding a monomer emulsion during the mixing of ordinary fresh concrete. The polymerisation of this monomer is initiated after mixing.

The monomer of a plastic material is the basic molecule which is in a liquid state. By polymerisation, some thousands of molecules are bonded together in a chain or network, thus forming a solid plastic-material.

- The main **advantages of PCC** are:
  - Improved workability
  - Improved impact strength
  - Improved flexural strength
  - Reduced shrinkage
    (as a result of a lower water/cement ratio)
  - Lower Young-modulus
  - Higher Deformation limit (up to five times)
  - Better toughness
  - Less danger of cracking
  - Better bond to old concrete (e.g. for repairs)
- The **disadvantages** are:
  - High material cost
    (about five times that of ordinary concrete)
  - Worse creep characteristics
  - Long-time performance not yet completely determined

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11.5.3 Polymer-Impregnated Concrete (PIC)

Polymer Impregnated Concrete is formed by impregnating the air voids in a normally hardened concrete with a monomer, and subsequently polymerising by heating or exposure to radiation. The polymer content of PIC depends on the volume of air voids in the original concrete and lies between 3 and 7% by volume. The following characteristics are claimed to have been improved as compared to the original concrete.

- Improved properties of polymer-impregnated concrete:
  - Compressive strength increases: 3 to 4 times
  - Tensile strength increases: 4 times
  - Young's modulus increases: 2 times
  - Water permeability and water absorption becomes negligible
  - Higher resistance to frost/thaw cycles
  - Higher resistance to chemical attacks

12. MORTAR

12.1 Overview

12.1.1 Definition of mortar:

- Mortar is a mixture of:
  - Cementitious material
  - Fine aggregate (sand)
  - Water
  - Admixtures (sometimes),

12.1.2 Application of mortar

- The most important types of mortar are:
  - Masonry mortar
  - Plastering mortar for exterior of structure
  - Plastering mortar for interior of structure

12.2 Cementitious materials for mortar preparation

- Portland cement
- Portland blast-furnace slag cement
- Portland pozzolan cement
- Masonry cement
- Quicklime
- Hydraulic lime
- Hydrated lime
- Gypsum

The combination of different cementitious materials is very often used in mortar practice.

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12.3 Aggregates for mortar preparation

Natural or crushed sand is used as aggregate in mortar. The maximum grain size of aggregate varies with the type of mortar; generally it lies between 1 to 4 mm. The aggregate surface should be free of injurious amounts of organic impurities and friable particles.

12.4 Admixtures

The following admixtures are used for mortar preparation:

- Water-reducers
- Plasticizers
- Grouting admixtures
- Bonding admixtures
- Colouring admixtures

Just as for concrete, there are specified requirements for the storage, measurement and mixing of mortars.

In the following table are summarised typical mortar mixes used in Switzerland

Mortar mixes and their properties (plastic consistency)

Type of mortar	Cementitious material, kg/m^3		Sand (dry), kg/m^3	Compressive strength, MPa	
	OPC	Hydraulic lime		at 7 days	at 28 days
Cement mortar	300 to 450	---	1500 to 1620	10 to 20	14 to 30
Combined lime-cement mortar	100 to 150	200 to 300	1440 to 1620	1.8 to 3.2	3.0 to 5.5
Hydraulic lime mortar	---	300 to 400	1490 to 1570	0.4 to 0.7	0.8 to 1.4

12.5 Packaged, Dry Combined Materials for Mortar (Dry Mortar)

As the name indicates, „Packaged, dry combined materials for mortar" are combinations of cementing materials and aggregates which require only mixing water to be ready for use. The packages - in sacks or bulk (for order of large quantities) - are generally available at a production plant or building material supply stores. Several types of mortars, especially for rendering, are on the market. The plastics are sometimes added to the cementitious materials to improve the bond strength between structure surface and rendering.

12.6 Ready-Mixed Mortar

Ready-mixed mortar is a product of the ready-mix concrete industry. The development was started in Europe around 1975 - to compensate for the loss of input in the ready-mix concrete business.

Ready-mixed mortar is a trowel-ready and immediately applicable wet mortar; it is produced in the ready-mix concrete plant and transported to the building site in trucks commonly used in ready-mix business.

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Ready-mixed mortar is a cement mortar with chemical admixtures. The components are usually:

- Portland cement
- Sand (fraction 0 to 2 mm)
- Water
- Admixtures: air-entraining agent and retarder, sometimes a stabiliser is added; the admixtures have following functions:
  - **Air-entraining agent:** to obtain the necessary workability and trowel flow through introduction of 20 - 25 % of air bubbles into mortar mix
  - **Retarder:** to keep the mortar workable during approx. 30 to 40 hours (up to 2 working days)

The mortar is discharged on the job site into containers with known capacity (0.25, 0.33, 1.0 m^3) and keeps its workability during the 30 - 40 hours. After the ready-mixed mortar is used (as laying mortar for brick-work and block-work), the normal hydration begins.

In some countries (Germany, Belgium) the production of ready-mixed mortar represented about 8 - 15% of the total of all the ready-mix concrete companies.

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Chapter 1

Quarrying

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Quarrying

R.Playle
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1. INTRODUCTION

In the cement industry quarrying is the mining method for the production of raw materials in the process of making cement. Quarrying describes the surface mining of rock whereas 'open pit' mining describes the surface mining of minerals. Discontinuous mining systems such as scraping, ripping and dozing and blasting are commonly practiced in limestone quarries. Continuous mechanized mining systems, with bucket wheel or chain excavators, are used where the deposits are consistent and soft such as chalk.

Quarrying is the breaking of the rock in a safe and economic way and transporting the result to a plant for further reduction in size. Figure 1 graphically sketches the sequence of events required to bring quarry into operation.

This involves planning, purchasing of suitable equipment, drilling, blasting, loading and transport of rock in quantities sufficient to permit the continuous operation of the cement plant. All this must be accomplished as efficiently and safely as possible to maximize the return on investment. Finally environmental considerations must be remembered and careful control kept on noise and dust levels. Parts of the quarry that become worked out must be rehabilitated so that there are few scars on the landscape.

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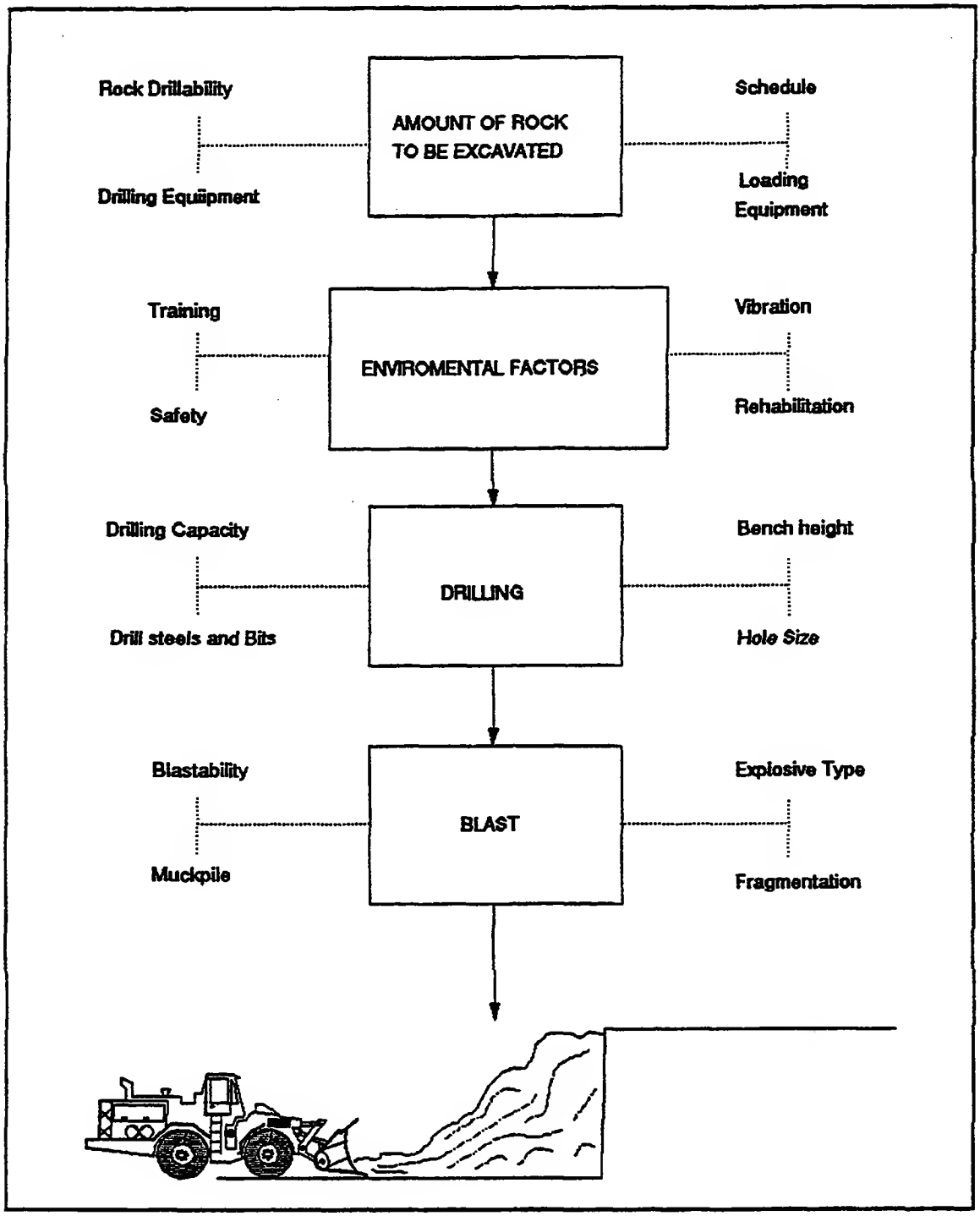
Figure 1: Quarry Operation



Blasting is the most widely used method to excavate limestone for cement production as the rock is usually too hard to be ripped or dozed. This involves the drilling of holes in the rock, placing a predetermined charge of explosive in each hole and detonating it. The result is a pile of broken rock which then has to be removed to the cement plant. As it is not economic to break each piece of rock to the required dimensions in the blasting process the rock is further reduce in size before being transported to the cement plant storage silo. This further reduction is accomplished by means of a crusher system.

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To obtain optimum results from the quarry operation the rock to be blasted has to be matched to the explosive and the drilling parameters. Resulting fragmentation and muckpile shape, broken rock from the blast, is dependent on all three factors.

2. GEOLOGY AND GEOGRAPHY

There are many rock types existing in the earth's crust and all are due to geological processes that started about 4,5 billion years ago. By contrast man has only been in existence for a few thousand years.

2.1 Rock Hardness

Rock hardness is classified by using 'Mohs scale of hardness'. This lists ten rocks with differing degrees of hardness in an ascending order. The rocks used as standards, their Mohs' hardness and basic classification are to be found in table 1.

Limestone, with a high percentage of calcium carbonate, is one of the sedimentary rocks found in the crust of the earth. It has an average Mohs's hardness of about 3.3. This means that it is usually necessary to blast limestone to free it from the surrounding rock.

Table 1 Moh's scale of hardness

Rock	Hardness	UCS (MPa)	
Talc	1	-10	soft
Gypsum	2	10	
Calcite	3	30	
Fluorite	4		medium
Apatite	5		
Feldspar	6	120	
Quartz	7		hard
Topaz	8	200+	
Corundum	9		very hard
Diamond	10		

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2.2 Influence of Rock Characteristics

There are a number of theories relating to the effect that geology and in particular rock hardness has on the breaking effect of explosives. In most cases results are based on the assumption that the ground is homogeneous and contains no jointing bedding or planes. However these theories can be used as a guide as to the hole burden and spacing necessary to give the required fragmentation when using a particular explosive. They remove some of the guesswork and trial and error from the planning. To determine these factors it is necessary to obtain certain rock properties. This can be partly accomplished in the laboratory and partly by examination of core samples. The properties required are:

Density	- specific weight
Young's Modulus	- stress to strain ratio
Poisson's Ratio	- measure of elasticity
Uniaxial Compressive Strength	- static load necessary to break a sample of rock

Results from these tests can give a guide as to the best explosive and the correct drill hole diameter to provide the most efficient drilling and blasting at the cheapest cost.

2.3 Influence on the Blasting Process

Geology, in the system of jointing and bedding planes, will dictate the bench geometry and face orientation as well as the resulting fragmentation. Examples of the influence of dip can be seen in figure 2 which shows quarry faces with rock formations both dipping into and out of the face.

Jointing has the same effect on drilling and blasting as schistosity and bedding planes. The most significant property of a joint is its inability to transmit tensile stress. Its tensile strength can be considered as zero, or minute in comparison with solid rock. Stress waves are reflected from joint walls and therefore interrupted in their travel through the rock. The visible result of this is large, blocky boulders in the muckpile.

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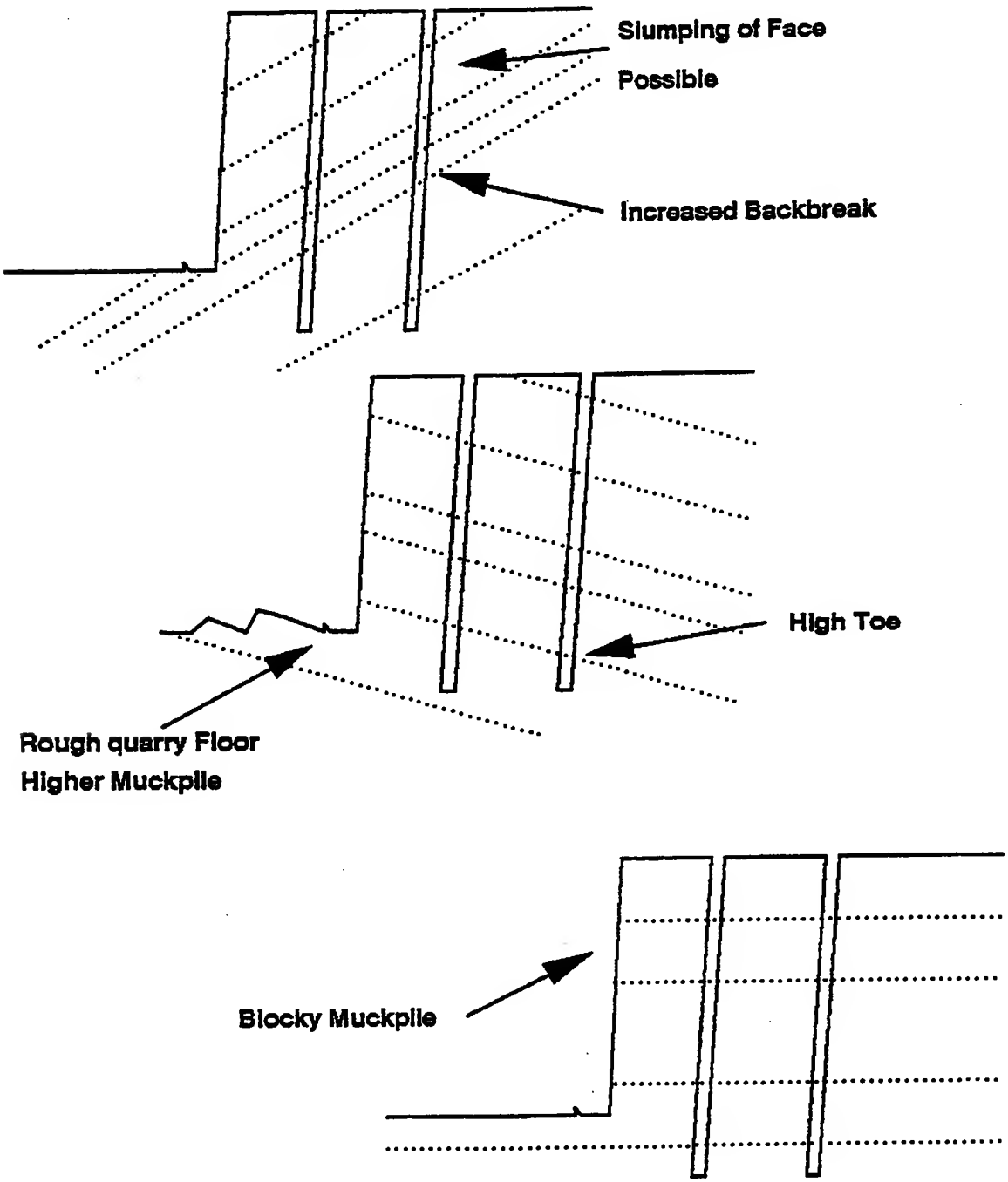
Figure 2 Jointing Effect on Breaking



2.4 Topography

Geography will be considered when planning a new quarry. The topography of the country will determine the development of the quarry, pit like operations will result in flat terrain. Contour or side hill operations would be found in hilly areas. Accessibility to the area, roads, services, and buildings will also determine the development and infrastructure required at the quarry site. Additional problems are created when the proposed quarry is close to a built up area. Complaints can be expected by local inhabitants, from blasting and the noise made by machinery. The proposed area of a quarry may be limited by its proximity to an area of natural beauty. Environmental damage to the area and the building of access roads have to be minimized if the quarry is in a tourist area.

3. DRILLING

Bench drilling is a term for designating the method where surface holes are drilled for blasting towards a free face. Blasted rock then has ample space in which to expand. Bench heights vary between 5,0 meters and 30,0 meters, depending on the thickness of the formation, however the limiting factor in the height of the bench is usually that of safety or hole diameter. Safety can be the inherent stability of the formation, the greater the stability the higher the bench and hence the longer the drill hole. An increase in the stability of the formation can be achieved by the use of inclined drill holes. This creates an artificial slope at an angle of between seventy and eighty degrees. If a failure of the bench occurs the rock will tend to roll down the incline face and not drop thus confining loose rock.

Larger hole diameters mean that longer holes can be accurately drilled. Toe position of a hole is important if successful breaking is to occur. With hole diameters of 64 mm or less a hole drilled over 20 meters is likely to wander from the line. Requirements of a drillhole are drawn in figure 3.

Quarry operations usually use an intermediate diameter drillhole of between 64 mm and 165 mm although very large quarries can have hole diameters of up to 440 mm.

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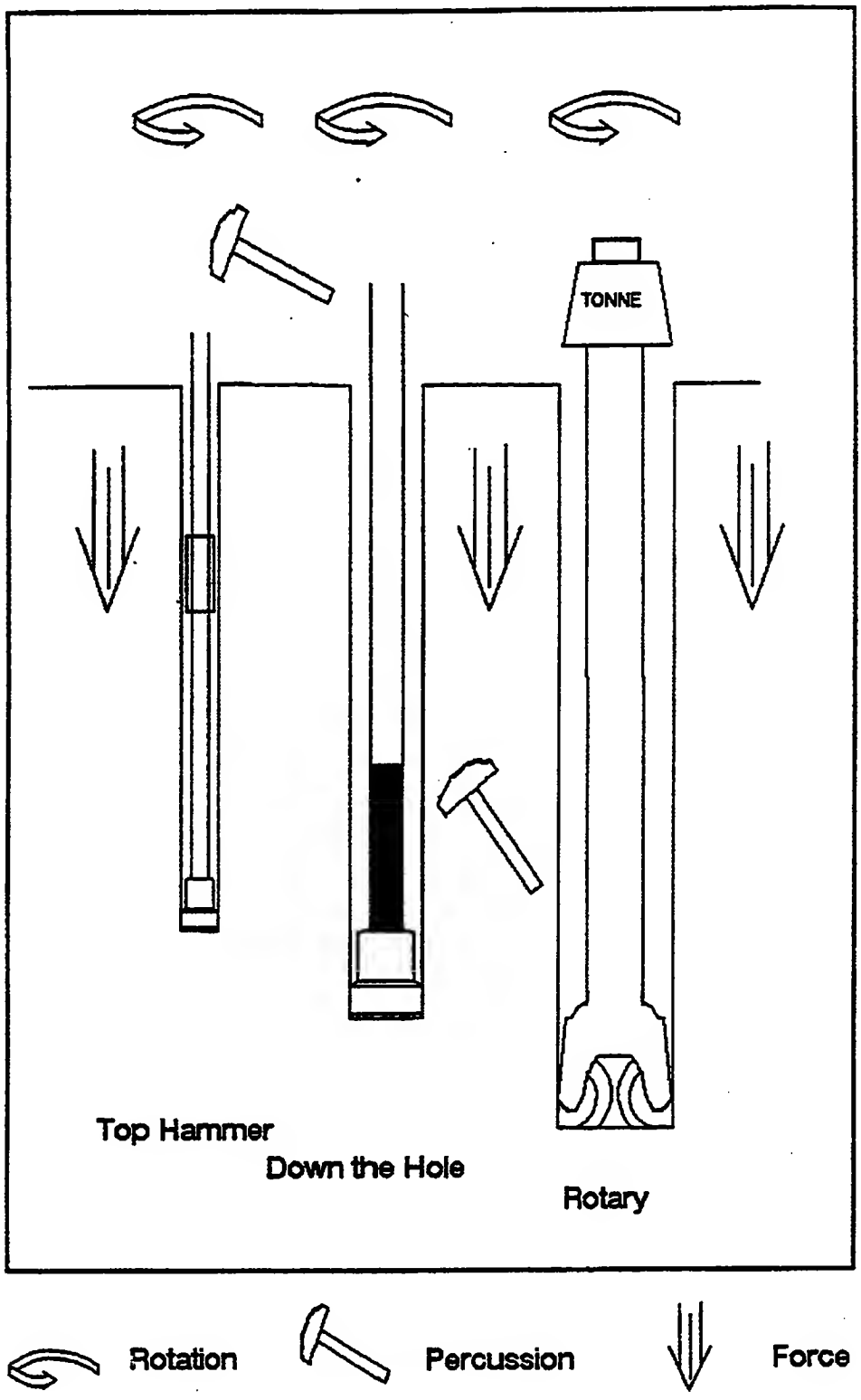
Figure 3 Drilling Methods



3.1 Drilling Methods

3.1.1 Rotary Drilling

Most commonly used in soft-medium rock masses that have an average Mohs' hardness of less than about four. This is a drilling method that was originally used for oil well holes. In rotary drilling energy is transmitted via a drill rod which is rotated at the same time as the drill bit is forced down. The bit is rotated continuously at between 50 and 90 revolutions per minute. Downward thrust is achieved by the weight of the machine. This force is used to push the roller inserts, the cutting edges on the bit, into the rock which on rotation break off small chips of the rock. The relationship between feed force and rotation rate determines drilling efficiency. Energy losses in rods are minimal in rotary drilling. All such drilling requires high feed pressure and slow rotation but the relationship varies with rock type. Softer rock requires lower pressure and higher rotation speed and vice versa.

3.1.2 Percussion Drilling

Practiced where the rock is hard to very hard and abrasive, divided into:

3.1.2.1 Top Hammer Drilling

A shank adapter is hit repeatedly by a piston in the drilling machine which creates a shock wave that travels down the drill string to the bit. Energy is discharged against the bottom of the hole and the surface of the rock is crushed. Air traveling down the center of the drill string flushes the resulting chips out of the hole. The whole drill string is rotated and crushes each segment of the hole face in turn. Drilling machine and string are arranged on a feed, usually either chain or screw, and the feed force can be kept in contact with the hole bottom by adding extra drill rods when the feed mechanism reaches the limit of its travel.

3.1.2.2 Down the Hole Hammer

The hammer and its impact mechanism operate down the hole. The piston striking directly on to the bit. No energy is therefore lost through transmitting the shock wave down a drill string. The drill tubes carry the compressed air for hammer and flushing.

3.1.2.3 Overburden Drilling

A new quarry operation is fortunate if the rock to be blasted is fully exposed on surface. In the majority of cases it is overlain with soil and weathered, powdery rock called collectively overburden. This must first be removed before the rock below can be removed. In cases where overburden is soft dozers and rippers can accomplish the work.

Conventional methods, as described above, are not suited to these ground conditions and problems could be experienced with stuck rods. A special drill that has casing tubes with its own ring bit, in addition to the normal bit, are manufactured for these conditions. This specially designed drilling rig has the following attributes:

1) Drilling will be continuous through varying ground conditions.
2) If stopped drilling can continue in loose ground easily.
3) The hole is prevented from collapsing when the bit is removed.
4) Flushing is efficient and rotation torque is high.

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3.2 Comparison of Drill System

Penetration rate of a down the hole drill is, in theory, independent of hole length and therefore will be capable of drilling longer holes with less deflection than the surface hammer. As the rotation mechanism is down the hole these drills perform better in fractured ground than the surface drills. The rods of the latter may jam when rock fragments fall down the hole.

Down the hole hammers, because of their mechanism, are limited in the size of hole they can drill, the minimum being about 92 mm. For smaller diameter a surface percussion drill will be used. Rotary drilling is the major method of blast hole drilling in large limestone quarries, where hole diameters of up to 150 mm are found. Hydraulic drilling has advantages over pneumatic in that the transmission of forces and energy is accomplished by the use of a fluid. Fluids are virtually incompressible and only a fraction of the energy is lost in transmission. Together with a high rotational speed this permits a higher drilling rate.

The machines used for rock drilling can be roughly grouped into the following based on their working principles:

3.2.1 Top Hammer

- Hydraulic rock drills
- Pneumatic rock drills

3.2.2 Down the Hole

- Pneumatic hammer and pneumatic rotation unit
- Pneumatic hammer and hydraulic rotation unit

3.2.3 Rotary

- Small and large hole rotary cutting
- Large hole rotary crushing

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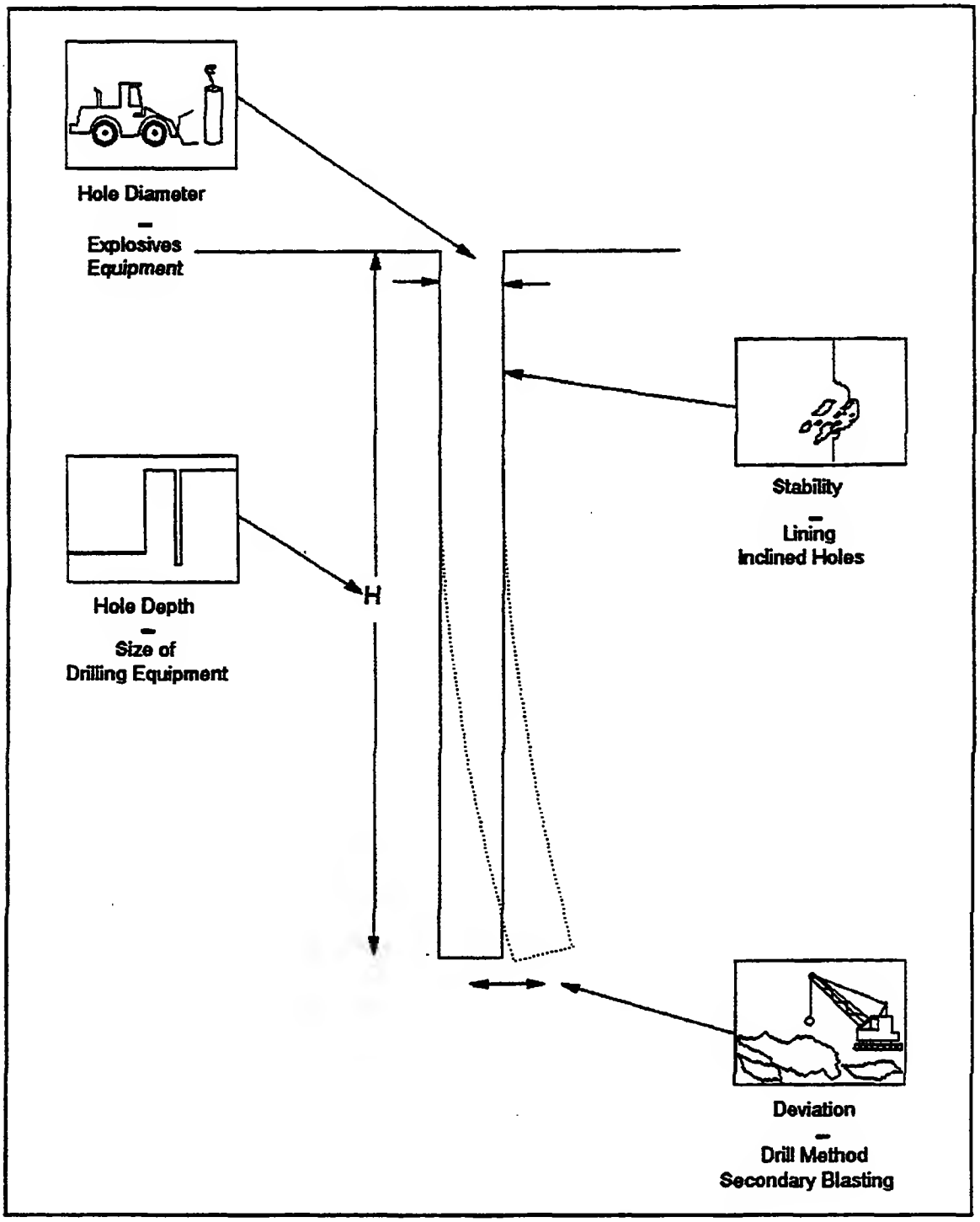
3.3 Drilling Parameters

Determining the size and number of blastholes as well as the drill rig required to drill them is based on many different parameters. Some of these demands are shown in figure 4.

Figure 4 Demands on Drilling



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Other demands include:

3.3.1 Capital

Cost outlay required to purchase rigs.

3.3.2 Nature of the Rock

Jointing, bedding planes, a smaller hole diameter will permit better distribution of the explosive in the rock and a better explosive efficiency. A larger hole diameter means a larger burden and more concentration of the explosive. Although drilling costs would be lower, fewer joint bound blocks would be intersected by drill holes resulting in excessive secondary blasting costs.

3.3.3 Fragmentation Size Distribution

Efficient drilling designs, combined with the correct choice of explosive result in better fragmentation. Correct collaring of holes improves the whole quarry efficiency. If holes are drilled out of position or of the incorrect length then fragmentation is going to be poorer. Too long holes will result in wander of the toe of the hole with resulting increases in burden and spacing again resulting in poor fragmentation and overall explosive performance.

3.3.4 Monthly tonnage requirements

Larger hole diameters are used where there is a greater tonnage requirement.

3.3.5 Cap Rock

Problems with oversize from the collar area. Smaller diameter holes require a shorter stemming length and higher explosive column in the hole.

3.3.6 Vibration and Airblast

Smaller hole diameters imply a smaller mass of explosive per delay reducing the level of ground vibrations.

3.4 Drilling Definitions and Equations

The relationship of the various parameters used in drilling is shown in figure 5. There are no 'hard and fast' rules. They only serve as a guideline and are based on experience.

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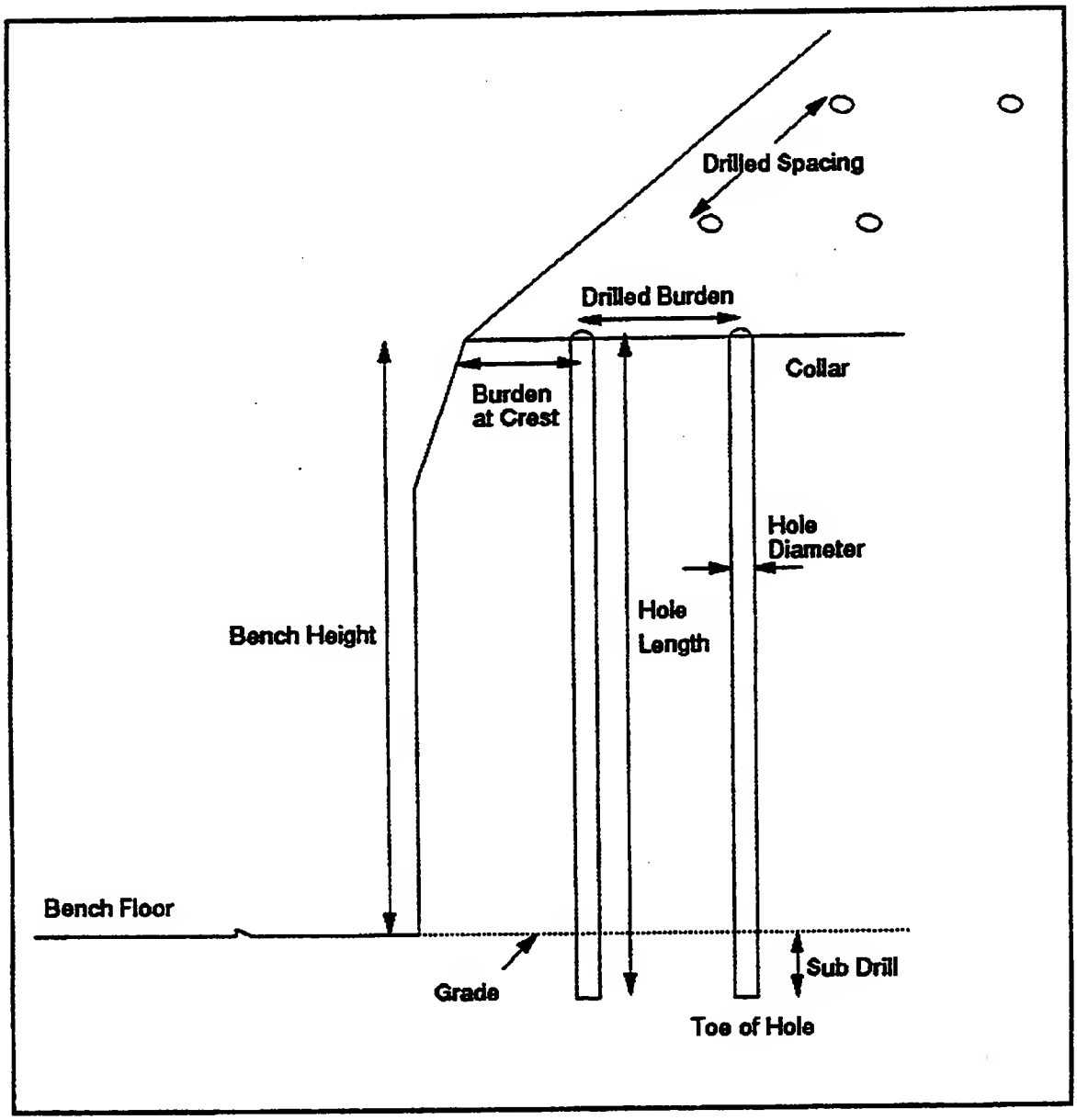
Figure 5 Drilling Technology



3.4.1 Borehole Diameter

This is important to obtain maximum fragmentation at the lowest cost. The cost of drilling, per cubic meter blasted, decreases as the hole diameter increases. However too large a diameter can cause problems with resulting airblast and flyrock if the bench height is too low. Excessive jointing at small intervals in the rock can cause a fragmentation problem if a large hole diameter is selected. For best fragmentation control the appropriate bench height to borehole diameter is 0.12 to 1 (in meters and mm). That is:

0.12 * D = H

where D = borehole diameter in mm

H = bench height in meters

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3.4.2 Burden

This is considered the most critical variable in the design of surface blasts. It is defined as the distance from a borehole to the nearest free face at the time of detonation. In planning, the burden is taken as the distance at right angles to the free face, from the free face to the first row of blastholes. Burden is a function of the charge diameter and therefore also depends on the drillhole diameter.

B = (25 to 35) * E

where B = burden in meters

E = explosive in hole, diameter in meters

3.4.3 Spacing

Is the distance between adjacent boreholes and is measured parallel to the free face. The optimum spacing to burden ratio is between 1 and 1.3. Too small a ratio leads to holes not breaking out, with resulting flyrock and ground vibration problems. Too large a ratio has the effect of increasing oversize as the blastholes break individually to the free face.

3.4.4 Bench Height

Distance, measured vertically from one level floor of a quarry to the next up or down. For a successful design it is important that the burden and bench height are reasonably compatible. The minimum is:

H = 2 * B

where H = bench height in meters

B = Burden in meters

If the bench height is low and the burden and blasthole is large then an excessive percentage of the borehole is taken up with stemming. Very high benches become a danger for personnel working beneath them.

3.4.5 Subdrill

The distance drilled below the level of the bench floor necessary to be certain that, on blasting, the toe of the holes breaks out.

3.4.6 Vertical and Inclined Holes

Drill holes are drilled either at an angle to the vertical in the direction of the free face or vertical. There are several advantages to incline drilling as detailed in figure 6. In addition there can be a reduced explosive cost due to increased burden. There is also less risk of backbreak and toe problems. However inclined drilling needs closer supervision in order to achieve good results. If the hole is underburdened then flyrock is more likely from blowouts.

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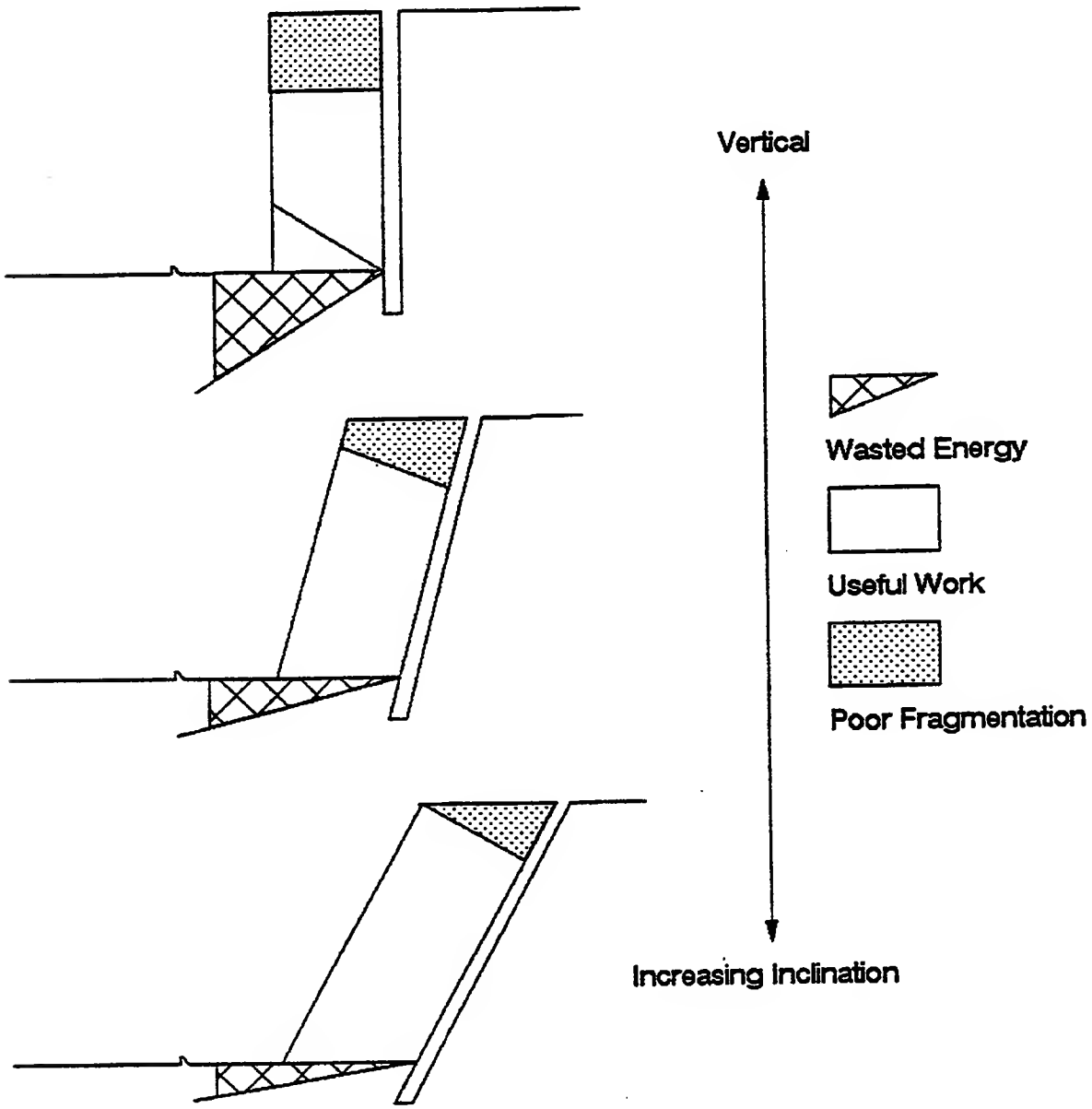
Figure 6 Effect on Inclining Holes



3.4.7 Drill Hole Deviation or Wander

The drill bit and drill string can be deflected from its planned straight course after collaring. The major causes are:

3.4.8 Collaring

The start of the hole where the drill bit is placed on a marked position and drilling commences.

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3.5 Drilling Patterns

Blast holes are drilled to a pre-planned pattern which will determine:

- number of holes and drilled meters
- cubic meters of rock to be blasted

Drilling plans are either made using prepared drilling tables for certain parameters or they can be made especially with a desired result in mind.

The most common pattern are:

3.5.1 Square

Equal burden and spacing

3.5.2 Rectangular

Burden is less than the spacing.
Easily marked out and easy to collar accurately.
Figure 7 shows these patterns.

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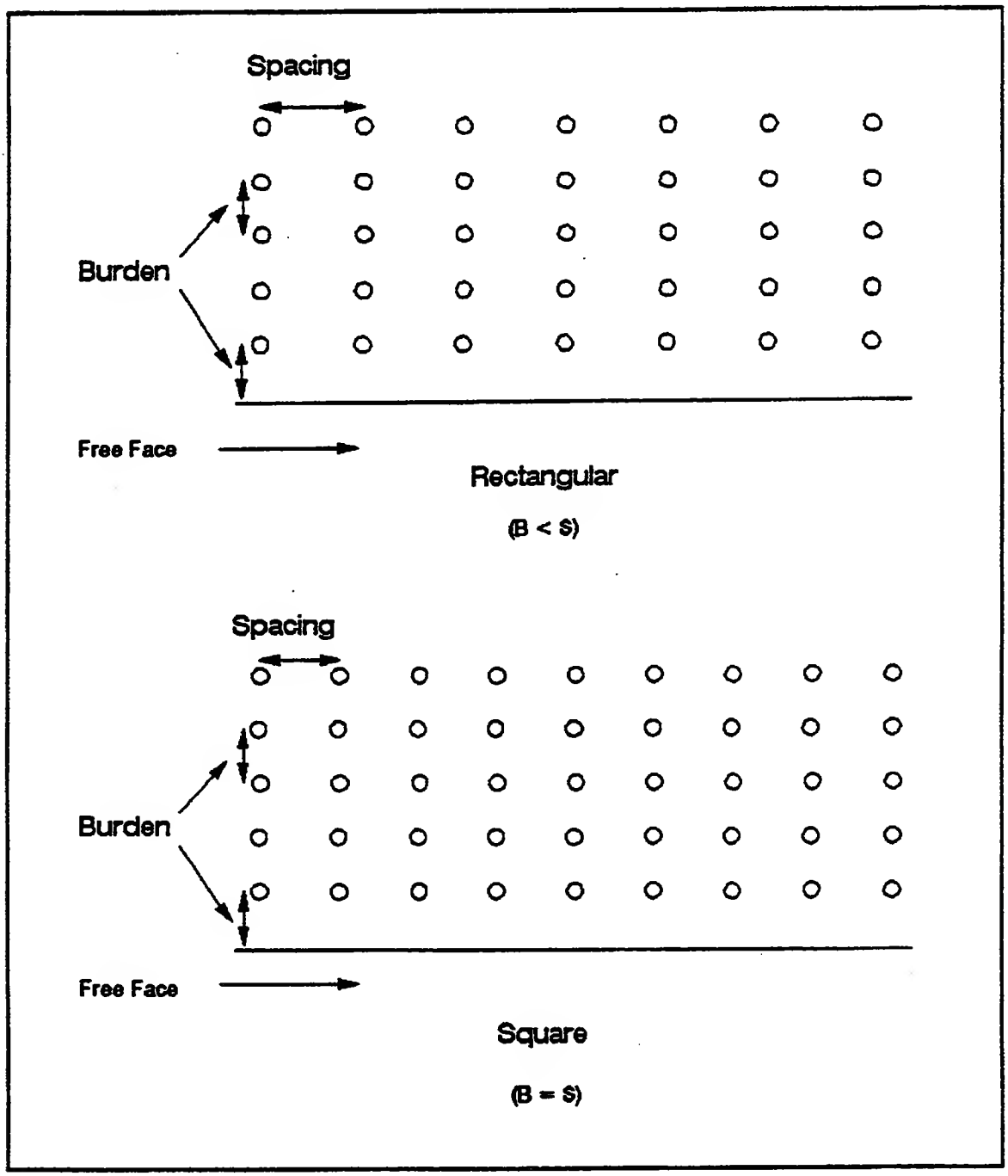
Figure 7 Inline Drilling Patterns



3.5.3 Staggered

Spacing to burden ratio may be one is more usually greater than one. The holes in alternative rows are in the middle of the spacings of the row in front. Figure 8 shows an example of each pattern.

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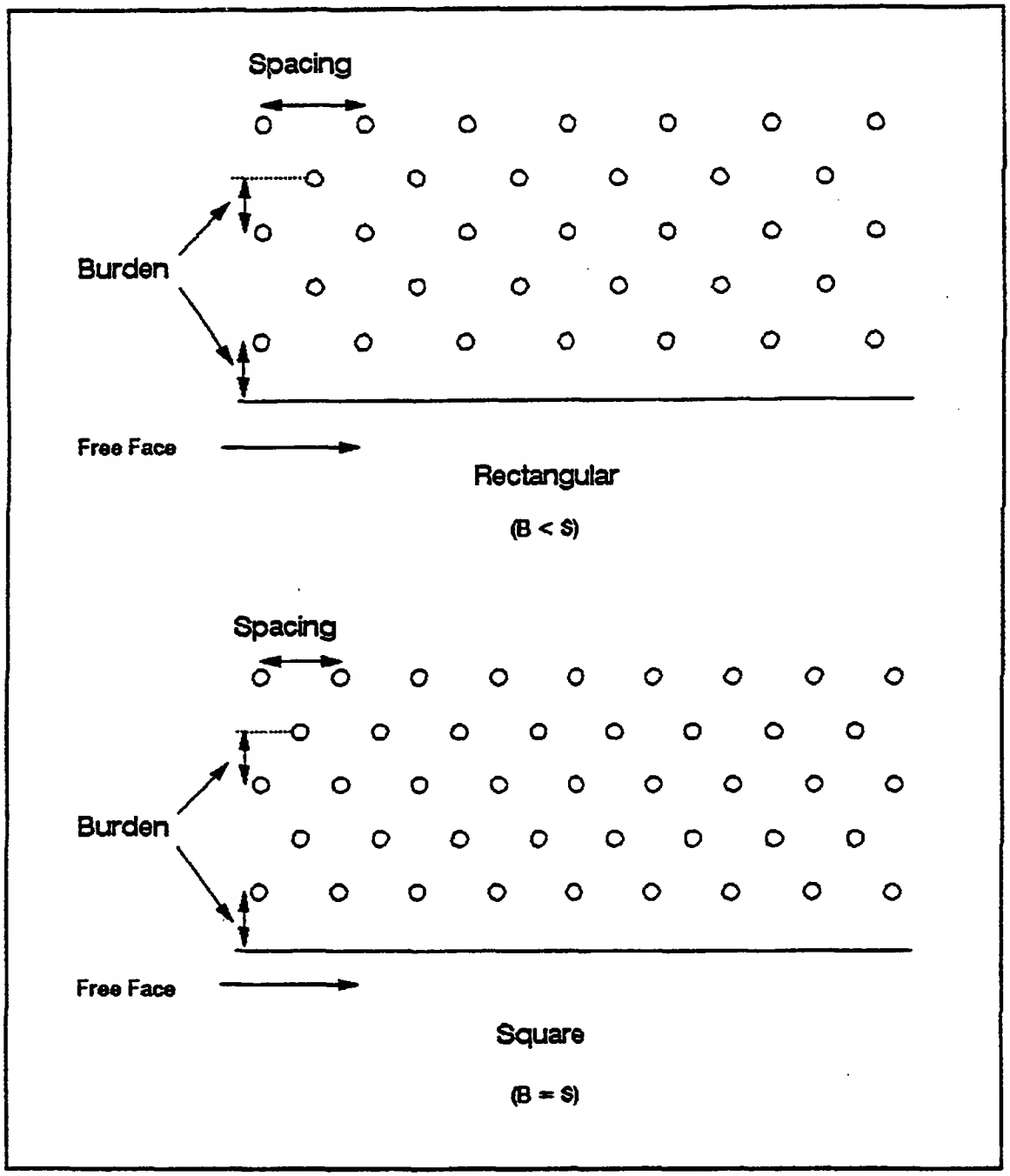
Figure 8 Drilling Patterns for Staggered Holes



Ideally holes should be drilled in a triangular pattern where the spacing to burden ratio is 1,15. This equilateral triangular pattern provides for the optimum distribution of the explosive charge, for any particular hole diameter, throughout the rock. In this way the cost of drilling and blasting can be optimized. The area around the hole influenced by the detonation of the explosive and the distribution of the resulting explosive energy throughout the surrounding rock is graphically shown in figure 9 and compared to other patterns.

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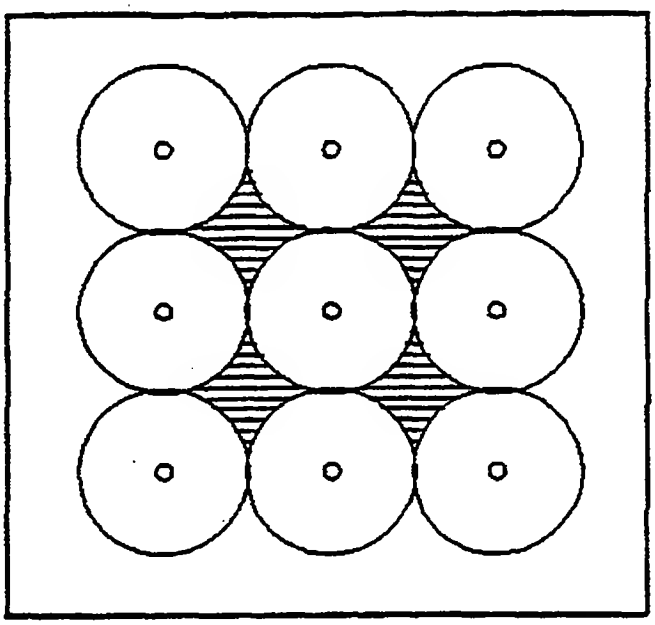
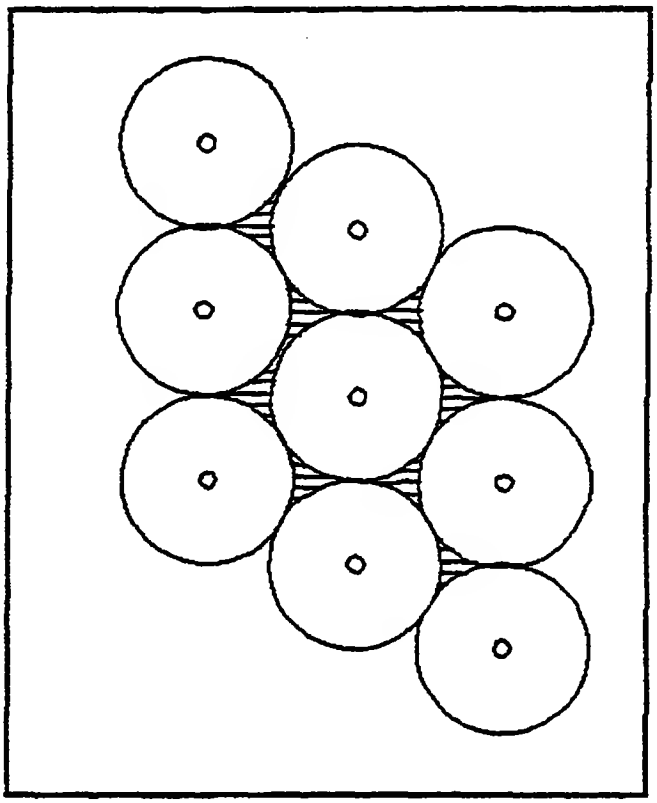
Figure 9 Comparison of areas of Explosive Influence around a Blasthole

Inline Pattern



Staggered Pattern



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4. BLASTING

Blasting is an intermediate step in the process of quarrying. Choosing the correct explosives is one of the most important decisions influencing the design and operation of a quarry. It cannot be taken in isolation as other factors, such as rock type and hole diameter, also influence the choice. The major objectives of a blast are to suitably fragment the rock and to displace it such that it is easy and safe to load out. Once a blast has been initiated it is uncontrollable and cannot be repeated if incorrect. The greater the effort put into the planning and preparation the better the results will be.

4.1 History of Explosives

Commercial explosives started with the invention of black powder. The first recorded mention of 'saltpeter' is in the 13th century in Arabia. Its first recorded use in mining was in the Royal Mines in Hungary and from here its use spread to the tin mines of Cornwall in England in the late 17th century.

Ascanio Sobero discovered Nitroglycerin (NG) in 1846 and it was used in its raw state in blast holes with black powder igniters. This proved to be very hazardous and Alfred Nobel, while seeking a safer way to transport this new explosive chanced on Kieselguhr which rendered nitroglycerin less sensitive to shock, this was the first dynamite. Dynamite was not only much more powerful than black powder but with its higher velocity of detonation (VOD) it was more effective in breaking rock. The only real drawback was that it was not waterproof. In 1875 Nobel found that by dissolving nitrocellulose in nitroglycerin he could increase the water resistance of his explosive, which was called 'blasting gelatin'. The next step in the making of NG explosives was to replace part of the expensive nitroglycerin with low cost Ammonium Nitrate. It has only about 70 % of the blasting strength of NG and was highly hygroscopic but research to increase the percentage used in Dynamites continued. This resulted in a relatively cheap high performance explosive with good waterproof characteristics still in use today.

Concurrently better and safer ways to initiate the explosive cartridges were being discovered. In 1831 a Cornishamn invented Safety Fuse, a continuous core of black powder wrapped in jute and twine and coated with varnish to make it waterproof. However, this fuse would not initiate Nobel's explosives and he solved the problem with the patenting of his Fulminate of Mercurycaps. This also showed that to maximize the energy produced from a detonating explosive a shock wave was required. With this combination a certain reliability in timing and detonation was introduced at the initiation of explosives. Bridge wires and electric blasting were patented in the late 1800's and together with detonator delays produced superior fragmentation.

Devastating explosions in two ships in 1948, when fuel oil leaked into their cargoes of Ammonium Nitrate (AN), led to the discovery of the cheapest of all explosives. If AN is mixed with about 6 % Fuel Oil (FO) it becomes a powerful relatively safe explosive known as ANFO. AN is now manufactured in small prills to give the explosive good flow characteristics.

The increase in blasthole diameter in the 1950s led to new explosives coming into existence. These were the Watergels consisting of AN, water, a thickener and a sensitizer. Research had been conducted to waterproof AN products and it was found that the way to increase resistance to water was to shield it chemically. The main advantages were, a high loading density, good performance and low sensitivity. Above all they did not contain any headache causing ingredients, a big problem with all NG based explosives.

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Emulsion explosives, developed in the 1960s, are a high performance explosive which detonates without a sensitizer being added. They are prepared in the form of oil in water emulsions. The basic ingredients are AN, water and fuel, the water in oil. A wide variety in water immiscible fluids permits a wide range of products, stiff to fluid. They are extremely stable over a wide temperature range.

Explosives today are defined either as 'poured', 'pumped' or 'cartriged'. Bulk loading of holes, from a mobile container often mixed to form an explosive on site, is safer and cheaper than loading cartridges into a hole. However where an exact charge per hole is necessary or small quantities required then packaged or poured explosive is preferred.

4.2 Breaking Process

An explosive is a chemical compound or mixture ignited by either shock, impact or friction. When ignited it decomposes rapidly as a detonation. There is a release of heat and large quantities of high pressure gases, which expand rapidly with sufficient force to overcome confining forces such as the rock around the drill hole. In commercial blasting the energy released by the detonation manifests itself in four ways, these are:

- rock fragmentation
- rock displacement
- ground vibration
- airblast

4.3 Explosive Properties

Each explosive has certain different characteristics or properties. Some of the major properties are listed below:

4.3.1 Velocity of Detonation

The speed at which a detonation wave travels through a column of explosives.

4.3.2 Density

Specific gravity, the standard being water.

4.3.3 Detonation Pressure

The pressure immediately behind the detonation front.

4.3.4 Energy

A measure of the potential of an explosive to do work.

4.3.5 Strength

The ability of an explosive to work.

4.3.6 Sensitivity

A measure of the minimum energy required to initiate the explosive.

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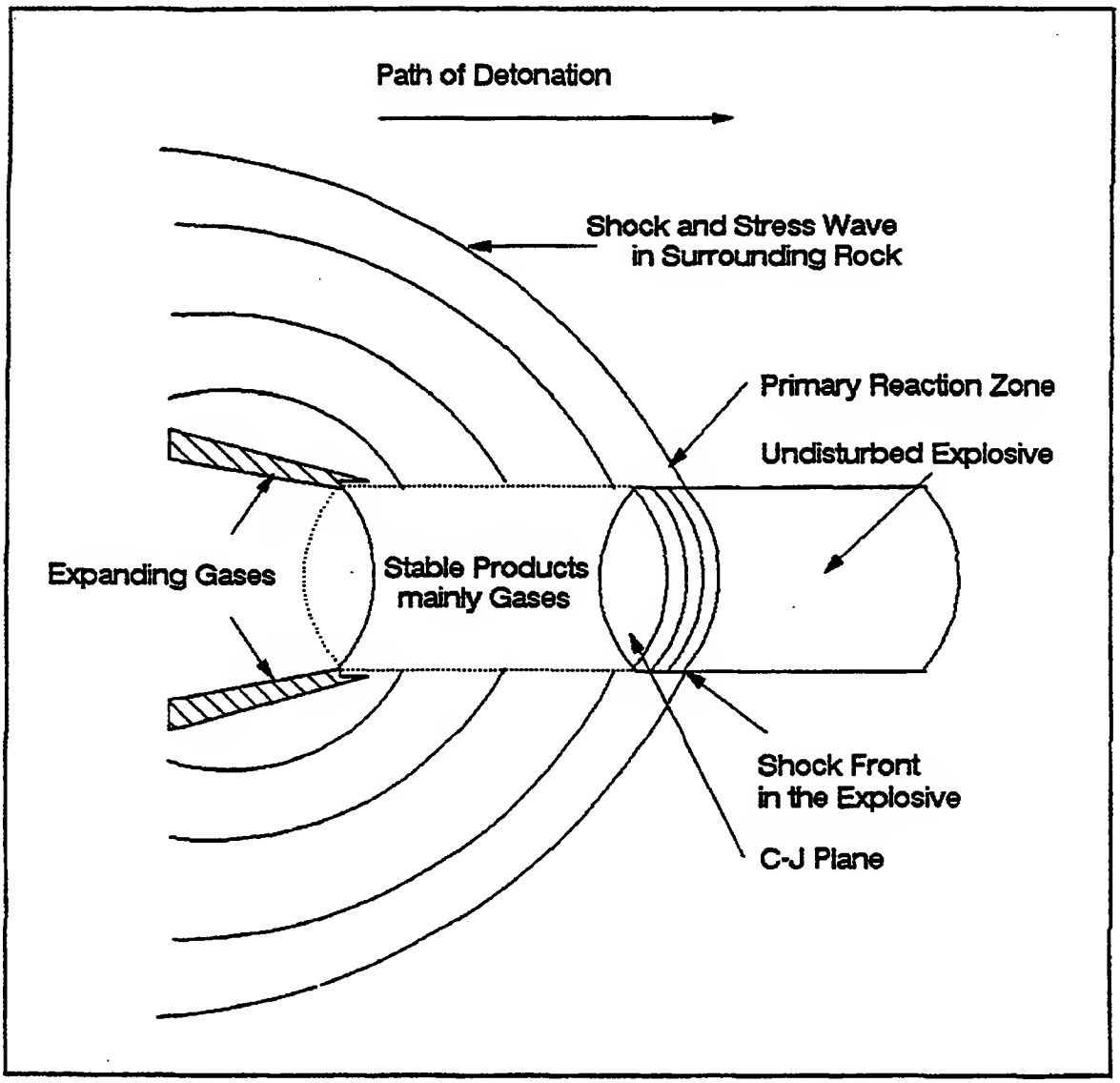
4.4 The Process of Detonation

In a detonation the chemical reaction moves through the explosive material at a velocity greater than that of the speed of sound through the same material. The definitive characteristic if this chemical reaction is that it is initiated by, and supports, a supersonic shockwave proceeding through the explosive. Deflagration of an explosive occurs when the shockwave moves too slowly to produce significant shock energy.

Generally explosives with a lower VOD tend to release gas pressure over a longer period than those of higher VOD. These, lower VOD explosives, have more heave, important in areas where material movement is needed. Figure 10 shows a schematic drawing of a typical detonation.

Figure 10 Sketch of a Typical Detonation



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In commercial explosives significant chemical reactions occur behind the primary reaction zone and effect the explosive performance. This is due in part to the need to make explosives safe to handle. The majority of products are gases at high temperature and pressure. In the order of 4000 degrees centigrade and 20 to 100 kilobars. These gases expand rapidly and produce the shock wave in the surrounding medium which in turn is transmitted into the rock around the borehole. Shock provides the energy for fragmenting the rock and gas, the heave energy to move the fragmented blocks to form the muckpile.

An explosion in a drilled hole is closely followed by the shock wave passing through the rock and stressing it, first in compression and then in tension. Tensile forces cause small radial cracks to develop from the hole which are subsequently expanded by the explosive gases entering them.

The free rock surface starts to move forward unloading the pressure. The tension increases in the primary cracks which expand to surface and complete the loosening of the rock.

4.5 Efficiency of Explosives

The efficiency of all explosives depends on several factors:

4.5.1 Coupling Ratio

The cross-sectional area of the drillhole filled with explosive. With pumped or pour loaded explosives this will be 100 %. Cartridges, not being able to fill the hole completely will always have a lower ratio.

4.5.2 Diameter

The larger the diameter of the borehole, which is the primary influence on the diameter of the explosive used, the greater the VOD of that charge. Figure 11 shows how the VOD of the common explosives is affected by the diameter.

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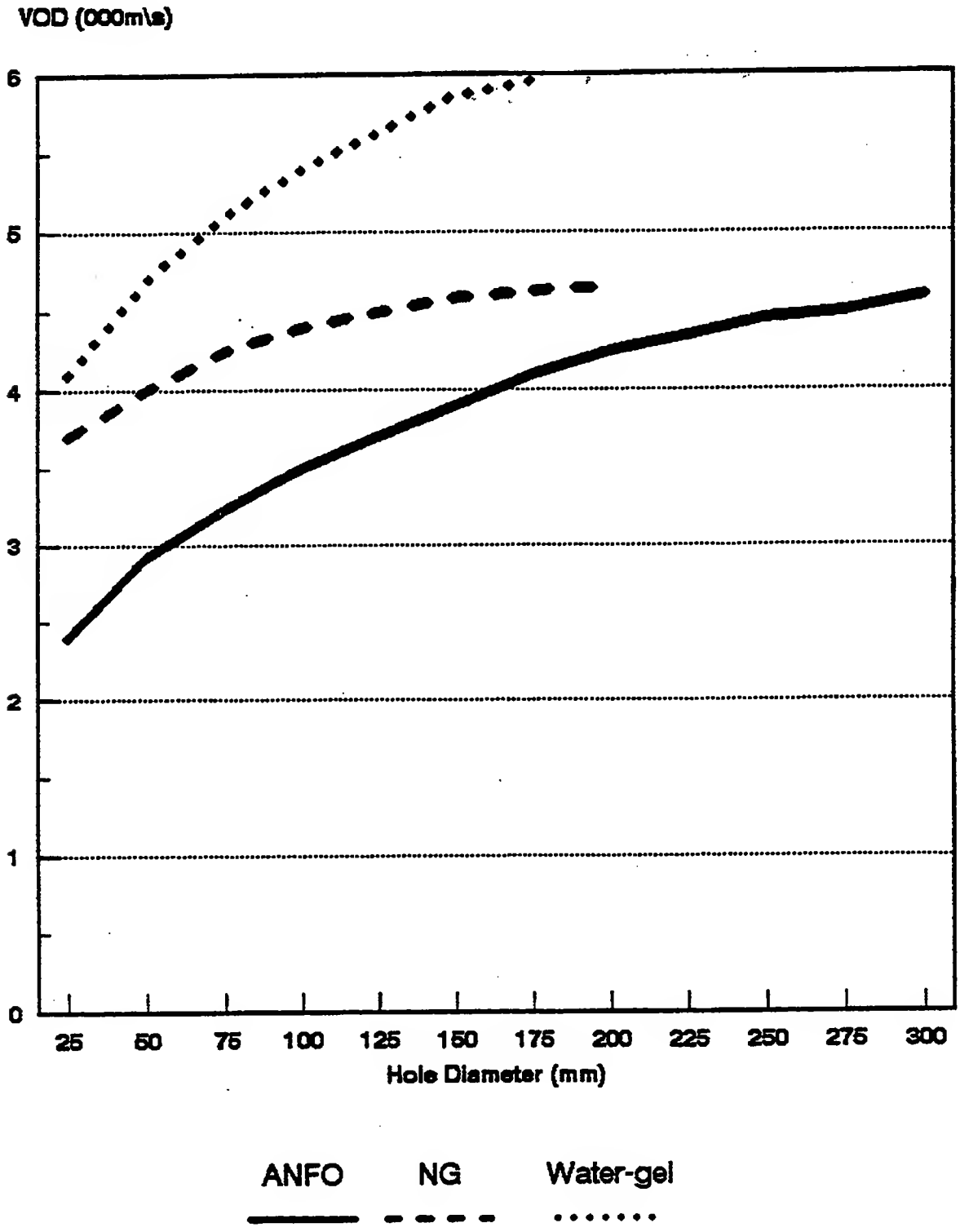
Figure 11 Comparison of VOD with Hole Diameter



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4.5.3 Priming

Adequate priming insures that the explosives will reach its maximum VOD as soon as possible. Inadequate priming can mean that the explosive efficiency is not fully exploited. A primer is an explosive, either a high VOD explosive or made from PETN, that accepts initiation from a detonator or detonating cord. It transmits this detonation to a mass of explosive contained in a drillhole. The cross section of the primer should be such as to match the hole diameter as closely as possible and as long as necessary to attain its maximum VOD. This will result in the detonation of the column of explosive at its highest potential to do work.

The primer should always be at the point of greatest confinement, if possible, at the toe of the hole. The exception is when horizontal band of harder rock is encountered in a series of drill holes. To break this effectively the primer can be located at this point in the hole.

4.5.4 Stemming

It is the non explosive material between the top of the explosive column and the collar of the hole. Stemming can consist of drill fines or gravel. Drill chippings are favored as they are readily available and their cost is minimal. The stemming height in a hole is dependent on the nature of the rock and amount of flyrock and noise that can be tolerated.

The use of effective stemming increases the amount of useful work that the explosive performs in the hole. This has a beneficial effect on the cost of the blasting operation.

4.6 Explosive Selection

Field and economic considerations influence the type of explosives needed to produce the desired results. Low cost, good fragmentation and adequate heave are the desired results. On the basis of cost no explosive can compete with bulk ANFO in dry holes. It accounts for about 80 % of all explosives used in surface blasting. Emulsion and ANFO emulsion mixes possess an increased efficiency value due to the fact that emulsion is a highly efficient explosive. Site conditions such as water, heave required and drilling cost can make the use of a water resistant, higher energy explosive, more attractive. The use of cartridged explosive in surface mining is unusual as the increased handling and storage costs required to place it into the hole makes it expensive. It has an application where a small, exact charge is needed in order to limit vibration or flyrock. Table 2 summarizes and compares the properties of the three most common explosive types.

Table 2 Comparison of properties

EXPLOSIVE	ANFO	WATERGEL	NG Based
Detonator Sensitive	No	No	Yes
Density	0,80	1,30	1,40
Relative Energy	1,00	1,82	1.66
Gas Development	34	39	38
Waterproof	No	Yes	Some-Yes
Bulk Loading	Yes	Yes	No

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4.7 Powder Factor

A mathematical relationship between the weight of explosives and a quantity of rock expressed in kilograms, or grams, per ton broken. It is based on an assumption that explosive weight and explosive energy are one and the same. Different explosives have different energy outputs and cannot be compared to each other on this basis. To overcome this the term 'Energy Factor' has been introduced. In general the powder factor is related to the income producing unit of the operation.

4.8 Energy Factor

This is defined as the amount of explosive energy, in kilojoules, in a given quantity of rock.

Energy Factor = kJ/tons

5. INITIATION SYSTEMS

An initiation system is a combination of explosives and accessories designed to convey a signal to a column of explosives to detonate at a particular instant. The signal function may be either electric or non electric. Common systems by which explosives are detonated are:

- Initiation by an electrical impulse
- Detonating cord
- Safety fuse and plain detonator
- Initiation by a shock wave (Nonel)

The most widely used system is the electric detonator and associated circuitry.

Electric and Nonel detonators are made in three different designs.

1) **IED** - instantaneous electric detonator. This has no delay element and is used, for example, to initiate detonating cord at the start of a blast.
2) **MS** - IT is the same basic construction as the IED but includes a delay element. The delay interval is measured in milliseconds (MS). Generally used for bench blasting in a quarry.
3) **LP** - The delay element is of a different composition which permits a long delay (LP) measured in seconds or parts of seconds.

5.1 Electric Detonators

An electric system uses electric power with an associated circuit to convey an impulse to electric detonators. These will fire and initiate an explosive charge. Inside the detonator the electrical energy is converted to heat by passing the firing current through a high resistance bridgewire. The heat energy ignites a pyrotechnic surrounding the bridgewire. The resulting flash ignites the delay element which then burns for a designated time and in turn ignites a base charge in the toe of the detonator, this initiates the primer. Figure 12 shows a section of an electric detonator. The electric blasting circuit consists of three elements:

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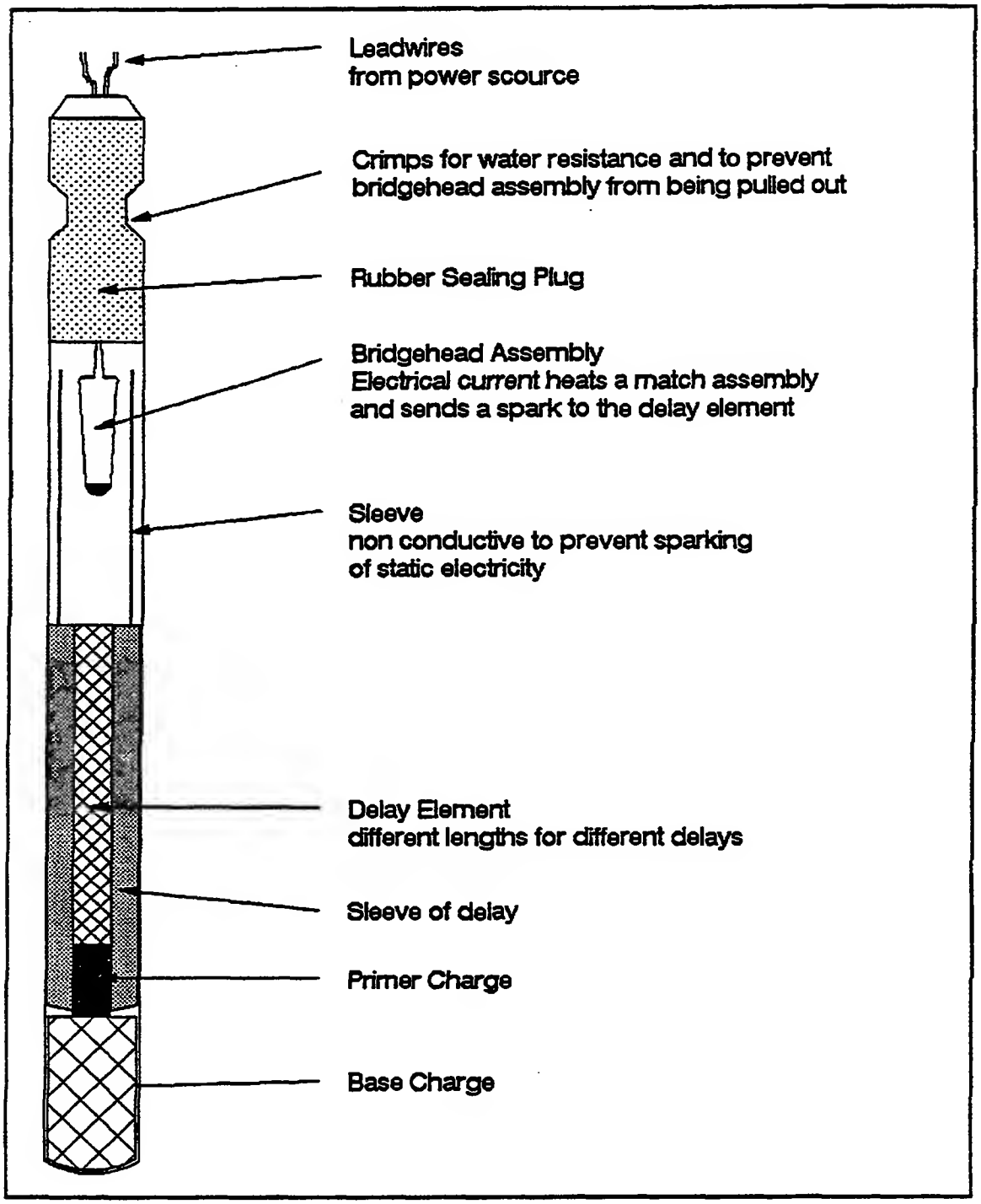
Figure 12: Section of Detonator



5.1.1 Detonators

With lead wires connected in series or parallel or a combination of each.

5.1.2 Circuit Wiring

Connects the detonator circuit to the power source. Two wires of a resistance of about 1.5 ohm per 100 meters known as the blasting cable.

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5.1.3 Power Source

Commonly called a shot exploder, to provide the electrical energy to the firing circuit.

5.2 Detonating Cord

This is a flexible tube containing a center core of high velocity explosive, usually PETN (pentaerythritol tetranitrate) that is used to, either, detonate explosives or transmit a detonation wave from cord to cord. The core of explosive is covered with combinations of materials to protect it from misuse or water and to give it strength. It is used in conjunction with relays to provide short firing delay intervals. Certain detonating cords can be used down large diameter holes as their tensile connected to a lower strength cord on surface which contain the relays. Detonating cord comes in reels of about 200 meters and can be cut to any length. A simple knot is enough to ensure propagation of the shock across a join.

5.3 Blasting Cap

Initiated by a safety fuse, it comprises an aluminum tube loaded with two charges. A base charge of a high explosive, PETN and a primer charge of lead azide. The primer charge changes the burning of the safety fuse into a detonation and initiates the base charge. It is used with igniter cord, which initiates the fuse, as an inaccurate delay system to detonate multiple small diameter hole blasts.

5.4 Nonel

In order to reduce the problems associated with electric delay systems, stray currents and earth leakage being two of the main ones, a system of non-electric (nonel) was invented. It uses a thin transparent plastic tube of 3 millimeters in diameter to transmit a low energy signal to a detonator at 2000 m/s. The tube contains a thin coating of reactive material on the inside which in itself will not detonate explosives. The signal is initiated by either a detonator or detonating cord. Various lengths of tube with a detonator are sold as a unit.

5.5 Delay Blasting

Millisecond delay blasting was introduced to the quarries many years ago. When blasting the rock movement time is very important, particularly in multiple row blasts. With one row of holes movement generated by blasting is directly away from the face, in an almost horizontal direction. As the number of rows increases so will the rock movement tend towards the vertical (flyrock). This is caused by the low velocity of the broken impeding the movement of that behind. The time requirement between rows to permit the rock to move in multi row blasts is 8 to 10 milliseconds per meter of drilled burden. For multiple row blasting the optimum delay is within the period that results in good fragmentation without the presence of cutoffs. Where the number of rows exceeds five or six an increase in the delay time is needed to successfully break the back rows without causing flyrock.

The true burden is dependent on both the drill and delay pattern selected. Initiation sequence affects the principal direction of rock movement. The best fragmentation is achieved when each charge is given enough time to detach the rock surrounding it before the next charge detonates. An additional free face is available to the next charge and the residual stresses in the rock are high enough to assist in the breaking by the subsequent charge.

The simplest type of delay blasting is to fire a single row of holes with delays between the holes. This system is applied to large diameter drillholes used in many quarries and results in better fragmentation than instantaneous blasts, where no delays are used.

Blasting laws often limit the mass of explosive that can be detonated at one instant in time. This is particularly so where ground vibrations must be kept to a minimum. For example the proximity of quarries to built up areas.

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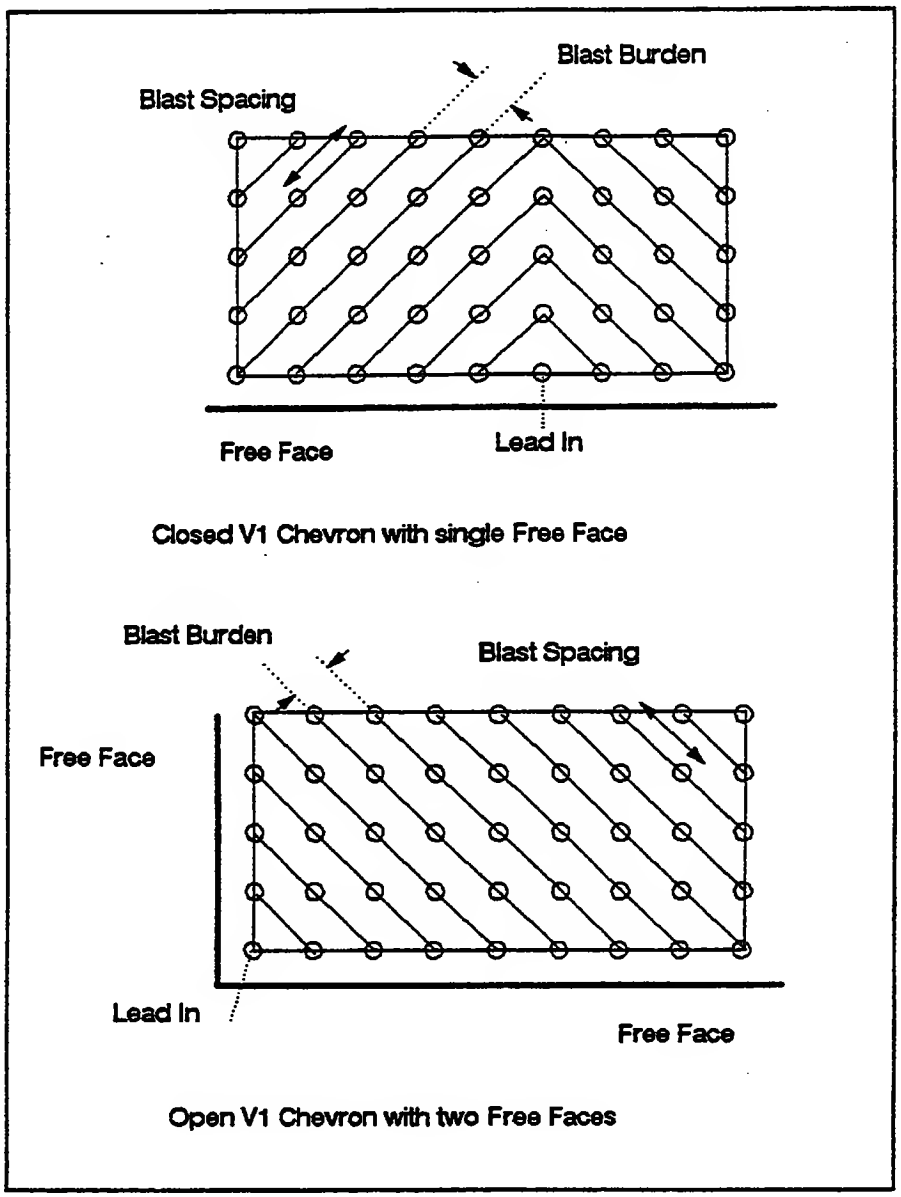
5.6 Blasting Patterns

The length of most primary blasts, in relation to the number of rows, is such that initiation of a row will mean that a large number of holes will be detonated simultaneously. A 'V' pattern reduces this number as well as reducing the blasted burden of the holes. Change in firing direction will reduce the mass of explosive detonating at any one instant and therefore the vibration level. The type of pattern used will also regulate the position and height of the muckpile.

Figure 13 shows a 'V1' connection which can be either an open or closed chevron.

Figure 13: Timing Patterns



Blast Spacing to Burden ratio < 4 to 1

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6. EFFECTS OF BLASTING

6.1 Fragmentation

The extent to which rock is broken into pieces by blasting.

Degree of fragmentation desired is dependent on the loading and crushing equipment and use of the product. In the economics of blasting cheaper means coarser fragmentation but this requires larger loading equipment to handle the oversize. Bigger machinery is designed for bigger tonnages not to load out oversize. Every quarry manager should consider the increased cost of drilling and blasting against the profitability of an increase in output obtained by an increase in fragmentation in order to obtain the lowest cost per unit for the whole operation. Many factors affect fragmentation. The major ones are discussed.

6.1.1 Terminology

'Fragmentation' is a general and highly subjective term in which 'good' does not necessarily mean 'small pieces'. Depending on the chief requirement of the operation and the final product management will focus on one of the following:

- Fines
- Oversize
- Mean Fragment Size

It is these terms that define fragmentation.

6.1.1.1 Fines

Rock particles of suitable chemical quality but too small for processing or sale. These represent an effective loss of reserve and production. Fines incur costs in production and additional costs in storage or disposal if not used. In a limestone quarry the fines can either be represented as increased capacity to the plant, they need no further reduction in size by the crusher, or as the overuse of explosive energy negating the use of the crusher.

6.1.1.2 Oversize

Rock particles too large to be handled by the available loading or crushing equipment. This causes delays in production while being moved out of the way and increased maintenance costs to loading equipment not designed to handle them. They generally increase working costs, the majority have to be re-broken, and reduce the safety of both workers and equipment. It is normal practice for management to measure the blast purely on the percentage oversize contained in the muckpile and to disregard the percentage fines.

6.1.1.3 Mean Size

Defined as the mesh size through which 50 % of the muckpile can pass. While not as obviously critical as the above it provides a meaningful guide to ease of digging. Being at the midpoint of the size range 'mean size' is not as sensitive to small variations in the 'oversize'. For example a 100 % increase in boulder count may represent less than a 1 % increase in mean size.

6.1.2 Quality of Explosives

The product of strength per unit weight and charging density, known as strength per unit volume, determines the effectiveness of different explosives in breaking rock.

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6.1.3 Rock Characteristics

In jointed rocks the correct direction of blasting is important. In hard, solid and slightly fractured rock the extent of the crushed zone around the blasthole is dependent on the charge per hole length.

6.1.4 Blasthole Loading

Proper fragmentation occurs when there is enough force in the compression wave to travel to the free face and back. The quantity of explosives is enough to fragment the rock. If not enough stemming is placed in the collar of the blastholes then, on detonation, gasses will escape from the holes reducing the efficiency of the explosives.

6.1.5 Drilling Accuracy

If drilling accuracy is suspect the planned drilling parameters will have to be decreased to maintain explosive efficiency.

6.1.6 Timing and Pattern

Better breaking has been found when using a delay interval, in milliseconds, of about ten times the drilled burden in meters. The interval should be chosen after taking into account the burden and spacing and the number of holes to be blasted.

The use of 'V' patterns will somewhat reduce block size by causing collisions of the rocks during heave. It also results in easier to load muckpiles. In any blast the stemming area produces the most oversize as there is usually a lower distribution of explosives than in the rest of the blast. To improve fragmentation the quantity of explosive in the collar area can be increased in a number of ways. The column charge can simply be increased. Smaller diameter and shorter length holes can be drilled in between the main holes and charged with a small quantity of explosives. A separate charge can be introduced into the stemming, known as decking.

6.2 Muckpile

This is defined as a pile of blasted rock that is to be loaded for removal. The 'throw' or movement of the rock from the blast increases as the specific charge increases and may be controlled by varying it. As blasting proceeds in multi row conditions the muckpile in front of the face will gradually lie closer to it. The shape of the muckpile is influenced by several factors.

6.2.1 Drill Hole Angle

An inclined hole will project the rock further than a vertical hole resulting in a flatter outline of the blasted rock.

6.2.2 Surface Timing

A closed 'V1' pattern will result in a steeper muckpile. As the two sides of the 'V' detonate the rocks collide in the air and drop. Open 'V' pattern or lineblast will give a flatter outline to the resulting rock mass.

Diminishing the burden or increasing the hole size will result in the rock moving further and giving a flatter muckpile. The type of loader used in the quarry will determine the most efficient muckpile outline. A front end loader will find it easier to load out a flatter muckpile and a face shovel a steeper one.

6.2.3 Free Face

It is assumed that blasting cannot be effective unless a free face exists towards which movement can take place. This is not so, the free face improves the efficiency of the

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breaking but does not prevent fragmentation from occurring. Choked faces inhibit movement and cost more in explosives consumption, but probably do not significantly affect the fragmentation. In solid ground, without a free face, high powder factors are necessary for good fragmentation. A degree of movement is enabled through the porosity of the rock and voids left by previously fired holes. The practice is not common as other factors mitigate against it, namely ground vibrations, airblast and flyrock.

6.2.4 Fragmentation Analysis

The only sure method of obtaining a muckpile fragmentation analysis is to screen the whole muckpile. Any method developed would also depend on this for an absolute correlation. This is clearly impossible and all methods, therefore, have an unknown degree of error. This can be minimized by working on comparisons, the error then being constant. With time and experience the error can be assessed and almost eliminated.

6.3 Ground Vibrations

The trend towards larger holes and bigger blasts and an increased population has highlighted the problems of ground vibrations. Areas of concern when blasting takes place are shown in figure 14. Damage due to old age or settlement of a building is very hard to distinguish from blast damage. The main criteria is to reduce the level of vibration to such an extent that complaints are minimal. At the same time good public relations will ensure that people living locally will feel reasonably well disposed to the quarry.

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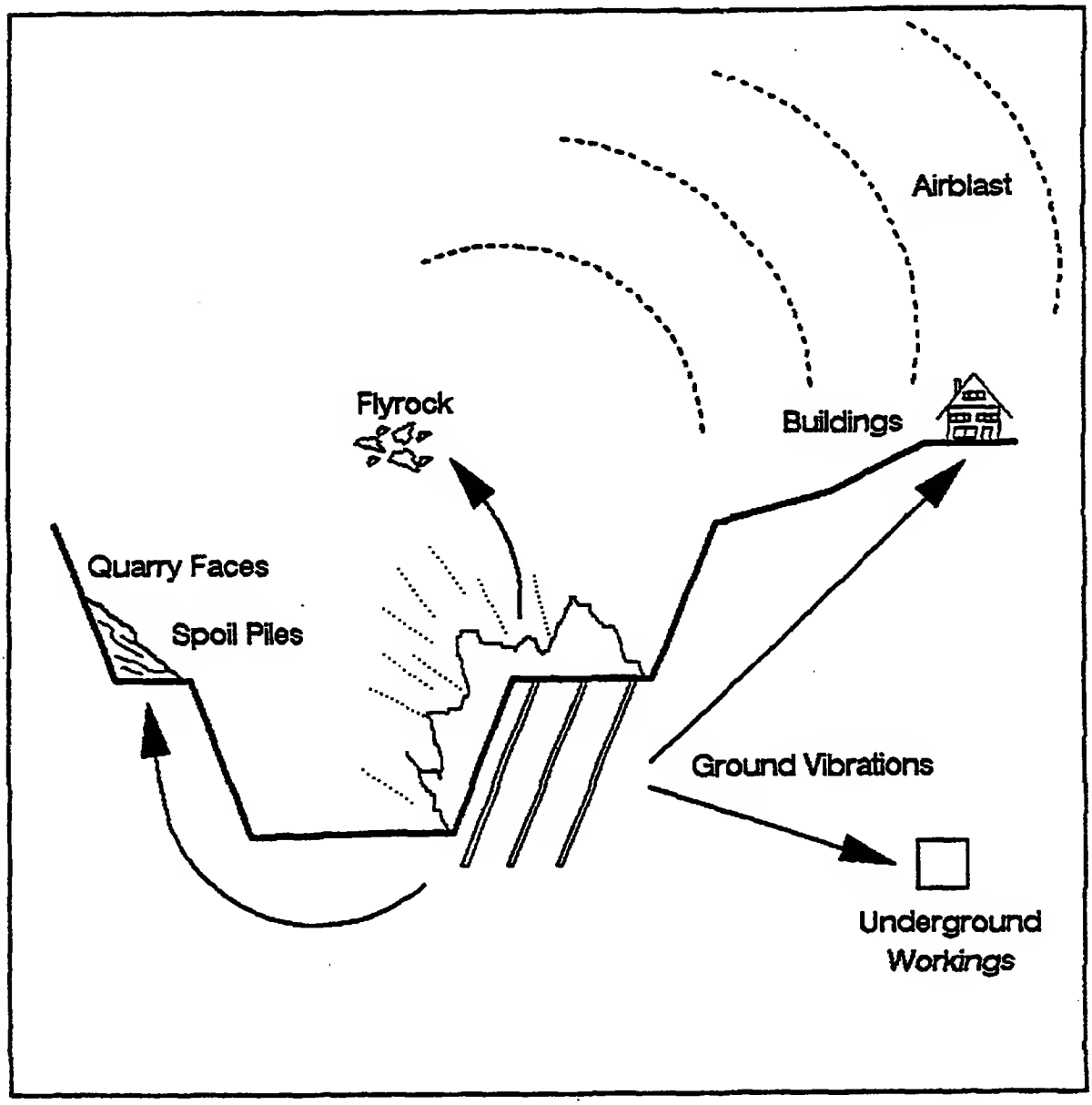
Figure 14: Areas of Concern



When an explosive detonates in a hole it generates an intense stress wave on both transverse and longitudinal wave motions. This motion crushes the rock around the hole up to about one drillhole radius and permanently distorts and cracks it for several more. Most of the energy is spent on shattering the rock but because of the imperfect nature of the explosive, some of the energy is transmitted through the ground as vibration in the form of elastic compression waves. This represents a transfer of energy from one point in the rock to another. Initially there must be some displacement of the rock as certain forces act to displace it from its equilibrium position and introduce new energy to the system.

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6.3.1 Source of Ground Vibrations

If the rock does not exhibit an elastic response, energy is absorbed by it and only dampened waves come from the blast area. If elastic response is exhibited then the action of the blast causes nearby portions of the rock to oscillate about their rest positions, similar to a spring. Oscillatory conditions are set up and the disturbance is transmitted from one element to the next as from the rock to a building. During wave motion there is no bulk movement of matter. The transmission of waves is affected by distance from their source. The rock through which they travel is never perfectly homogeneous, it contains deformities such as joints, bedding planes and different rock types.

Total energy of a ground motion wave generated varies directly as to the mass of the charge detonated. As it travels outward from the inception point the volume of rock affected by the compression wave increases and the peak ground motions decrease. The ground motion wave of a column charge of explosives, where the length to diameter ratio is greater than 6, takes the form of an expanding cylinder. The volume of this compression cylinder varies as the square of its radius. Thus the peak level of the ground motion is inversely proportional to the square of the distance from the blast. In the majority of quarries the detonation of a borehole takes the form of a cylindrical blast. For example a 104 mm hole has to have a minimum single charge length of 0,625 meter for it to be a cylindrical charge.

6.3.2 Defining Peak Particle Velocity

The empirical scaling formula relating peak particle velocity (PPV) to scaled distance has been developed from actual field results. Scaled distance, d/(W^1/2), combines the effect of total charge weight per delay, W, on the initial shock level with increasing distance, d, from the blast. The formula contains two site factors, K and m, which allow for the local influence of the rock on the rate of peak particle attenuation. Geometric progression is included in the slope exponential (m) in the following equation:

V	=	K(d/(W^1/2)^-m
where V	=	Maximum PPV (mm/sec)
d	=	Distance from blast (m)
W	=	Mass of explosives per delay (kg)
K & m	=	Slope of graph
d/(W^1/2)	=	Scaled distance for a cylindrical charge

Site factors are determined from the logarithmic plot of PPV verses scaled distance. The graph is drawn and the best line representing the data is inserted, see figure 15.

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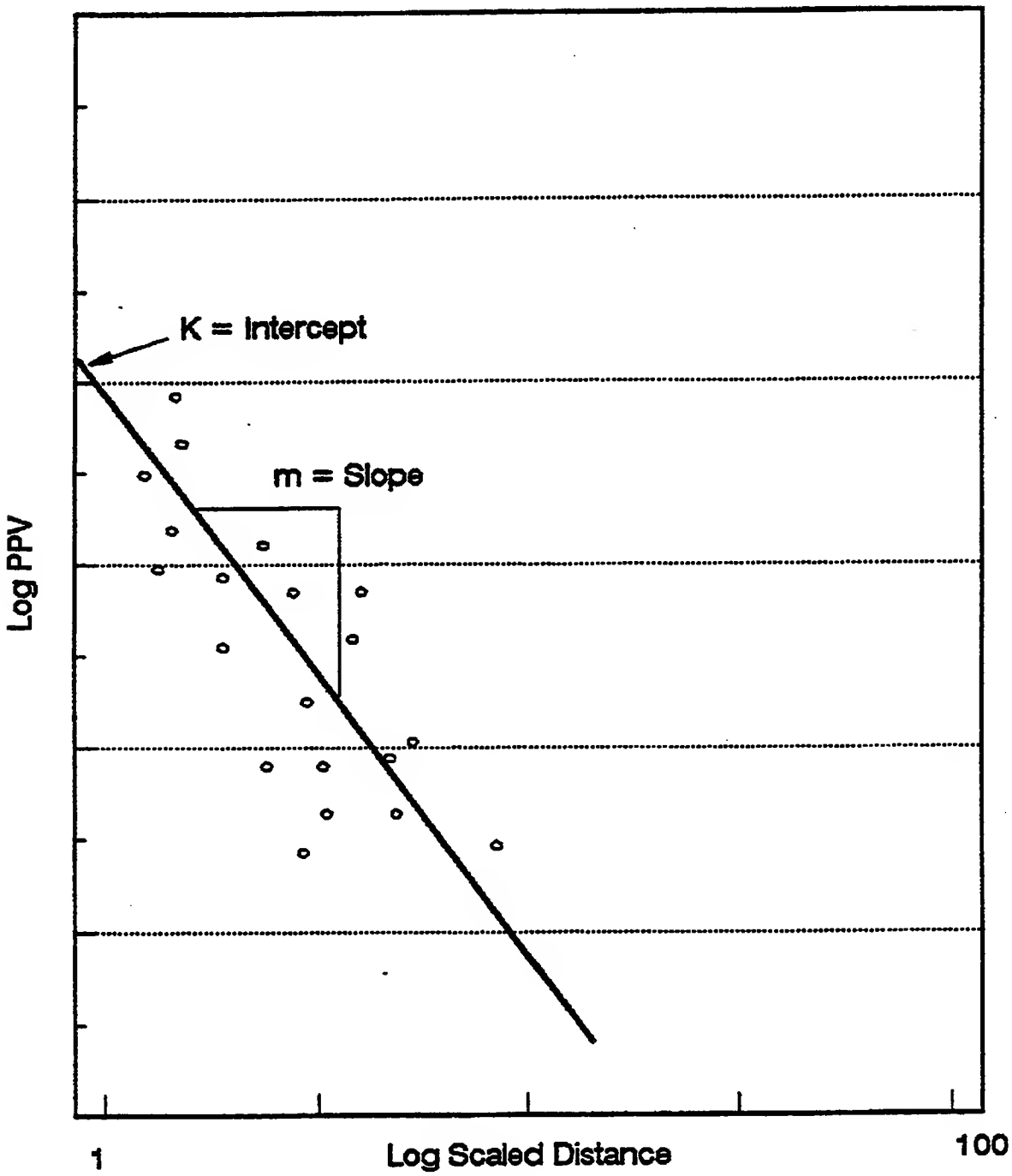
Figure 15: Qualitative Graph of Ground Vibration



Showing the Empirical Site Factors

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Work done by the US Bureau of Mines implied that the best damage indicator is PPV as a function of frequency below 40Hz. A number of studies have correlated levels of PPV with resulting damage and an example of the levels is shown in table 3.

Table 3 Ground vibration levels

PPV (mm/s)	Effect
300	Rock falls in unlined tunnels
200	50 % probability of major plaster damage
135	50 % probability of minor plaster damage
80	Threshold of damage
50	Limit of safe blasting > 12 Hz by USBM
25	Limit of safe blasting < 12 Hz by USBM

The intensity of seismic waves that can be tolerated by structures varies as to the method of construction. A steel reinforced building can withstand a higher level of motion than a privately owned house built of brick. Plaster is commonly used on the inside walls of houses. It is relatively weak when compared to other building materials and is used as the basis for damage criteria.

6.3.3 Techniques to reduce Vibration Levels

To minimize the level of ground vibrations it is easy enough to reduce the charge per delay. This however may not be practicable as it implies reduced production. Several other steps may be taken to reduce levels of vibration:

6.3.3.1 Blast Design

This should give maximum relief of burden. Internal free faces in the blast can reflect compressional waves.

6.3.3.2 Powder factor

An excessive powder factor can increase ground vibrations and may cause excessive throw of the muckpile. Vibration levels can also be increased by an insufficient powder factor. It delays and reduces the effect of rarefaction waves reflected from free faces.

6.3.3.3 Spacing to Burden Ratio

To be greater than one. Overburdening of holes can cause them not to break out to the free face. This results in excessive explosive energy used as shock waves through the rock.

6.3.3.4 Accurate Drilling

Poor drilling can over or under burden the holes with the same results as for poor spacing to burden ratios.

6.3.3.5 Sub Drill

The toe of a drill hole is the most confined area and has the most difficulty breaking out. Over drilling increases the amount of sub drill and therefore the vibration levels.

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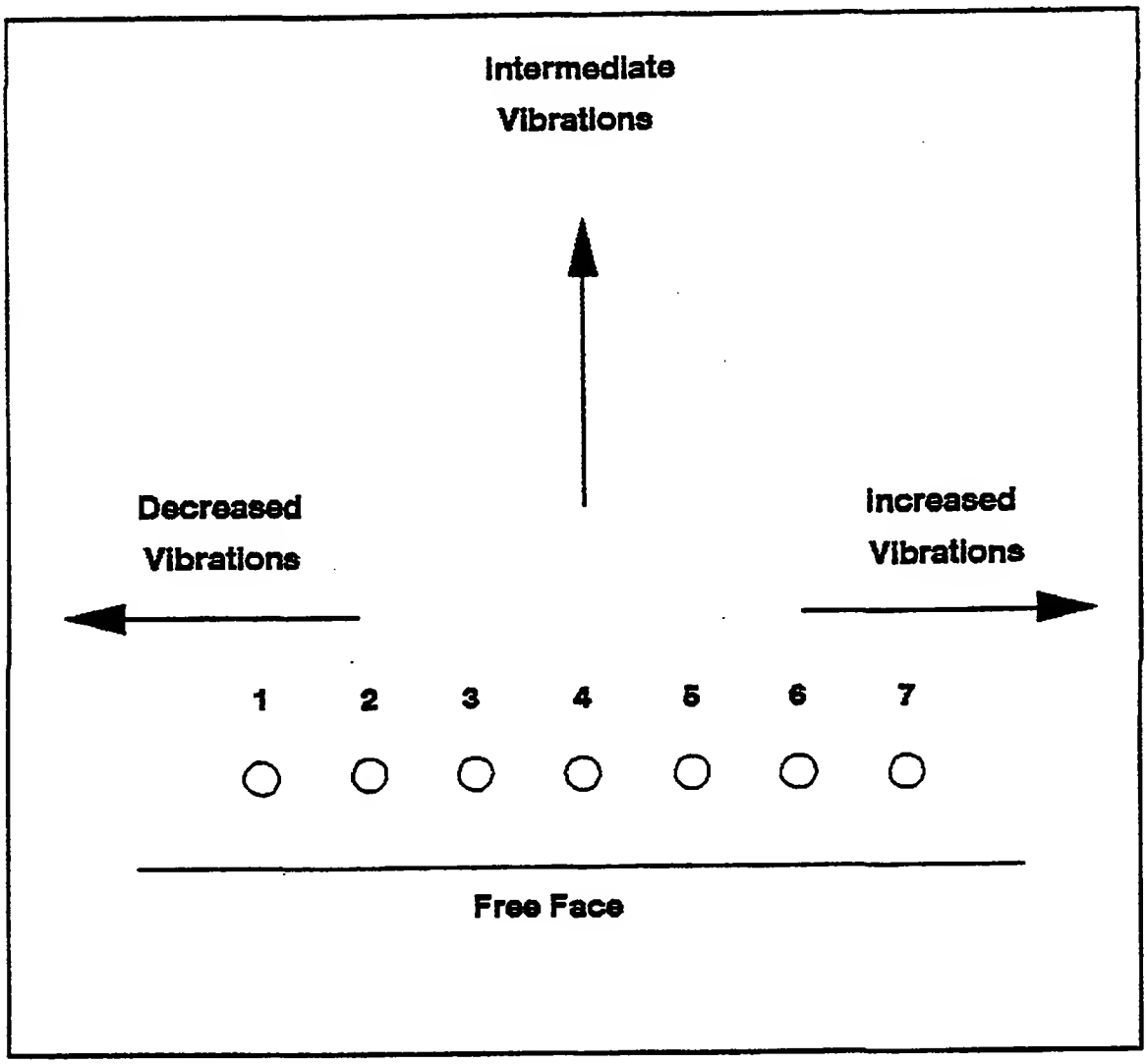
6.3.3.6 Adequate and Accurate Delays

A long delay period between holes will guarantee that they do not detonate simultaneously. Shock waves can be reinforced by waves from subsequent holes in a line detonating if the delay interval is insufficient, see figure 16.

Too few delays in a blast results in a high mass per delay. Studies have shown that millisecond delays in commercial detonators are not very accurate. This can result in close timing or, in extreme cases, an overlap.

Figure 16: Delay Sequence and PPV



5 **Blasthole and Delay number**

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6.4 Airblast

The compressional wave in air, either from unconfined explosives or by indirect action of a confining material subject to explosive loading, is known as airblast. Noise is that portion of the spectrum in the range 20 to 20 000 Hz. Airblast at levels below 20 Hz is known as concussion. Large burdens, typically found with large diameter boreholes, means that the airblast contains a considerable amount of energy at frequencies below 20 Hz. Low frequencies can damage structures directly, but more often causes higher frequency vibrations in windows and doors.

Table 4 Airblast

Noise (dB1)	Effect
180	Conventional structures break
170	Most windows break
150	House windows may break
140	Large plate glass windows may break
136	Interim limit USBM damage to hearing
80 - 100	Loud Radio / TV

6.4.1 Atmospheric Conditions

Atmospheric conditions may affect the intensity of the noise at a distance from the site of the blast. The speed of sound in air varies at different altitudes and temperatures. In addition the wind speed will affect the distance that sound waves carry. Normally the air temperature decreases with altitude, the adiabatic lapse rate, at a rate of about 2.0 degrees centigrade for 300 meters increase in altitude. An inversion, a decrease in temperature with altitude, will cause noise to be reflected back to earth causing excessive noise levels to be heard in unexpected areas.

As the time of day can determine weather conditions, inversions occurring more often in the morning, blasting times should take note of this fact to minimize possible complaints. The effect of wind on noise levels is greater during the cold months because of the higher wind speeds. This helps to prevent inversions occurring. Figure 17 sketches the effect of varying temperatures with altitude wind on noise levels.

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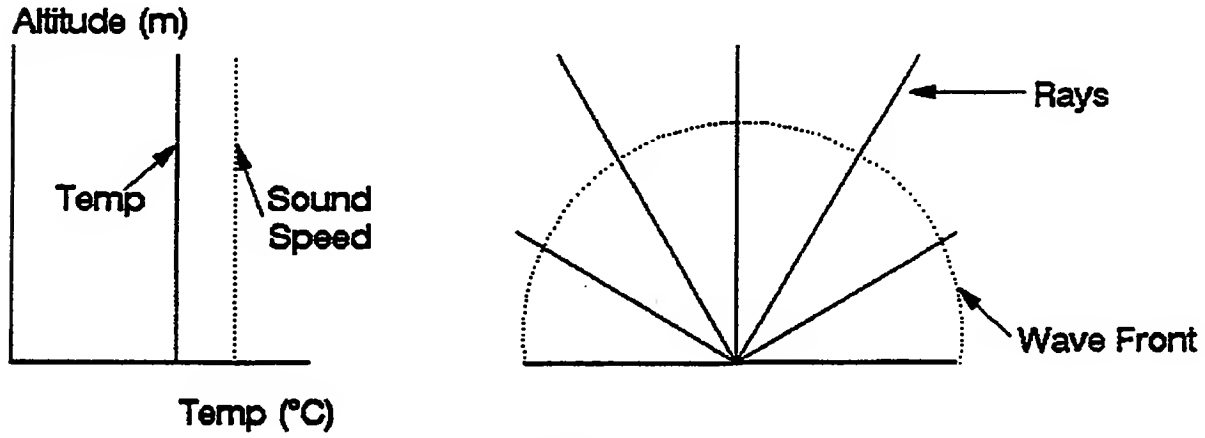
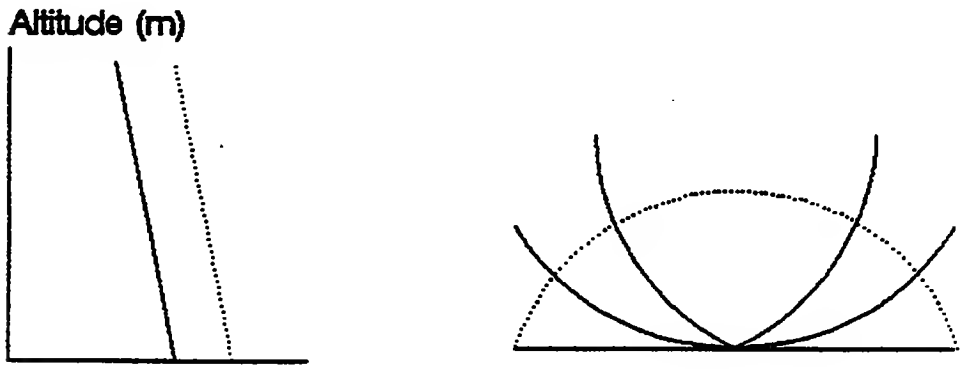
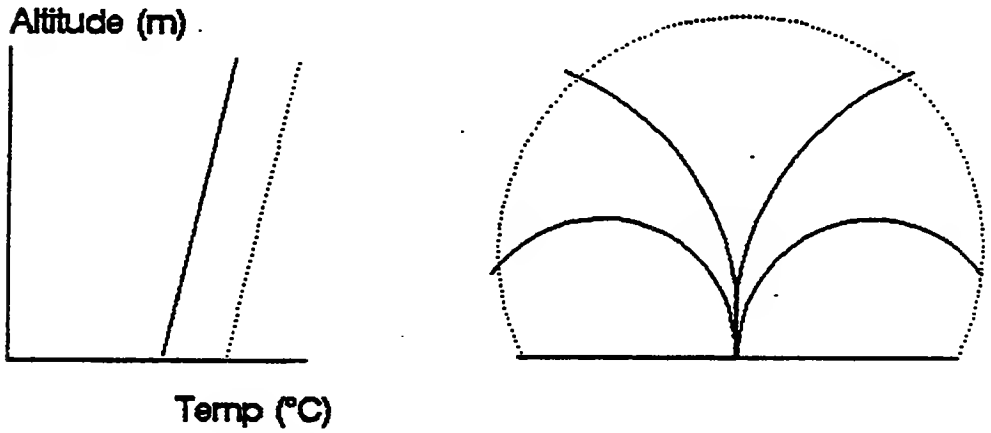
Figure 17: Airblast - Effects of Temperature and Altitude



No Problems



No Problems



Inversion - Potential Problems

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6.4.2 Minimizing Airblast

Minimizing airblast not only depends on the correct weather, a clear day with light wind, but also:

6.4.2.1 Use of Adequate Stemming

Confine the blast stemming blown from the hole results in higher noise levels.

6.4.2.2 Secondary Blasting

Not using mudblasts, breaking rocks with a lay-on charge which is covered with mud.

6.4.2.3 Sequential Blasting

Sequence of blast must proceed in a proper order. Out of sequence shots cause excessive airblast.

6.4.2.4 Time of Day

Scheduling of blasting operations when people are busy. Rush hour generates a high level of noise which will cover blasting noises.

6.5 Flyrock

This is the undesirable throw of rock from a blast. Rocks have been thrown many hundreds of meters and caused both material and bodily damage. It is probably the biggest cause of personnel injury and property damage in the whole blasting operation.

Flyrock is caused by a number of factors:

1) Overloading of holes or insufficient burden
2) Poor quality stemming
3) Incorrect timing of surface delays

Regardless of the care taken with a blast flyrock is always possible and every precaution taken to prevent damage. All blasting personnel must take cover when the blast is detonated. The direction of the blast must be away from any structure which could be damaged.

A quarry situated close to built up areas has to prevent flyrock at all costs. Good blast design is the best method of minimizing this hazard.

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7. LOADING AND MUCKING

The methods of loading and hauling in quarries can be divided into four categories:

1) Continuous mining systems such as the bucket wheel excavator.
2) Other non-blasting systems such as ripping and bulldozing or scraping.
3) Haul road, using loaders and trucks and a ramp to exit the quarry.
4) In pit crushing in which a loader or face shovel supplies a mobile crusher.

Equipment to load and transport rock is dependent on the hardness of the rock to be quarried. Soft rock or overburden can be removed without the need for blasting by ripping and dozing with specialized machinery. There are both discontinuous and continuous systems for the removal of rock without blasting.

7.1 Selection of Equipment

Initially the quality and quantity of any deposit must be established to enable a decision to be made as to the mode of operation. Seismic wave prospecting can be used to discover the hardness and extent of the deposit. This data is compared to previous data obtained in the exploration of other deposits. For many commonly found rocks a range of rippability values in terms of their seismic wave velocities have been tabled with a fair degree of accuracy. Figure 18 shows a comparison of wave velocity ranges over which different loading equipment can be expected to work.

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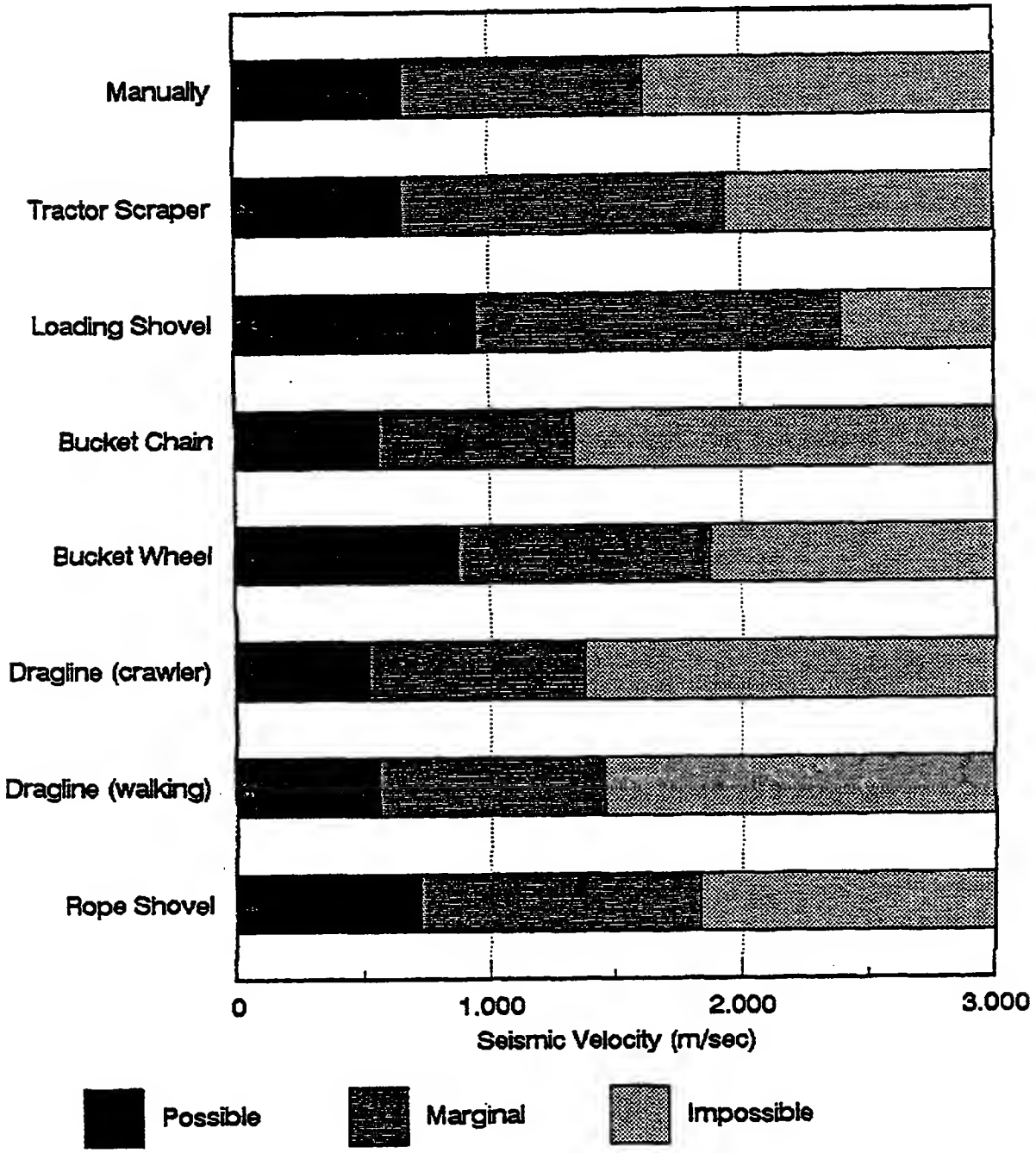
Figure 18: Excavation Method for Seismic Velocity without any blasting to loosen rock



Depending on the planned tonnage to be excavated per year and the consistency of the deposit, both physical and chemical, over large areas determines the equipment to be purchased. Large tonnages of relatively soft material with a shallow deposit favor a continuous mining system. Smaller production levels with varying quality of limestone favor a more mobile system such as a truck and shovel.

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The final selection must take into account overall economics and will be based on balancing the following against each other:

1) Capital Cost
2) Technical Suitability
3) Maintenance and Repair
4) Manufacturers Acceptability

7.2 Non-Explosive Mining

There are several systems of exploitation that do not require the use of explosives. They are generally confined to quarries that have soft rock with a uniform quality over large volumes with few inconsistencies permitting a continuos mining machine to advance in a systematic manner with little or no manoeuvring. Chalk or Marly Limestone are good examples of these softer rocks that are excavated as raw materials in the manufacture of cement. Soft rock permits the material to be exploited without it first having to be drilled and blasted. The rapid advance of mining technology has developed machines that have applications in harder rocks but to be cost effective require a high tonnage output. Capital cost of most continuos mining machines are too high to warrant their use in what is essentially a small tonnage per year operation at a cement plant.

7.2.1 Continuos Mining Systems

Excavation of raw material can be accomplished by the use of a Bucket Wheel Excavator (BWE) or Bucket Chain Excavator (BCE), examples are sketched in figure 19, or a newly developed Surface Miner. If considered as an alternative to conventional exploitation methods the capacity of the plant has to be sufficient to accept the high tonnages involved, some BWE's will move 100'000 mts per day. There also has to be a continuos system of removing material to the plant if the machine is to have a high utilisation time. This usually takes the form of a conveyor belt. Any alternative, such as truck or rail systems, can quickly result in the excavator waiting for loading equipment. Non-continuos systems of transport require detailed planning for them to be used to their fullest and minimise downtime of such a machine. Optimum operation is normally achieved by building an excavating machine and a complete transport system specifically for the required application.

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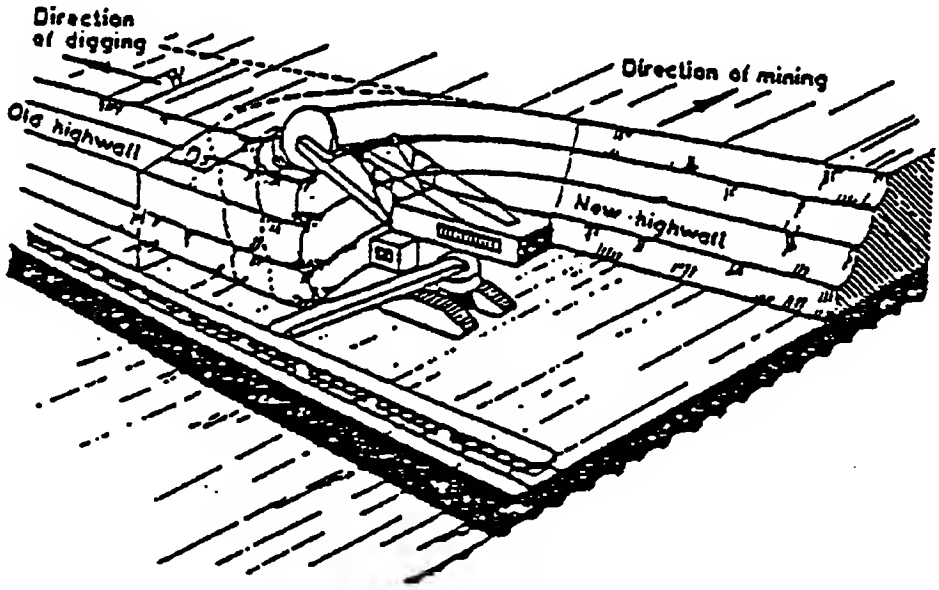
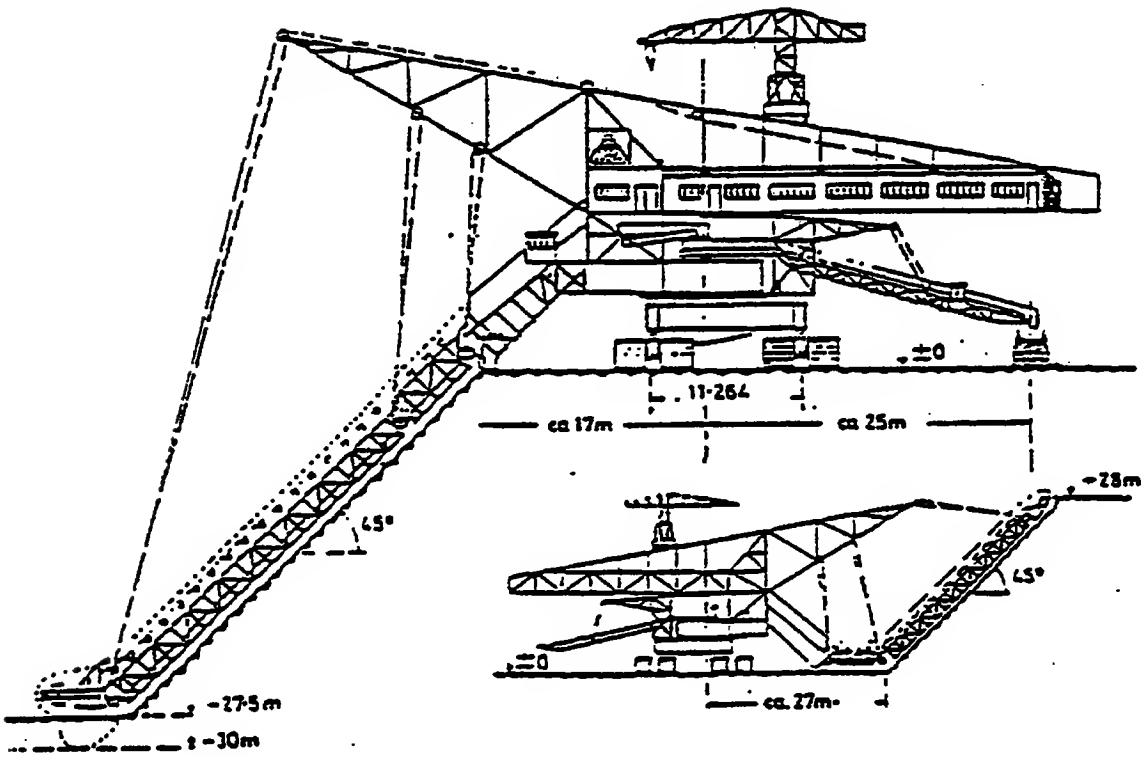
Figure 19: Continuous Mining System



Bucket Wheel Excavator



Bucket Chain Excavator

The advantage of a continuos mining system are that it reduces:

- dependence on diesel fuel, a primary fact if this has to be imported.
- labour complements; although it may be necessary to have a fleet of support vehicles such as bulldozers, personnel carriers and cranes to service the continuos miner
- noise, dust and air pollution in the operation of winning rock.
- haul road maintenance
- power consumption

Development of light weight machines, for example, tyre mounted bucket wheel excavators or surface miners which are considerably cheaper to purchase and have lower capabilities has made the use of these machines in our industry in the future more likely.

7.2.1.1 Bucket Wheel Excavator (BWE)

Methods of BWE operation are described according to the position of the machine and how it cuts the quarry face. Figure 20 shows typical methods of excavation with a BWE. In each method there are two alternative cutting techniques, horizontal, or terrace cutting and vertical, or drop cutting These are used in the selective mining of specific layers for reasons such as quality control.

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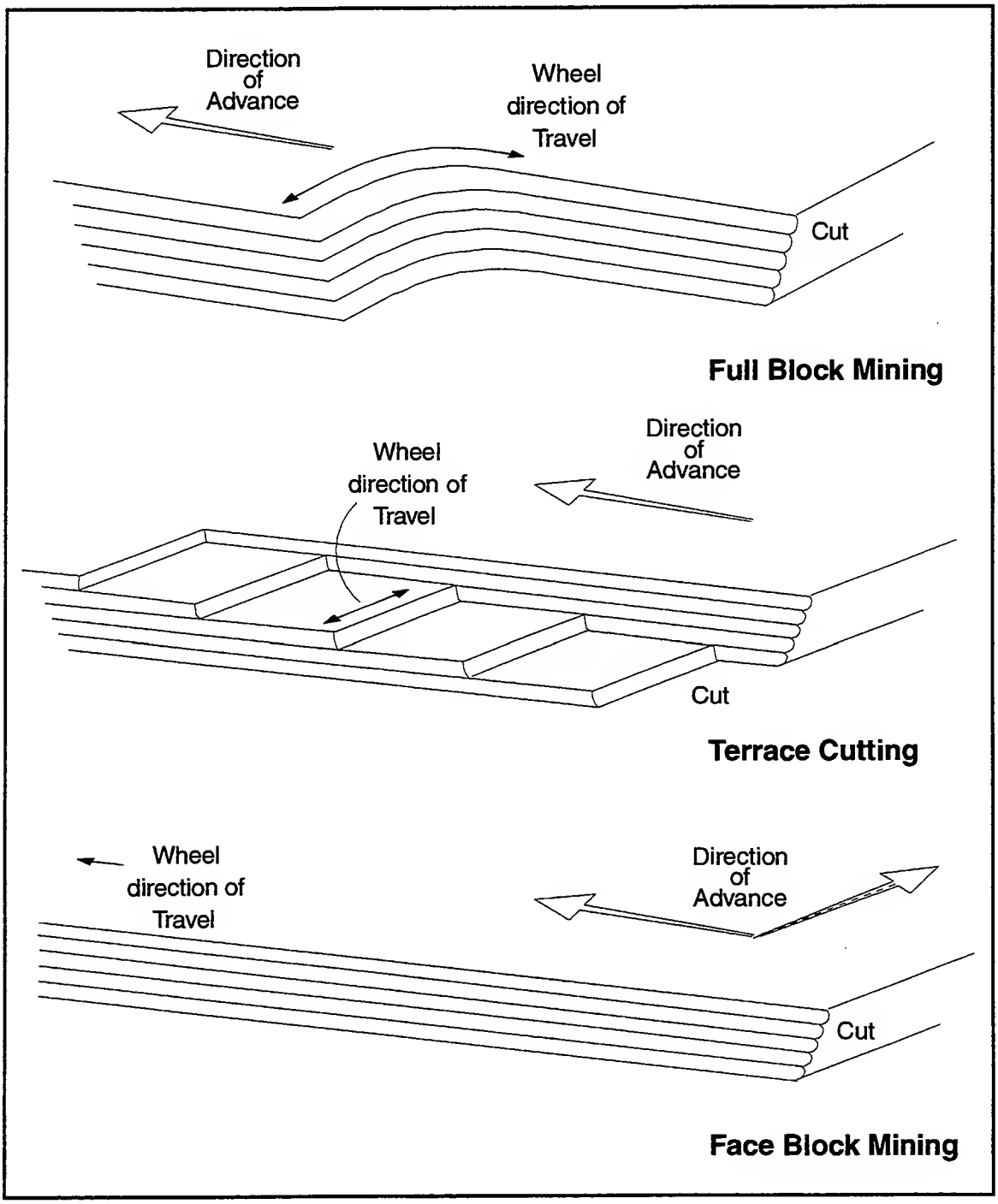
Figure 20: Excavation Methods with a Bucket Wheel Excavator



Originally these excavators were designed to move easy digging materials such as sand, clay and loam. Today they are principally found in the lignite fields of Germany and Australia moving large quantities of overburden. However a newer generation of BWE has incorporated modifications to its design giving better operating parameters in the digging of harder materials. It is this type of machine that may be considered for use in the cement industry of the future. Although in extreme cases blasting can be considered an assist to digging of rock with these machines it should not be considered as a normal practice.

The BWE has little operating flexibility and a detailed study should be conducted before purchasing a machine. Included in this should be detailed quarry layouts based on exploration results and the geology of the deposit. Choice of machine in terms of output and maximum vertical digging depth has to be carefully considered. As a rule the vertical reach of the BWE should equal the final quarry depth.

Large BWEs can compare favourable with a hydraulic or electric shovel in the following areas:

- instantaneous power demand
- weight to output ratio
- power consumption
- wider operating benches and more stable quarry slopes
- close control in selective mining; albeit over large areas
- easier and cheaper rehabilitation of overburden
- an ability to exploit material both above and below the working floor

Selection of a machine for a particular production rate can be more complex than for other types of excavator. Machines of apparently equal capacity can vary in profile dimensions, weight, horsepower etc. A complete system may include a belt wagon, hopper car, mobile stacker and a cable reel car. With all these components the overall availability of the operation can easily fall below 50% unless the operation is carefully planned.

Digging height and cut width will be proportional to the boom length. Efficiencies will increase with greater digging and cut heights but have to be evaluated on the basis of the operational plan and the total economics of the operation given the increased cost of the equipment. In some cases it will be more efficient to use two smaller machines than one large; particularly if selective mining is considered or the stratagraphic layer to be mined is thin. Two BWEs will permit some production if one machine breaks down.

Additional considerations for the selection of a BWE are:

- type of bucket
- tooth design
- transfer feeder for material from the wheel to the belt
- suspension systems
- wheel drives
- boom type

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A long depreciation period is necessary for a BWE to be economical. Larger machines can be considered to have a life span of more than 30 years. Due to its tremendous digging capacity and its high initial cost to achieve the necessary low cost per tonne necessary the machine must operate around the clock with only stops for planned maintenance. It is not uncommon for such a machine to have planned weekly operating hours of 132.

Operating costs are based on two rules :

- digging wheel costs represent 85 to 95% of the total excavator maintenance costs
- total repair costs amount to 5 to 8% of the total excavator costs per year.

Features that greatly increase the cost of a unit are:

- Efficiency Increases in the digging ladder
- Capability of digging below the working floor
- Increases in the normal wheel and bucket size

7.2.1.2 Surface Miners

This a relatively new inovation in the field of continuos miners for use in limestone quarries. It was developed from road repair machines for the quick repair of asphalt road surfaces. Machines consist of crawler mechanism for driving and steering mounted in a frame with a mounted drum of between 1.0 and 1.8 metres diameter and about 3 to 4 metres width fitted with rows of teeth or picks. The drum on quarry production models is often centrally mounted. This position of the drum ensures that the machine's weight maintains pressure on the drum while it digs into the rock. Material is removed by means of a short conveyor belt mounted behind the drum. This will either feed cut rock to a conveyor or load it into a haul truck. The cutting drum rotates in an up-cutting direction, the picks chip at the rock as the drum rotates, and has spirally welded deflection segments attached to ensure that the cut material is directed to the extraction conveyor mounted behind the drum. The rows of picks are tungsten carbide tipped to maximise life and they can dig a "trench" of between 250 mm and 600 mm depth. Broken material is transferred to an on-board conveyor and removed.

Production tonnage's depend very much on hardness of rock and the mass of the machine. The quantities of broken rock produced vary from a low of 300 mts per hour with a small machine up to 800 mts per hour with the largest at present manufactured. Typical product of this type of machine has a high degree of fines, -5.0 mm, with a maximum size of 120 mm. However this can be somewhat adjusted by altering the number of picks and the rotational speed of the drum. One advantage of these machines is that the exploitation and primary crushing are a single process; it does not require a separate primary crusher and in certain cases no crusher is needed in the process. However in feasibility studies a secondary crusher has always been included and if feed material to the raw mill has to be – 25 mm then a secondary and tertiary crusher will be necessary to reduce material to an acceptable size for feed to a ball mill.

Careful planning of the quarry operation is necessary to realise the full economics of the operation. The machine is slow; speeds of about 2.0 ks per hour while digging rock and 6.0ks when moving to and from site can be expected. The quarry floor should be reasonably flat and level over a relatively large area to accommodate a machine. As the exploitation operation is a continuos one material is preferably removed from site by a conveyor belt. However haul trucks can be used. These run alongside the miner in the same direction and are loaded on the move by the discharge conveyor on the surface miner. Figure 21 shows the building of stockpiles of limestone in a quarry where neither haul trucks or a conveyor belt were available. The stockpiles were subsequently loaded using a front end loader.

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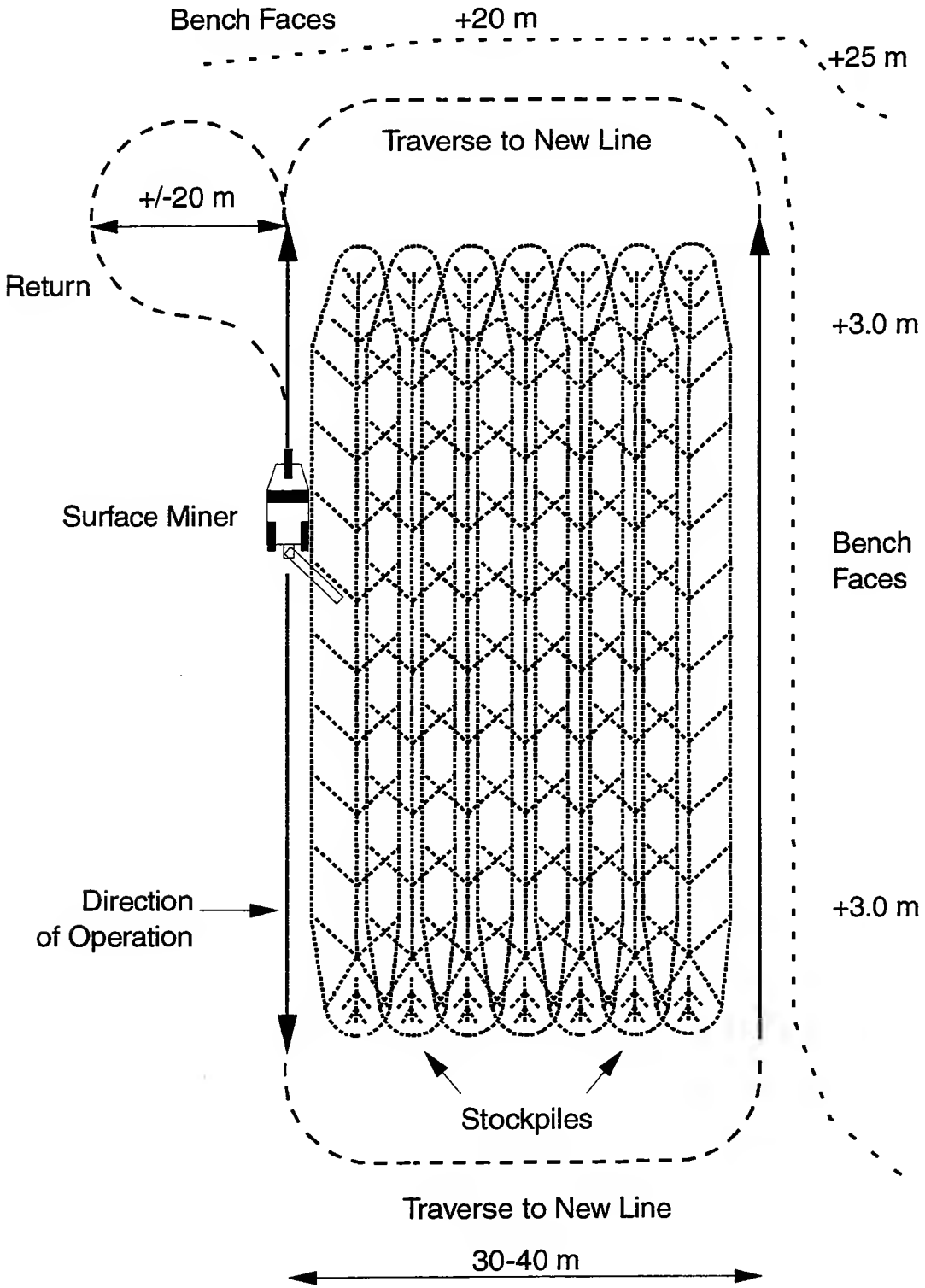
Figure 21: An Example of Surface Miner Operation



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The advantages of this type of machine are:

- labour costs are low, only a single person is required to operate it
- eliminates blasting and minimises dust, the machine is quiet in operation.
- produces ready crushed material

Disadvantages are:

- complex machine requiring considerable maintenance
- as a single production unit breakdowns result in a stoppage of raw material supply.
- requires detailed planning to operate economically and with a high percentage utilisation
- not really suited to haul truck loading

A Surface Miner is considerably cheaper than a BWE but can still cost in the region of USD3.00 million. To maintain some production it would be necessary to purchase two machines if this form of mining was considered.

7.2.2 Semi-Continuos

Other non-explosive systems of winning material find more favour due to the ability of such a system to relocate and their relatively low capital cost per unit. Examples of these are the use of a shovel to load material directly into a haul truck or the use of a ripper on a bulldozer to loosen the rock and a load haul system to move material to the crusher. This enables a plant to have the operational flexibility provided by the use of multiple mobile equipment while not having to invest in new mining systems.

7.2.2.1 Excavation by Face Shovel

The principle reason for the preference of front end loaders in our industry is that they are versatile and highly mobile. They do not have the break-out and crowd forces of a face shovel but this is compensated for by blasting techniques sympathetic to the loading characteristics of the machine. However in certain cases hydraulic face shovels are used in a quarry. This occurs most often where the drill and blast operation has had to be terminated because of environmental considerations and the rock is too hard to be dug with a FEL. In addition manpower costs limit the number of units that can be purchased precluding ripping and dozing.

Material is broken out of the quarry face by a hydraulic face shovel. As the limiting criteria in such a case is breakout and crowd force these machines are often large and have a loading capacity far in excess of the plant requirement. Because of their relative immobility it is the norm to have several of these machines on-site. One such quarry operated four shovels, one on overburden, and three on raw material production; each in a different layer. Material is loaded to haul trucks in the normal fashion.

7.2.2.2 Dozing and Ripping

Material is excavated by a bulldozer pulling either a single or double "pick" at the rear of the machine. These "picks" dig into the ground and are pulled along by the bulldozer tearing out the rock to a maximum depth of about 75 cms. Subsequently the bulldozer pushes the broken rock into piles where it is loaded and transported to the plant. If the rock is soft enough not to require ripping the bulldozer will simply push it into piles with the use of the dozer blade.

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The selection of the most suitable equipment for a particular quarry is important as not every machine will be capable of doing the job efficiently. It may be more cost effective to use blasting to loosen the rock even though it may be soft enough to rip or scrape. Selection of machinery for the ripping or dozing operation depends upon:

1) Down pressure available at the tip. The ripper must be able to maintain penetration of the rock.
2) Flywheel horsepower. Power to advance the tip.
3) Gross weight of the machine. This determines if sufficient traction horsepower is available to permit full drawbar pull.

Various combinations of ripping and dozing are in use, the ideal case being a one pass system. In some situations the dozer is used to physically push the ripper where the rock conditions are hard.

7.2.2.3 Scraping

Scraping involves both the loading and transporting of the rock in one machine. Conditions rarely exist where this system can be used as the rock must be soft or loose. These machines are used in conjunction with rippers to excavate material loosened by them. Scrapers can be used in overburden removal where it is soft.

7.3 Haul Road

This involves building ramps up or down the side of quarries and using dump trucks and loaders to haul rock from the blasted area to the primary crusher, usually situated outside the quarry. It is a very versatile method of operation and is suited to any terrain, from flat ground to mountainous, and any size quarry. This method of moving rock to the crusher is the most common in small to medium quarries. The trucks will range in size from 20 to 80 tons. Loaders range in bucket capacities from 3 to 13 cubic meters, based on the specific gravity of the rock to be loaded.

7.3.1 Loaders

Four types are commonly used in quarries:

- Electric Shovel
- Wheel Loader
- Hydraulic Excavator
- Tracked Loader

Electric mining shovels have been in use for many years and because they are simple machines tend to continue working efficiently. Their main disadvantages are their large initial cost and, if electric, lack of maneuverability. They have now nearly disappeared from the average sized quarries superseded by hydraulic loaders and shovels.

The advantage of wheel loaders is their high mobility and with recent advances in technology in respect to size and digability they can equal or exceed the excavating rate of face shovels. Front loading hydraulic excavators are a recent development. With their extremely short cycle time and relatively high power they are becoming more popular. The capacity of these loaders is quickly reduced with increasing distance between loading point, at the muckpile, and delivery to a truck. The optimum is 10 to 20 meters and for any distance greater than 120 meters the capacity of the loader is almost halved.

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The most impressive feature of the hydraulic excavator is the high mobility of the bucket and the high breakout force which can be transmitted to the bucket. Because of its mobility it can be employed at a variety of locations in the quarry. A further advantage is that a hydraulic excavator will load at less cost per ton than a wheel loader. They allow for the careful, thus reducing spillage, and quick loading of trucks thus achieving short cycle times. This improves the efficiency of both loader and trucks and helps to keep down cost. Drives and hydraulic assemblies are, today, designed for easy exchange which reduces maintenance times. A higher degree of technical knowledge and cleanliness of the maintenance area are required than for non hydraulic equipment. This has, in the past, resulted in low availability of machines where cleanliness was not adhered to and maintenance and repairs was carried out on the quarry floor. Although these excavators are more mobile than the electric face shovels capacity rapidly falls off with increasing loading distance between loading and delivery.

Table 5 compares the features of the three types of loaders commonly found in quarry applications.

With the track type loader the dozer pushes the rock over an edge so that it can travel downwards towards the crusher site, on the lower levels, by gravity. This negates the use of trucks with the need for their associated well maintained haul roads. Narrower benches with lower stripping ratios can be planned as well as steeper gradient roads. Today dozers are used more often to remove rock from the bottom of a bench where it may impede subsequent blasting operations and in conjunction with loaders and trucks to optimize the cleaning of faces.

As the tonnages per hour loaded increases the effective cost per ton decreases. This will happen when bucket fill efficiency and swing time improves as fragmentation size decreases. Shovels, loaders and excavators were made to load broken rock not to dig it.

Table 5 Comparison of loader characteristics

Summary of Features Machine Type	**Face Shovel**	**Rating** **Front End Loader**	**Hydraulic Excavator**
Mobility	V Poor	Good	Poor
Break out Force	High	Low	High
Long reach	Yes	No	Yes
Versatility	V Poor	Good	Poor
Operator fatigue	Low	High	Low
Clean up ability	Nil	Good	Marginal
Load over truck end	Yes	No	Yes
Capital cost	High	Medium	High
Trade in value	Low	High	Low
Limited space operation	Good	Poor	Good
Separation of big rocks	Good	Poor	V Good
Operate in poor ground	Yes	No	Yes

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7.3.2 Trucks as Haulers

In quarries the haul truck is the most economic and versatile method of moving rock or overburden from ever advancing work faces. Haul trucks range in size from 17 to over 200 tons. Those commonly found in quarries are between 35 and 80 tons. Payloads and horsepower has risen on many models in recent years and together with improved engine technology and fuel injection has reduced the unit cost of hauling. To match the system the right number of trucks is required with a bed configuration and size to make a good loading target permitting the quick and easy loading distribution. The majority of these dump trucks are of the rigit, two axle, variety.

An articulated truck, of two or three axles, is designed to work in areas requiring high maneuverability where the ground conditions are poor. They are generally of small capacity, 25 to 40 tons and there bed configuration is such that they must usually be loaded over the tail. These trucks are becoming increasingly popular in areas of selective mining where, with a backhoe as a loader, careful loading is the norm.

7.3.2.1 Matching of Loaders to Truck

Full trucks, loaded quicker, will reduce hauling costs. Tire wear is reduced as less rock is spilled on the roads. Easy loading will allow efficient scheduling of the trucks and less waiting time.

Possibly the main factor when selecting a loader is how many passes will it take to fill the hauler? An even number of passes is a must to eliminate the odd half bucket at the end or to send the hauler away without a full load. A loading cycle should be completed in two minutes. This is about the time it takes for the crusher to consume a truckload, if it is a correct match. It allows three or four swings of the bucket to fill the truck. A loader that takes two swings may result in damage to the truck's suspension as the load dropped, by the bucket, into the back will be greater than the trucks' designed capabilities. Too many swings means excessive time to fill the truck resulting in long waiting periods for the trucks to load. The optimum passes to load a truck are shown in table 6.

Too many trucks to cater for a small loader gives truck overcapacity, too few will result in delays if one truck breaks down. To cater for re-sizing of a fleet may see the need for a high lift loader. This will result in a small decrease in the bucket size and an extra pass or two.

Table 6 Truck loading

Loading Machine	No. of Passes
Excavator	3 to 4
FEL	3 to 5

Figure 22 shows a match between loaders and trucks of various sizes.

These figures maximize truck usage and results in minimal loader standing time. If at the same time the loader has little maneuvering or waiting time the transport capacity is equal to the loading capacity. This is the most economical level of loading and hauling.

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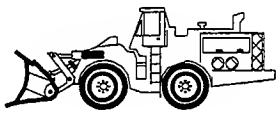
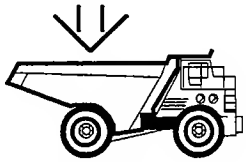
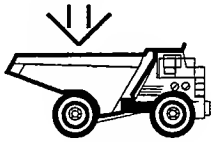
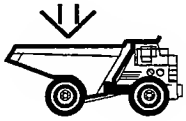
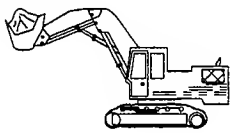
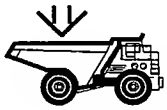
Figure 22: Matching Loaders to Haul Trucks

Production (t/hour)	Loader	Bucket Capacity (m^3)	Truck	Passes
1600		10.3	80 t Load 	4
1400		9.10	60 t Load 	4
800		5.40	50 t Load 	5
700		3.80	40 t Load 	6
600		4.30		5

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Some manufactures publish payload ratings for their loaders. There are specific standards by which payload is calculated. If unavailable then a 'rule of thumb' is that the payload can be estimated by multiplying bucket volume by broken material density. An approximate figure for material density is 1800 kgs per cubic meter. This is the approximate weight of one cubic meter of broken granite or limestone.
For an 8 cu-m bucket: 8 * 1800 = 14.4 tons

7.3.3 Rolling Resistance

This is the retarding force of the ground against the wheels of a vehicle. This force must be overcome before a vehicle can move. Many things determine the rolling resistance or the retarding force acting against a rolling wheel. Among the most important are:

1) Internal friction
2) Tire flexing
3) Penetration of the wheel into the surface
4) Weight on wheels

These effects should amount to about 2 % of gross vehicle weight. This is the rolling resistance factor of a truck traveling on a concrete highway. Shown in figure 23 is the rolling resistance for a 60 ton truck traveling over various roadway surfaces. it can be seen that the resistance can increase dramatically if the surface is not well maintained.

Rolling resistance does not however affect tracked type vehicles. This is because they carry their roadway with them and steel is always hard and smooth. Only internal friction has any affect on these vehicles.

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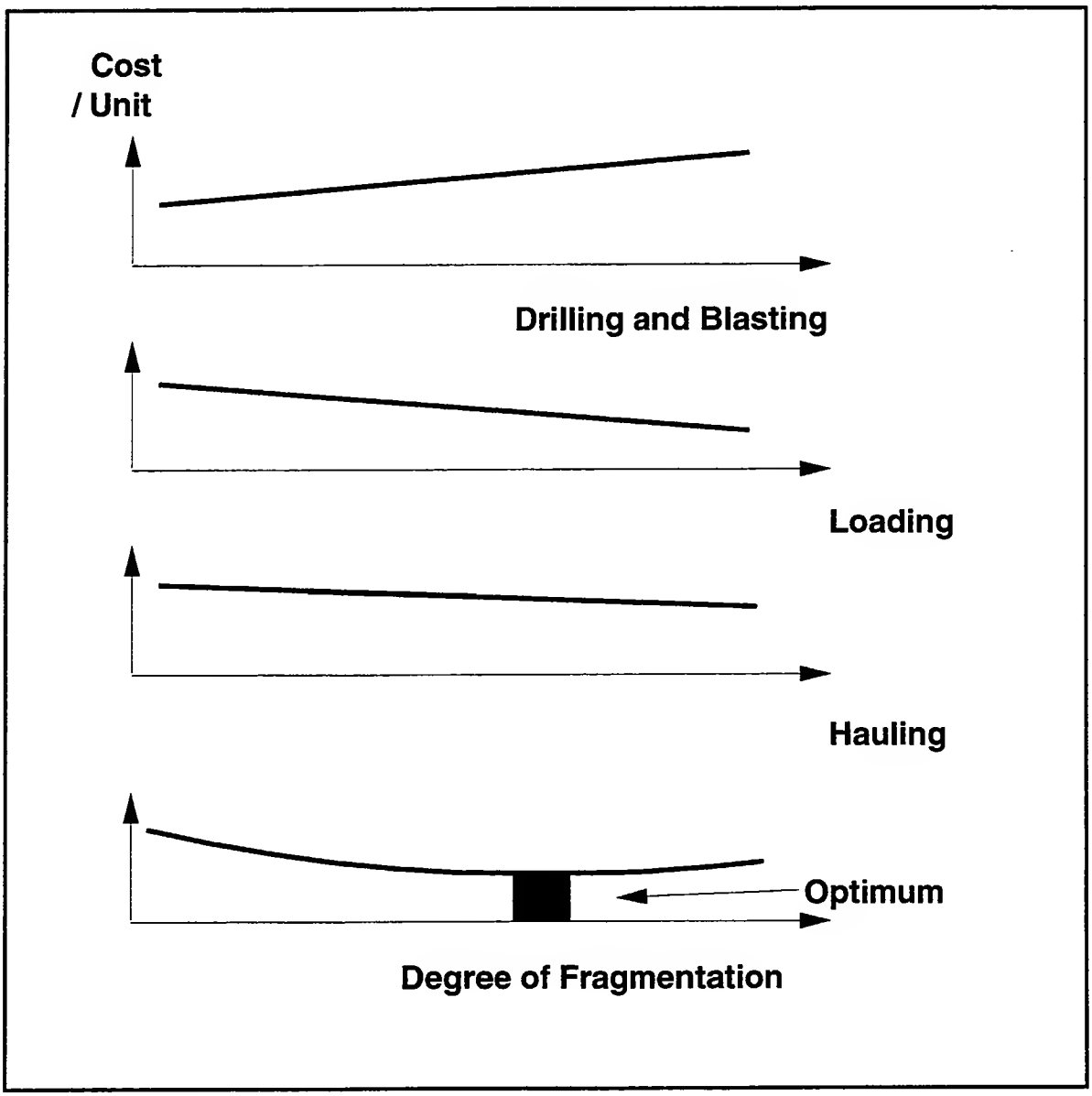
Figure 23: Rolling Resistance



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7.3.4 Grade Resistance

It is the force of gravity which must be overcome when going uphill. It acts against the total weight of any vehicle. Grades are usually expressed as percent (%) slope, which is the ratio between vertical rise or fall and the horizontal distance over which this occurs. When the grade is uphill, adverse, the effect is to demand more power. If downhill, favorable, then it is a helping force producing additional pull to propel the vehicle. This downhill effect is commonly called grade assistance.

7.3.5 Effective Grade

This is the total resistance expressed as the grade which would have to be negotiated to get the equivalent resistance to motion.

Since Total Resistance = Rolling Resistance + Grade Resistance

and
1 % of grade produces a grade resistance of about = 10 kgs/t of vehicle weight

Effective Grade % = Rolling Resistance (kgs/t) / 10 + % Grade

7.3.6 Cost Relationship

From figure 24 it can be seen that as the cost of drilling and blasting increases so the cost of loading and hauling decreases. This is mainly due to the easier loading conditions that exist when the fragmentation of the muckpile is finer. Loading is easier, there is a smaller percentage of oversize rocks in the muckpile, and hauling is quicker as the trucks do not have to wait at the crusher while blockages are cleared, to give two examples. Studies have proved that over blasting of the rock, resulting in fines fragmentation, will actually decrease the overall cost of the quarry operation.

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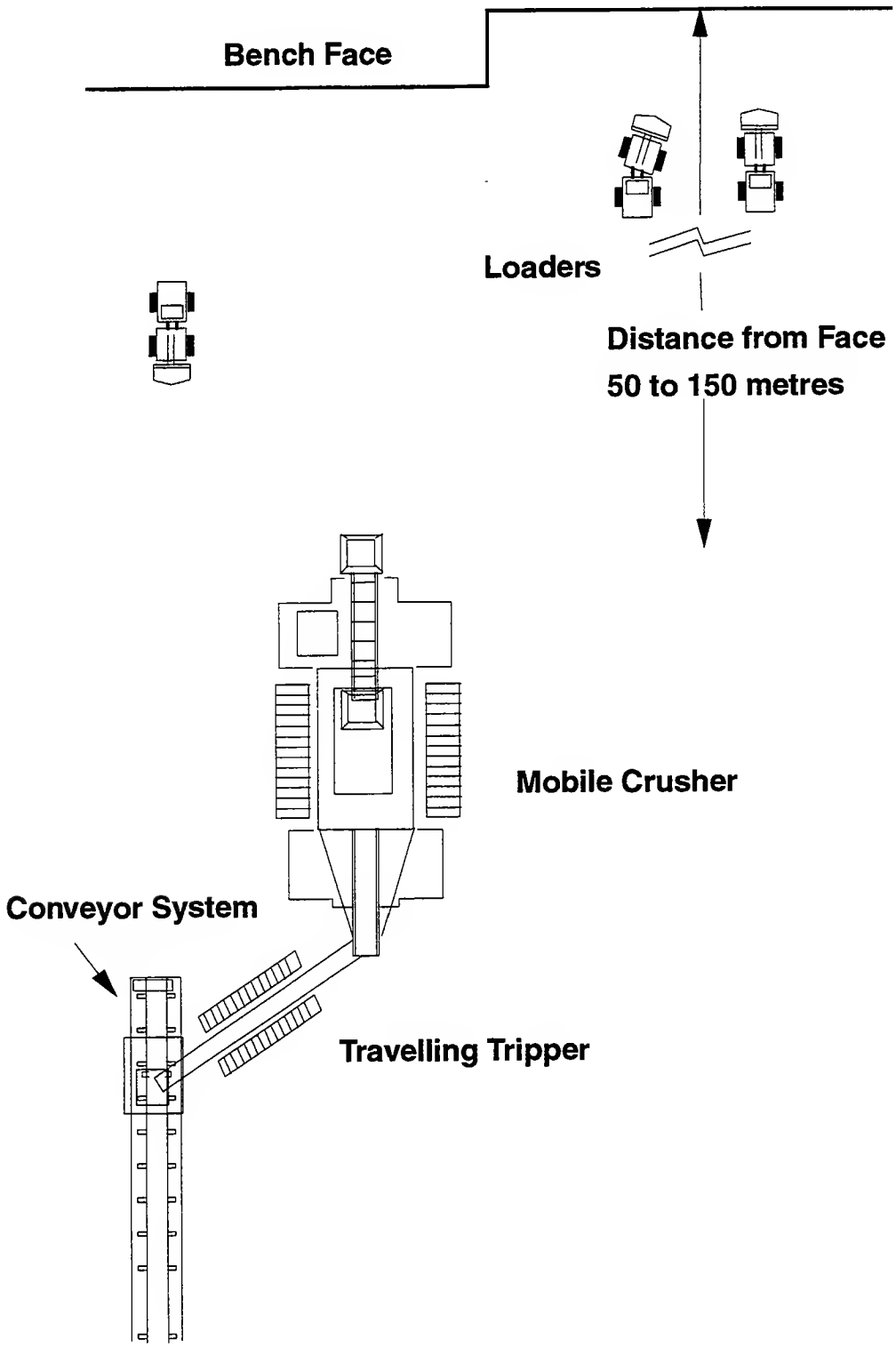
Figure 24: Cost as a Function of Degree of Fragmentation



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7.4 In Pit Crushing

Transport between the loader and the fixed crushing plant is nearly always by a discontinuous transportation method, trucks. This can be considered a disadvantage in that either end of the transport system is in continuous operation. As the quarry faces are always advancing the site of the crusher does not remain optimum for long. To facilitate this an ever increasing number of trucks is required to maintain production with associated increased costs. An answer to this problem has been to install in-pit mobile crushers which considerably reduce the need for trucks. Rock is loaded from the muckpile to the primary crusher and is then fed to a conveyer belt to be transported to the plant. The crusher in the quarry is usually mobile so that it can follow the advancing faces. This is made possible by the addition of conveyor belts when the unit is moved forward. Figure 25 sketches a typical system. Secondary and tertiary crushing systems are, if required, on a fixed site outside the quarry. The capital investment to purchase a mobile crusher is higher than for an average fleet of trucks. However the savings in manpower and increased output from the loading equipment reduce the overall working costs and make the unit justifiable.

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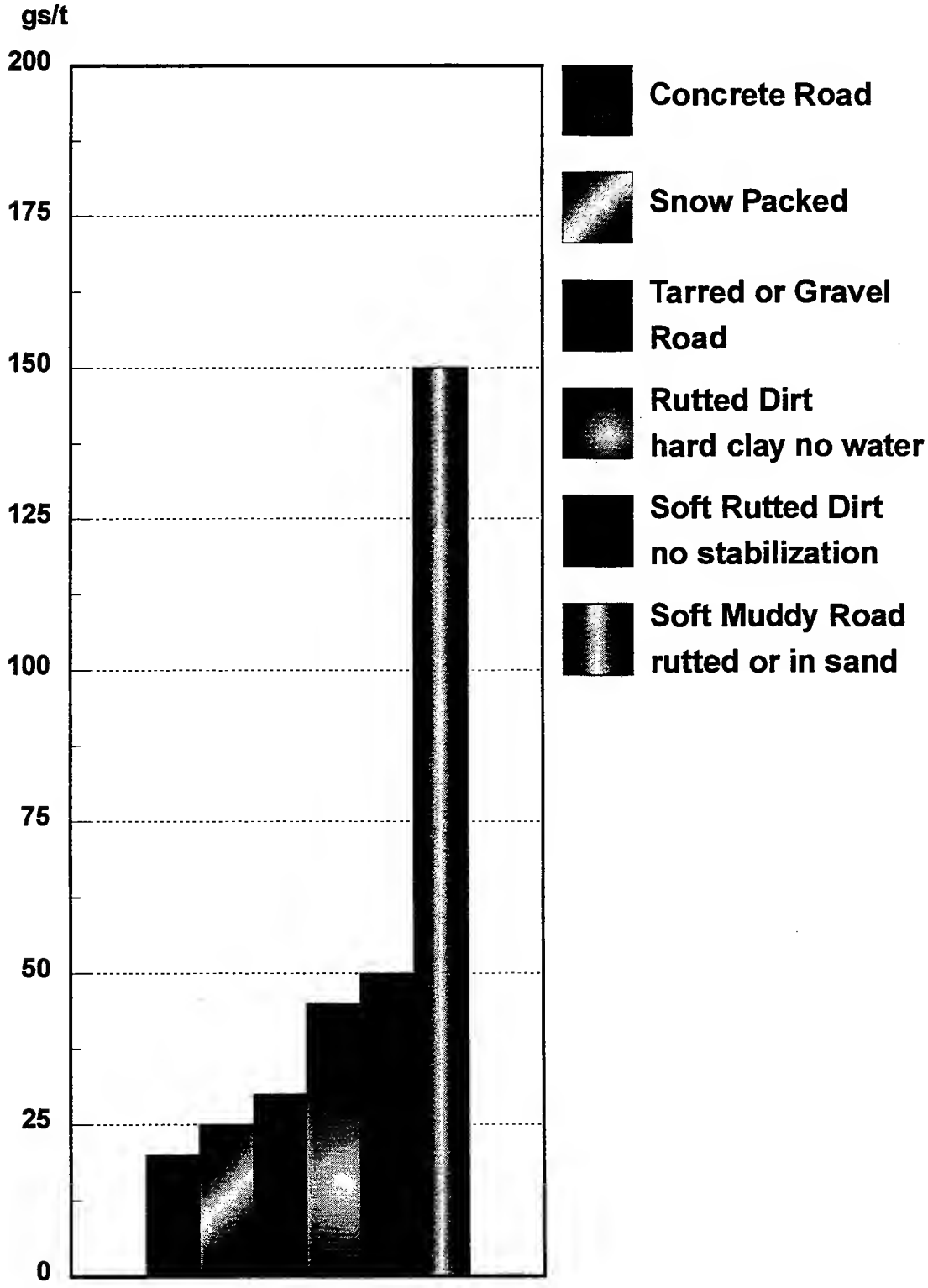
Figure 25: Mobile Crushing System



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8. CONTRACTING OUT THE QUARRY OPERATION

There is a trend to contract out the quarrying operation in our industry. It has resulted from the belief that experience in quarrying is limited and that capital investment can be saved from not having to purchase mobile equipment. This initially manifest itself in the contracting of the drill and blast operation where it was found to be cost effective. Out-sourcing the entire quarry operation appears to have several advantages.

- payment is for guaranteed work done
- it is possible to inspect the work before payment is made
- competition between contractors can lower the cost
- provides additional expertise in the day-to-day running of the operation
- eliminates peak vehicle requirements

It is necessary to supervise the contractor as in all operations and this can be done in a number of ways. The most common method of recording tonnage delivered to the crusher is to send every truck load over a weighbridge. However this is both time consuming and wasteful of the contractors resources. It is more common to pay a contractor for the amount of clinker produced. This is a much more reliable indicator of the tonnage moved. The requirements of the plant should be detailed and the following points should form part of the contract:

- payment on the quantity of clinker produced
- limits on the size of material delivered to the crusher; preferably a size distribution of the incoming raw material
- limits on the quality acceptable
- tonnage on a daily basis
- aspects of the companies safety policy
- an agreed environmental protection policy

Included in the working cost of any out-sourcing must be that cost for the owner to supervise and manage the operation. In many cases the regulations specify the responsibilities of the owner and specific mention is made of his duties which cannot be given to the contractor in a quarrying operation. Due in part to this fact the long term cost of operation is usually cheaper if not contracted out.

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9. RECLAMATION

Public acceptance of quarries is, in part, dependent on the creation of a suitable after use. A range of uses is possible from lakes to development of urban housing schemes. All to often reclamation of an old quarry is unrealistic or unfeasible.

9.1 Restoration to Agricultural Land

This is probably the most common restoration option. The techniques used are based on a complete understanding of the soils, geology and hydrogeology of the site. The various soil horizons will be stripped separately and stored for future use. Handling has to take into account damage to the soil that occurs when moving it. When replaced a five year care program of cropping ensures that the restored farmland is productive. The quarry is usually filled with overburden or imported wastes to bring it to its former outline. The use of domestic and industrial wastes is not recommended as seepage of ground water could lead to contamination of rivers and lakes, and also have a detrimental effect on the reclamation project. High costs of returning worked out areas to farmland, occasionally several times the market value, has led to it being developed for other uses.

9.2 Forestry

Restoration to woodlands is usually only possible on the worked out floor of quarries or on sites filled with inert material. There are also problems associated with drainage due to compaction by earth moving equipment. Artificial ridges and furrows can overcome this. Tree planting restores the balance of tree loss caused by the demands of agriculture.

9.3 Recreational

Many wet pits have been turned into small lakes for sailing and boating. If a constant supply of oxygen rich water is available a quarry site could be turned into a fish farm. Where the surface is dry, playing fields or golf courses have been established. The same principles of reclamation apply as for agricultural use. If this type of reclamation is adopted then it can be highly profitable to the owners of these worked out quarries.

9.4 Nature Conservation

These sites tend to happen rather than be designed. Many old quarries become filled with rain water and attract wildfowl, especially if in a remote, sparsely populated area. Once established they can become important nature conservation areas.

9.5 Urban Uses

Many quarries came into existence because of their proximity to urban areas. With the rapid growth in population in the last few decades they now find themselves surrounded by urban areas where once was open country. Few problems are posed by the use of quarry floors for buildings. The rock base is stable for foundations, only the slope angle will have to be modified. If domestic housing is proposed then a supply of topsoil will be needed to establish gardens. Unsightly factories can be hidden from general view by placing them in old quarries.

Building on fill is not to be recommended. Subsidence may affect buildings and there could be problems with gases emanating from the fill. In some cases methane explosions, with gases from fill, have occurred.

With a positive after use a quarry is much more likely to be tolerated by a community. Reclamation should be ongoing as the quarry progresses. Even a line of trees, to shield the working from view, will assist in its acceptability.

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10. THE FUTURE

With rising costs affecting profits the need to increase productivity to minimize this means that the move towards mechanization and the use of computers will increase. However the cost of replacing any machinery incapable of mechanization or new computer hardware must be taken into account. The benefits can include, reduced operating costs, maximum equipment utilization and increased safety for workers.

Some of the areas in the quarry where increased productivity can be achieved are:

10.1 Drilling

Drilling rigs can be computerized leading to more accurate spotting of holes and more accurate drilling. Drillholes will be accurately laid out with use of computer programs, and a disc given to a rig operator who inserts it into an on board computer. This will line up the rig and collar the hole independently of the operator, removing him from the error equation in the optimization of the drilling and explosive costs.

10.2 Explosives

The use of mobile manufacturing plants in the manufacture and delivery of explosives. Instead of having to build and service magazines for the storage of explosives and accessories they will be delivered to the blast site, from the manufacturer, at a time requested by the blaster. The composition of the explosive can be varied to suit ground conditions on site, even within a single hole.

10.3 Accessories

Electronic detonators which contain a microchip to give extremely accurate timing. The scatter in the actual detonation time of present delay detonators can cause either an overlap of shots of different delays detonating at the same time. This gives rise to increased noise and ground vibration levels as well as poor blasting results.

10.4 Controls

Computer aided survey packages to profile bench faces and to plot drillholes accurately. Actual burdens and spacings anywhere in the length of the holes could be plotted and more accurate estimation of the explosive required made. Monitoring, closely, the actual lengths off holes can prevent overdrill with resulting poor quarry floors.

10.5 Planning

Traditionally planning has been divided into short term, one to three months, and long term, two to five year, planning. Every year a week is spent updating these plans and explaining the difference between actual and achieved. The use of computer aided planning will keep a closer control on the raw materials production and optimize the extraction in both, quantity and quality. Planning can be updated whenever necessary as daily results can be quickly entered.

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HAUL ROADS

- A truck is productive if it is hauling rock.
- Highest productivity only on best haul roads (cycle times).
- The main roadway should be established in ground that requires a minimum of maintenance.

HAUL ROADS

- Features:

	Safety and Health
➢Long life	
➢Minimum bends	
➢Wide as possible	$$$$$$
➢Good surface	
➢No potholes	
➢Crossfall	Productivity

HAUL ROADS

- Enemies of the roads:
  - ➢Water
  - ➢Too heavy payload
  - ➢Inclination
  - ➢Frequency of trucks (Swinging trucks)

HAUL ROADS

- Enemies of the roads:

HAUL ROADS

HAUL ROADS

- Affects costs:
  - ➢Tires (5 to 15k US$ each)
  - ➢Fuel (rolling resistance from 11 to 3% => 1/2 fuel)
  - ➢Maintenance on equipment
  - ➢Productivity due to velocity of equipment
    - ➢ Down time / Death time
    - ➢ Overtime
    - ➢ More equipment necessary
    - ➢ More labour necessary

HAUL ROADS

➢Last but not least: Safety of Personnel

7 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

8 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

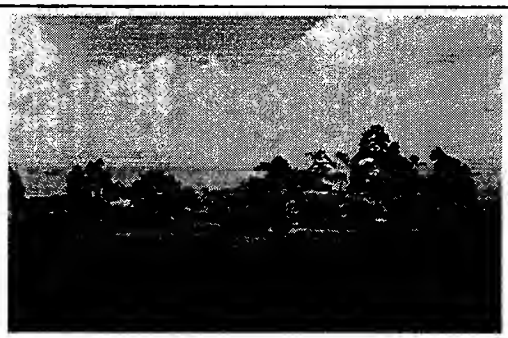
HAUL ROADS

9 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

10 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS



11 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

■ Geometry of a haul road:

- ➢Number of lanes
- ➢Safe distance
  - ➢ Drivers reaction time = 2 sec.
  - ➢ Allowance for error = 5 sec.
- ➢Road width
- ➢Super-Elevation
- ➢Safe sight distances
- ➢Camber
- ➢Bend

12 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

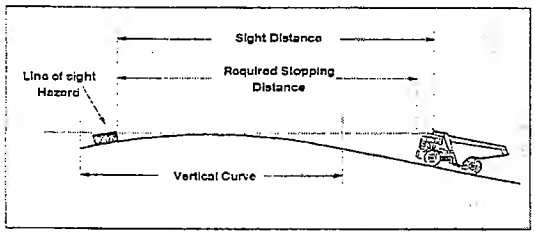


Figure 3 Vertical curve Line of Sight

13 Roads.ppt KZ/KZ - 06.05.00 HOLDERBANK

HAUL ROADS

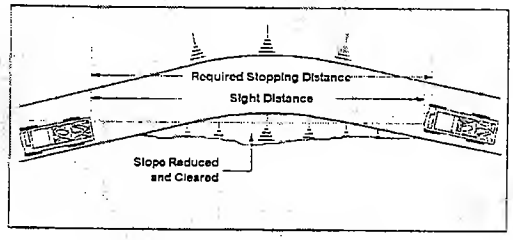


Figure 4 Horizontal curve Line of Sight

14 Roads.ppt KZ/KZ - 06.05.00 HOLDERBANK

HAUL ROADS

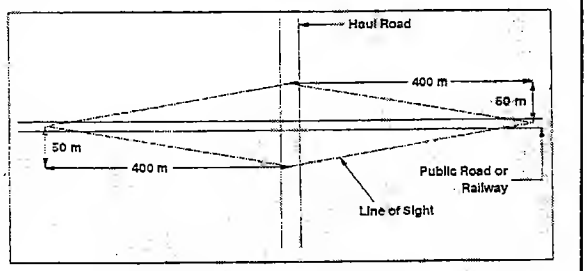


Figure 5 Line of Sight at a Level Crossing

15 Roads.ppt KZ/KZ - 06.05.00 HOLDERBANK

HAUL ROADS

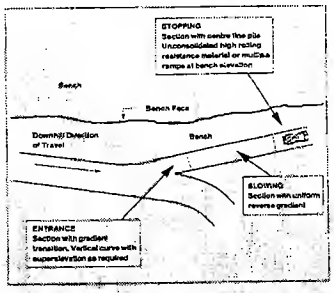


Figure 6 Runaway Precautions

16 Roads.ppt KZ/KZ - 06.05.00 HOLDERBANK

HAUL ROADS

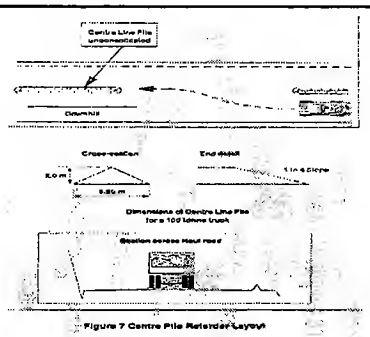


Figure 7 Centre Pile Reminder Layout

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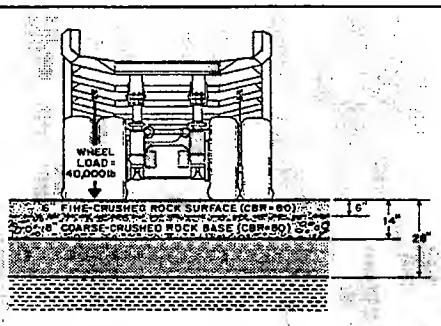
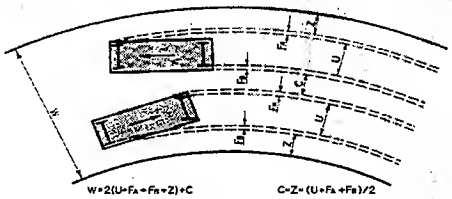
HAUL ROADS



18 Roads.ppt KZ/KZ - 06.05.00 HOLDERBANK

HAUL ROADS



W=2(U+FA+FB+2)+C C=Z=(U+FA+FB)/2

U = Track width of vehicle (center-to-center tires), feet
FA = Width of front overhang, feet
FB = Width of rear overhang, feet
C = Total lateral clearance
Z = Extra width allowance due to difficulty of driving on curves, feet

19 Roads.ppt KZ/KZ - 08.05.00

HAUL ROADS

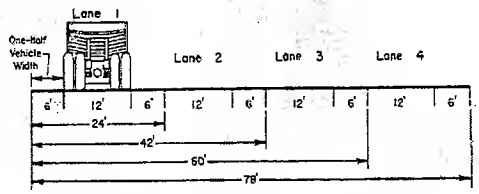


FIGURE 21. - Typical haulageway sections for 12-foot-wide vehicle.

20 Roads.ppt KZ/KZ - 08.05.00

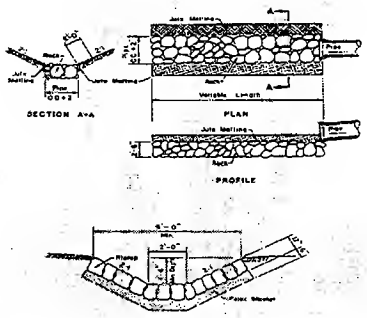
HAUL ROADS

TYPICAL CUT SECTION

TYPICAL FILL SECTION

21 Roads.ppt KZ/KZ - 08.05.00

HAUL ROADS



22 Roads.ppt KZ/KZ - 08.05.00

HAUL ROADS

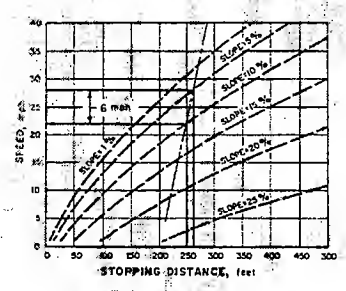
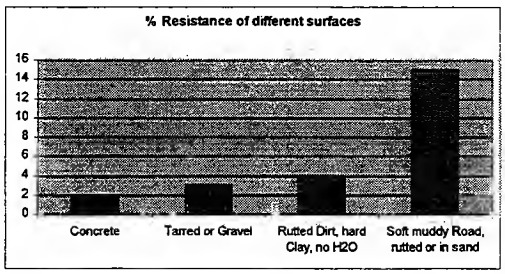


FIGURE 7. - Effect of grade reduction on stopping distances.

23 Roads.ppt KZ/KZ - 08.05.00

HAUL ROADS

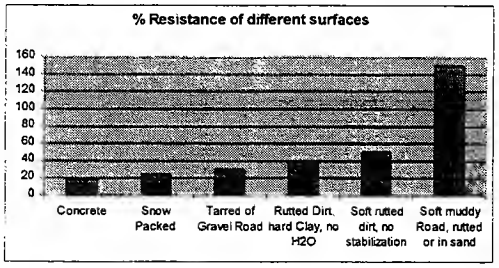
■ Rolling resistance



24 Roads.ppt KZ/KZ - 08.05.00

HAUL ROADS

■ Rolling resistance



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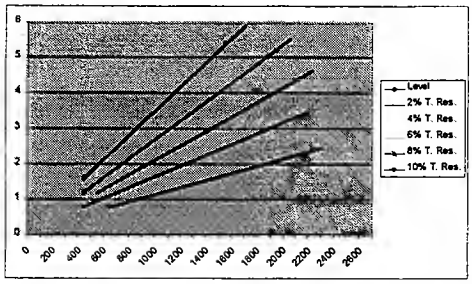
HAUL ROADS

■ Rolling resistance and truck travelling times



26 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

■ Costs of "Rolling Resistance"

➢An increase from 3 to 11% means:

✓ 50 % Fuel increase

✓ decreased productivity per truck

➪ therefore more truck time is necessary

∑ Double Fuel Bill !!!!!!!!!!!!!!!!

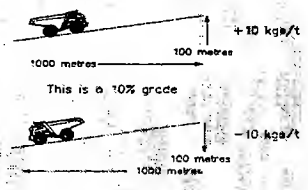
27 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

28 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

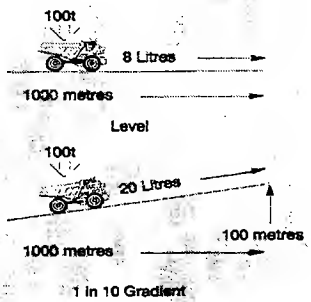
1% Grade produces a hindering or helping force of 10 kgs/t of vehicle weight



This is also a 10% grade

29 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS



1 in 10 Gradient

30 Roads.ppt KZ/KZ - 08.05.00 HOLDERBANK

HAUL ROADS

- Competition to the Haul Roads:
  - ➢Conveyor Belts
    - Quiet
    - Not affected to the weather
    - Dust contained
    - Labour efficient
    - Spillage minimal
    - Energy efficient
    - Continuous transport system

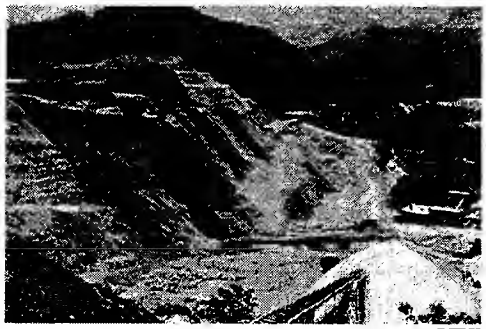
HAUL ROADS

- Competition to the Haul Roads:
  - ➢Conveyor Belts

Maximum size << compared to truck loads
Mobile crusher has to be installed

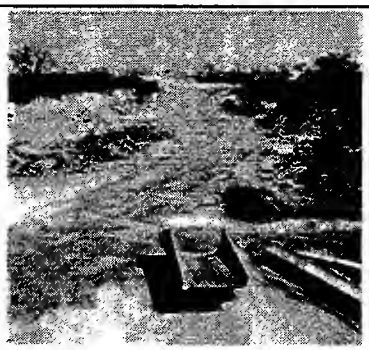
No haul road is necessary

HAUL ROADS



HAUL ROADS

HAUL ROADS



HAUL ROADS

Chapter 2

Crushing

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Crushing

R. Playle
VA 93/4017/E (Revision 2/98)

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1. INTRODUCTION

The technology of crushing is governed by the laws of physics involving mass, velocity, kinetic energy and gravity. Crusher selection, even today, is not a science but is still mainly a matter of personnel experience and testing. Field experience and tests conducted in a laboratory are the best method of indicating the reaction of a material to the various forces associated with reduction. From this the selection of the most economical crushing plant to produce the required product is made.

In general the total number of crushing stages in a comminution plant is determined by the overall reduction ratio required. In the cement industry the abrasiveness of the rock to be crushed is the primary determining factor of the number of reduction stages required. The plant feed originating from a quarry usually contains large rocks and boulders. The size of the first or primary crusher is usually specified by the total capacity required or maximum lump size of the feed. For hard abrasive rocks, such as Dolerite, a compression type jaw or gyratory crusher is used for the primary crushing duty. Product size of this type is normally about 80 % passing 1000 mm to 250 mm. Normally feed to a gyratory is not pre-screened but may be necessary when using a jaw crusher to increase throughput. For softer rocks, such as limestone, an impact crusher will give the best results. They are designed for the crushing of non or low abrasive materials and have a high throughput and reduction ratio with a product containing a large percentage of fines. The power consumption of these machines is lower per ton throughput than for an equivalent size jaw or gyratory crusher. Because of their mechanical strength they are also suitable for the crushing of hard materials, but at a very much higher cost than for an equivalent compression crusher. Figure 1 illustrates the various common types of crushers. Reduction ratios vary between 3 to 1 and 7 to 1 for compression crushers and up to 50 to 1 for impact crushers. To obtain the maximum reduction ratios from any crusher is not the most economic way of crushing a material as it results in hang-ups and increased power consumption which leads to excessive costs.

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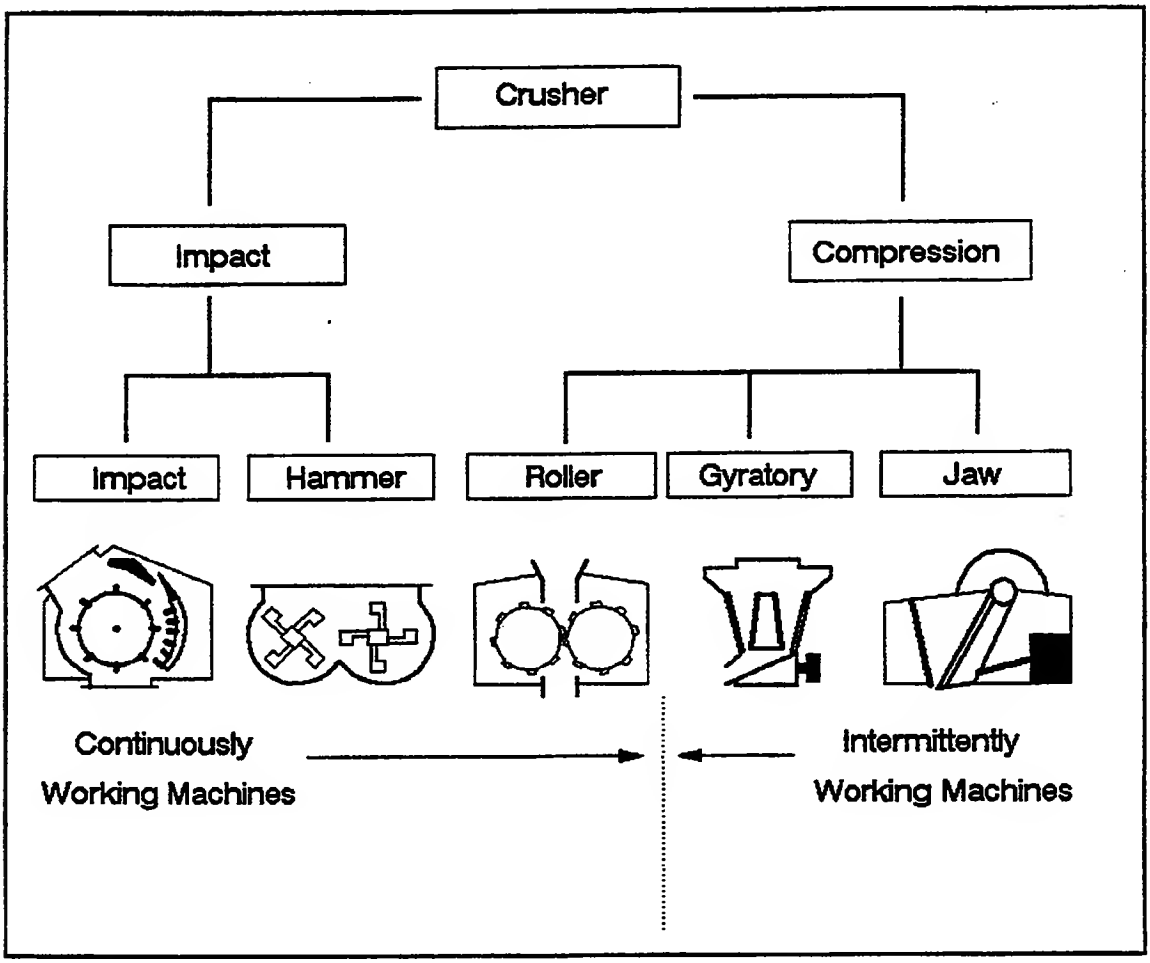
Figure 1 Crusher Types



To achieve the required reduction ratio it is usual to have two or more sets of crushing stages. These are then termed primary, secondary and tertiary stages of crushing. The type of crushers that suit a particular operation depend on several factors. These include:

- The work index of the material
- Abrasiveness
- Stickiness
- Throughput, tons per hour etc.
- Size of feed
- Power consumption
- Reduction Ratios needed

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2. HISTORY

Crushing machines were invented in the early 19th century. The first examples resembled stamp mills and were not particularly effective. It was in 1858 that E.W. Blake invented the jaw crusher using the mechanical principal of the toggle linkage. Today his invention is the standard by which all jaw crushers are measured. For primary crushing of hard abrasive rocks it is still the best machine. Following on in 1860 the gyratory crusher came into being. However only in 1881 was a patent granted to P W Gates for a machine which included most of the features of the modern day gyratory crushers. Initially crushers were small as the tonnage throughput was low due to hand mining being the norm.

With the advent of face shovels which led to a great increase in production rates, the need for bigger crushers became evident. By 1910 gyratory crushers with 1.20 meter openings were in use. The jaw crusher had been replaced as the prime means of crushing by the gyratory but now made a comeback. The single toggle jaw crusher was developed in larger sizes as it operated at closer settings than the double toggle and could therefore achieve a finer product. During this period the roller crusher was born and single roll crushers were installed at limestone plants in the USA. With the need for a finer product, due to the advent of cyanidation in the gold mining industry, the double roll or 'Cornish' rolls, consisting of double smooth-face crushing rolls were widely used. From this developed the toothed roller crusher, used today in the cement industry for the reduction of limestone and clays, particularly with wet sticky feed material.

Secondary crushing, at the turn of the century, was still in its infancy and need for a finer product was achieved by the use of a small gyratory after primary crushing by a large jaw or gyratory crusher. The earliest attempt to adapt the gyratory as a secondary crusher was the 'short head'. This was a reduction of the size of the crushing head in a standard machine. In the 1920's short head gyratory crushers designed specifically for secondary crushing were introduced.

3. PRINCIPLES OF CRUSHING

The main objective of crushing is the transforming and refining of a feed to a predetermined form, usually a limiting size, minimum and maximum, as defined by standards and directives.

3.1 Crushing Stage

Most plants consist of more than one crushing stage. Three crushing stages with different objectives can be defined:

3.1.1 Primary

In this stage the material is prepared for further processing. The machine which performs the function of the primary crusher is, very often, the only crusher at a plant. The term primary crusher denotes the reduction machine which continues the process that blasting has left off.

Product from the primary crushing stage should be as homogeneous as possible with a maximum size of about 150 mm, and a minimum amount of fines. In hard rock quarries two crushing stages are usually required to achieve this result. One crusher if the rock is of a softer nature. Once material has passed through the primary stage it is of a suitable size to be moved by conveyor.

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3.1.2 Intermediate or Secondary

This term applies to the crushing stage which follows immediately after the primary stage, it can consist of either a single or multiple crushers. The feed is all or part of the product of the primary crushing stage. The choice of machine will depend largely on the nature of the feed. Where it is necessary to control the size of the product closed circuit crushing is employed.

3.1.3 Tertiary

If a fine particle size product is required a further reduction in size of material may be needed. To achieve this a tertiary crushing unit would be installed to produce a good quality, regular size sand.

3.2 Feed Size

Normally each manufacturer has a range of different crushing chambers for each size of crusher so that there is a wide choice of feed opening and capacities to choose from. The reduction ratio is where the feed size and the desired product size should be considered together. This ratio makes it easy to calculate the number of crushing stages required in the plant to reduce the material to an acceptable size. Figure 2 compares the feed and discharge areas of jaw and gyratory primary crushers. It is normal to take between 80 and 90 percent of the feed opening's minimum dimension as the largest feed material size that can be easily accommodated by that crusher. In an optimized operation five percent of feed is usually oversize to the crusher and is screened off by a grizzly from where secondary breaking, by mechanical means, reduces it to manageable dimensions.

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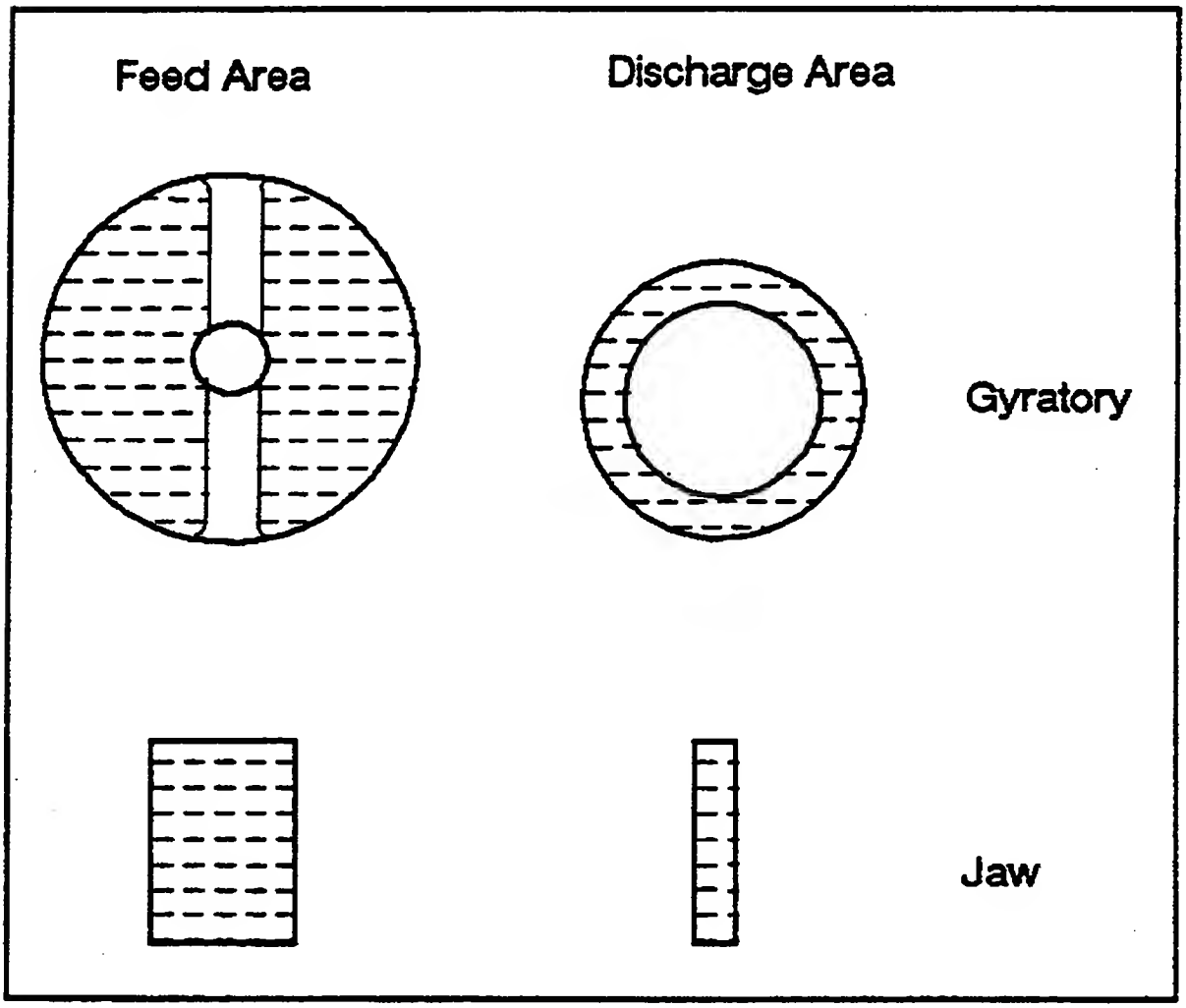
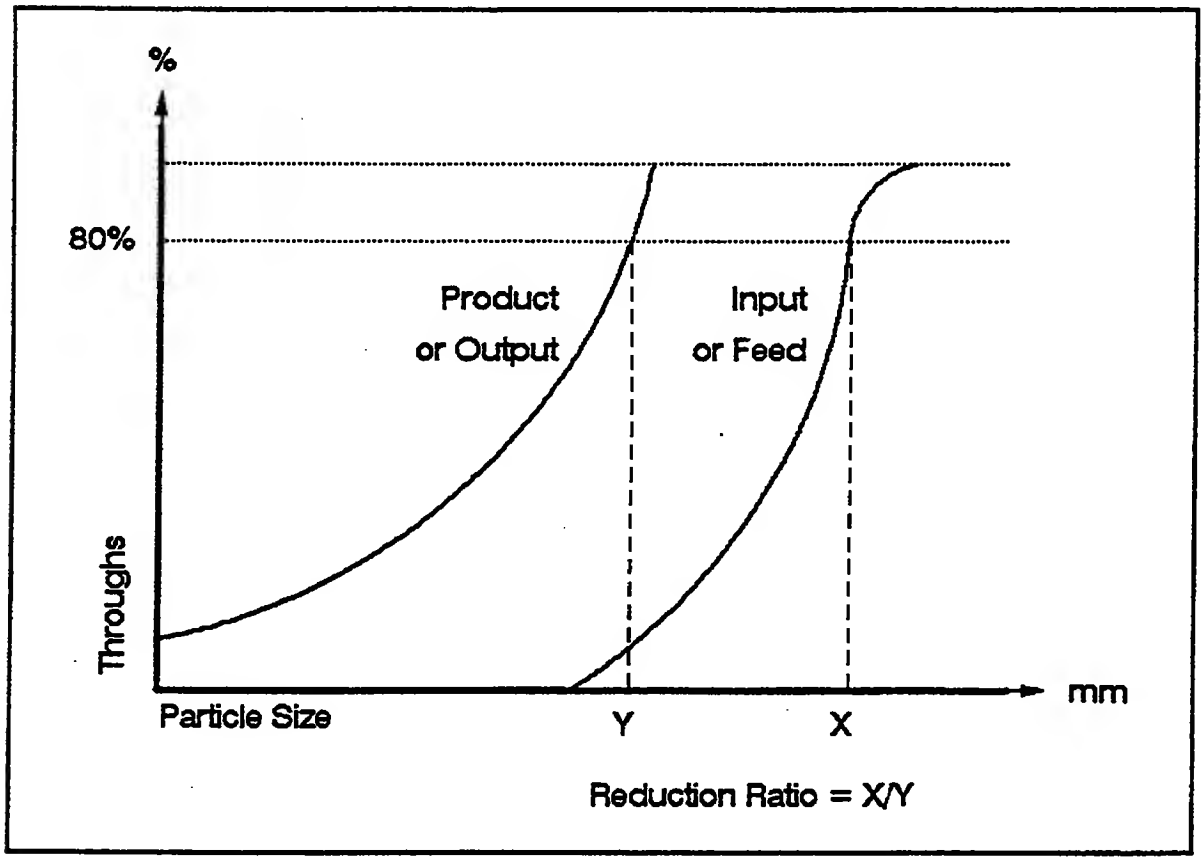
Figure 2 Comparison of Crusher Openings



3.3 Product Size

The reduction ratio, as shown in Figure 3, is based on an 80 % passing figure. That is if 80 % of the feed passes 350 mm and 80 % of the crusher product passes 100 mm then the reduction ratio for that operation is 3.5 to 1. It is dependent on the power rate transmitted to the material but also on its Work Index.

Figure 3 Reduction Ratio



4. TYPES OF CRUSHER

The technical term 'crusher' generally refers to machines comminuting medium to hard material. Size of material ranges from 100 mm to 1500 mm in feed to 5 mm to 300 mm in product. The terminology is by no means definite as the term 'mill' is often used for roller, impact and hammer crushers.

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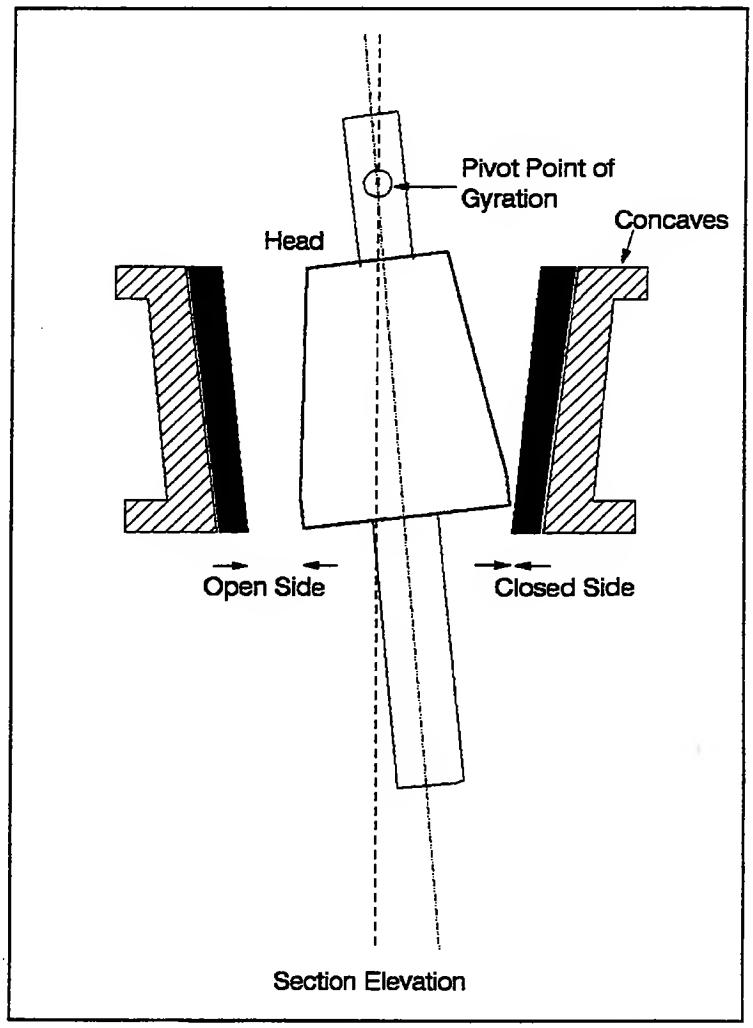
4.1 Gyratory

This type of crusher employs a crushing head in the form of a truncated cone which is mounted on a shaft. The top end is held in a flexible bearing while the bottom end is driven eccentricly so as to describe a circle. The bowl of the machine is a portion of an inverted cone of massive construction and rigidly fixed. The continuous crushing action is caused by the crushing head rotating within the bowl causing the nipping arc to move around the bowl. Figure 4 shows the principle of this type of crusher. It consists of a heavy steel or cast frame which includes in its lower part an actuating mechanism, eccentric and driving gear, and in its upper part a cone shaped crushing chamber lined with wear resistant plates. Spanning the top of the chamber is a steady rest containing a machined journal fixing the main shaft. The crushing member consists of the main shaft and its crushing head or mantle. This assembly is suspended in the journal by means of a large adjustable nut. The lower end of the shaft passes through the eccentric, offsetting the lower end of the shaft. Both crushing members are subject to heavy wear and must be fitted with substantial wearing plates.

Figure 4 Principle of a Gyratory Crusher



Gyratory crushers are capable of crushing very hard rocks with comparatively low maintenance costs but the value of spares required to be kept on hand is much higher than with other types of crusher. Feed problems can arise if the feed is not parallel to the steady rest.

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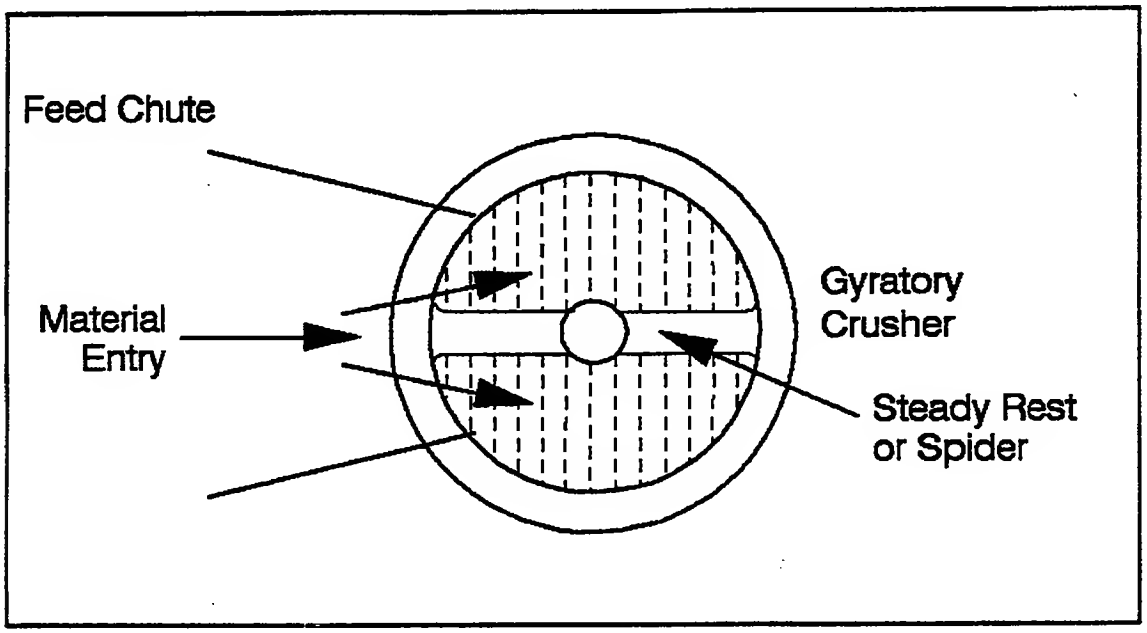
Figure 5 shows the best feed direction. Both sides of the crusher must receive equal amounts of feed to spread the wear caused by the impact of the feed on entry. Some breaking is done by impact of the feed onto the armored cover over the suspension nut but the major method of particle reduction is by compression.

Figure 5 Material Feed Direction to a Gyratory Crusher



4.2 Cone

This type of crusher, very commonly used as a secondary crusher, is usually either of the:

- supported head type
- suspended head type

Cone crushers with supported heads and feed plates on top of the cone were originally developed by Symons in the USA. They consist of an external fixed bowl and a moving internal cone with its axis running at an angle to the machine axis. The displacement of the cone at each stroke is at a speed that permits pieces of rock to fall freely by gravity and be caught further down the rising head on its return stroke. In the case of a machine with a suspended head, short stroke and lower speed, the rocks will slide down the crushing opening using the same principle as the gyratory crusher.

Throughput capacities vary depending on the range of the feed size and the setting of the outlet. They are most suited as a secondary crusher. However as a tertiary crusher with a shallow angled cone the crusher is sensitive to the moisture content of the feed material.

Adjustment of the outlet setting is either by wedge or the fixed crusher member rotated on a threaded ring. The latter method has been refined by the use of hydraulic controls so that settings my be altered without stopping the feed to the crusher. Further advances in computerized automation has led to a more accurate forecast of product size and power consumption.

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At the lower end of the crushing chamber the two faces of the 'liners' are shaped so as to be nearly parallel to each other. Figure 6 sketches the reduction principle. The effect of the parallel section of the liners is to ensure that the larger rocks have at least one dimension equal to, or less than, the setting and that all product will be less than twice this dimension. This setting is referred to as the 'close side setting' (css), see figure 7.

Figure 6 The Symons Principle of Crushing

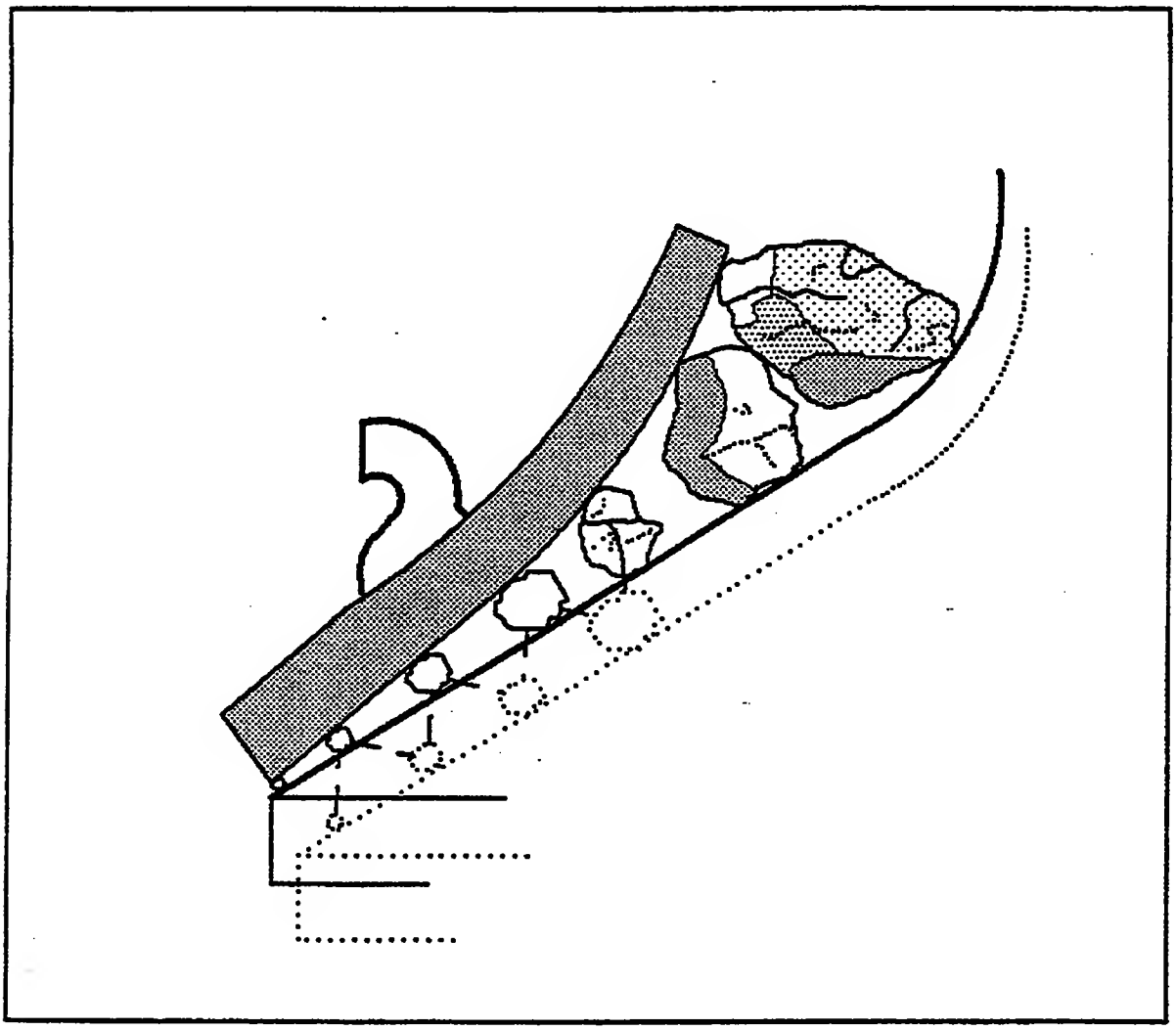

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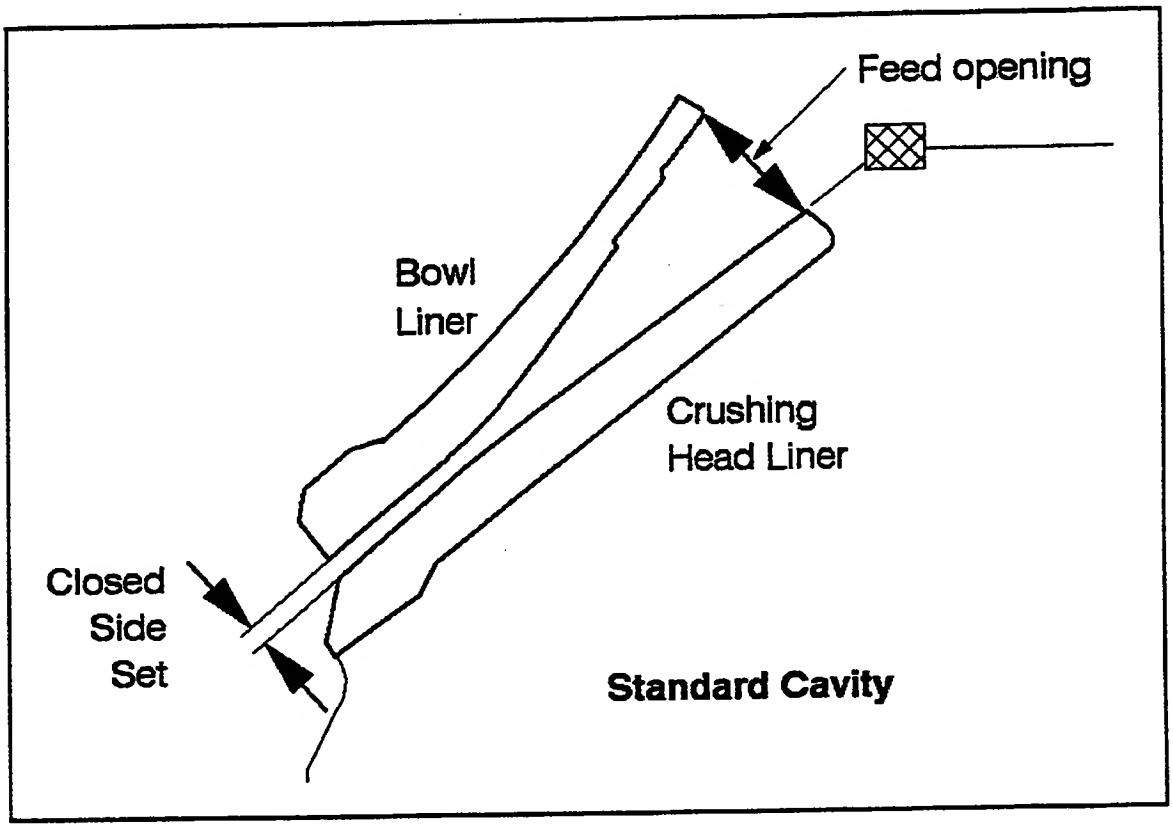
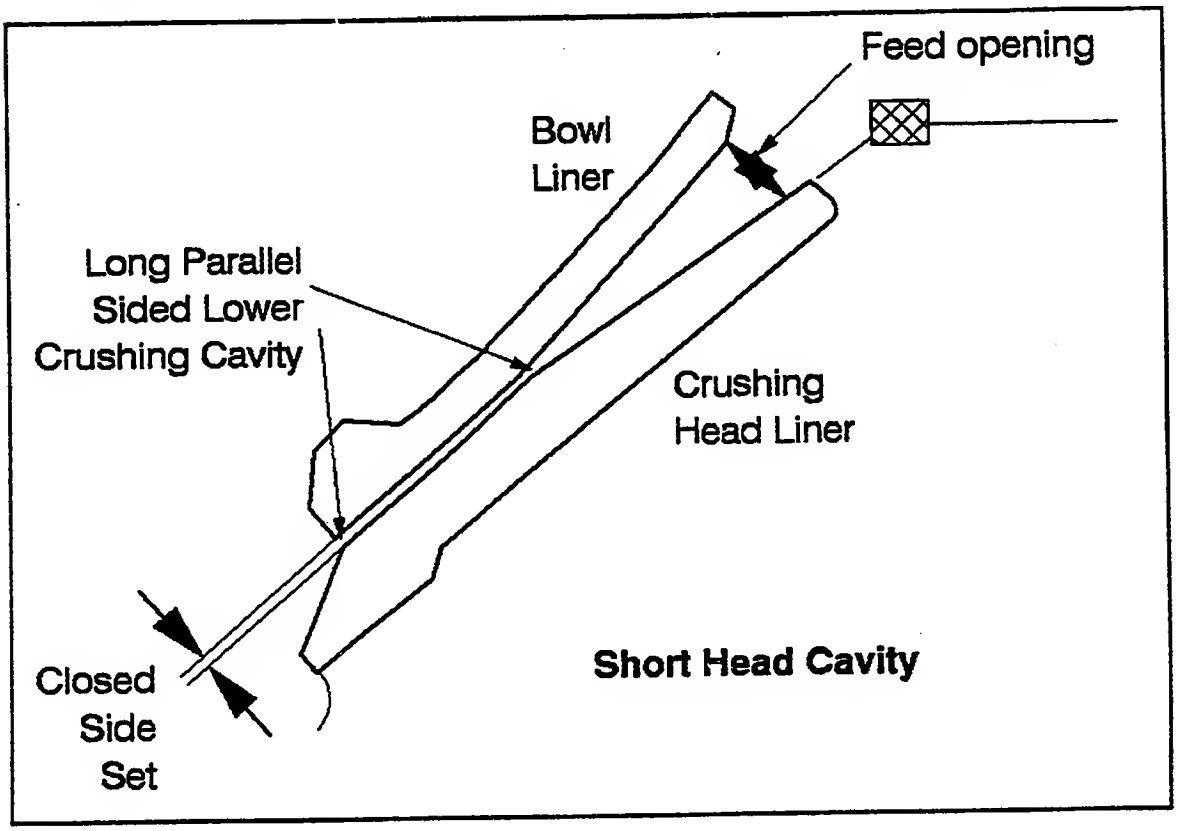
Figure 7 Crushing Cavities





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Reduction ratios of cone crushers are in the range of:

- used as a secondary crusher 6 to 8 : 1
- used as a tertiary crusher 2 to 3 : 1

Both crushing members are subjected to heavy abrasive action and must be fitted with substantial liners, usually made of manganese steel. As these types of crushers achieve the best results if the whole crushing cavity is utilized then they should be choke fed. There are a number of different liners profiles available to suit different material characteristics and to select the best it is advisable to consult the supplier. A wide angle between the crushing liners will increase the reduction ratio and the capacity. A narrow angle has the opposite effect.

A protection method installed on many cone crushers is a series of springs or hydraulic rams which allow the fixed cone jacket to rise up in the event of uncrushable material entering the chamber. The simple operation and low level of maintenance required make for very economical use.

4.3 Jaw

Rugged machines of simple design and easy to maintain. They consist of a fixed and a moveable jaw with massive side frames forming a wedge shaped crushing chamber. The cross section between the crushing faces decreases towards the discharge end, the side frames remaining a constant distance apart. Lining plates can have vertical serrations running their length. These assist with size reduction and moving material through the crusher. The reduction method in jaw crushers is compression. There are two groups of jaw crushers single and double toggle machines. Sections of both types are shown in figures 8 and 9.

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Figure 8 Single Toggle Jaw Crusher

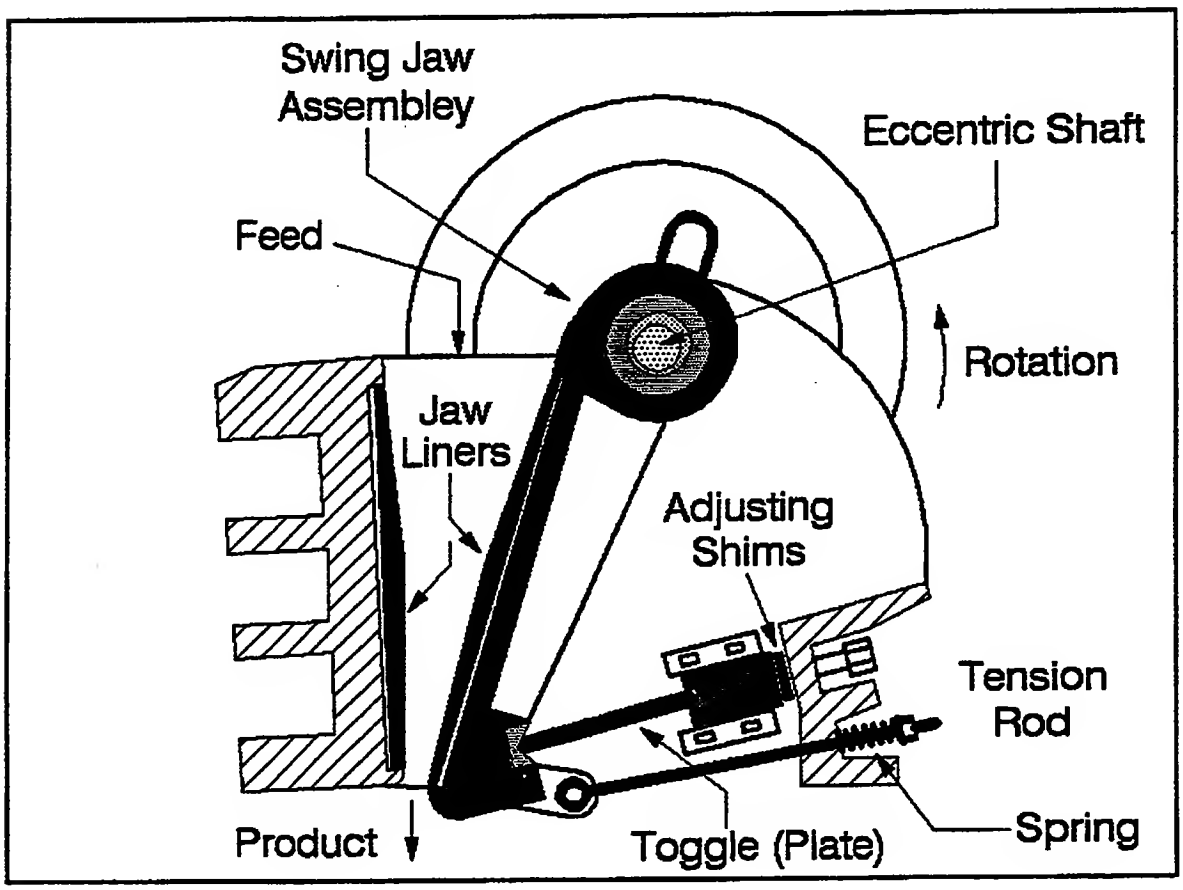
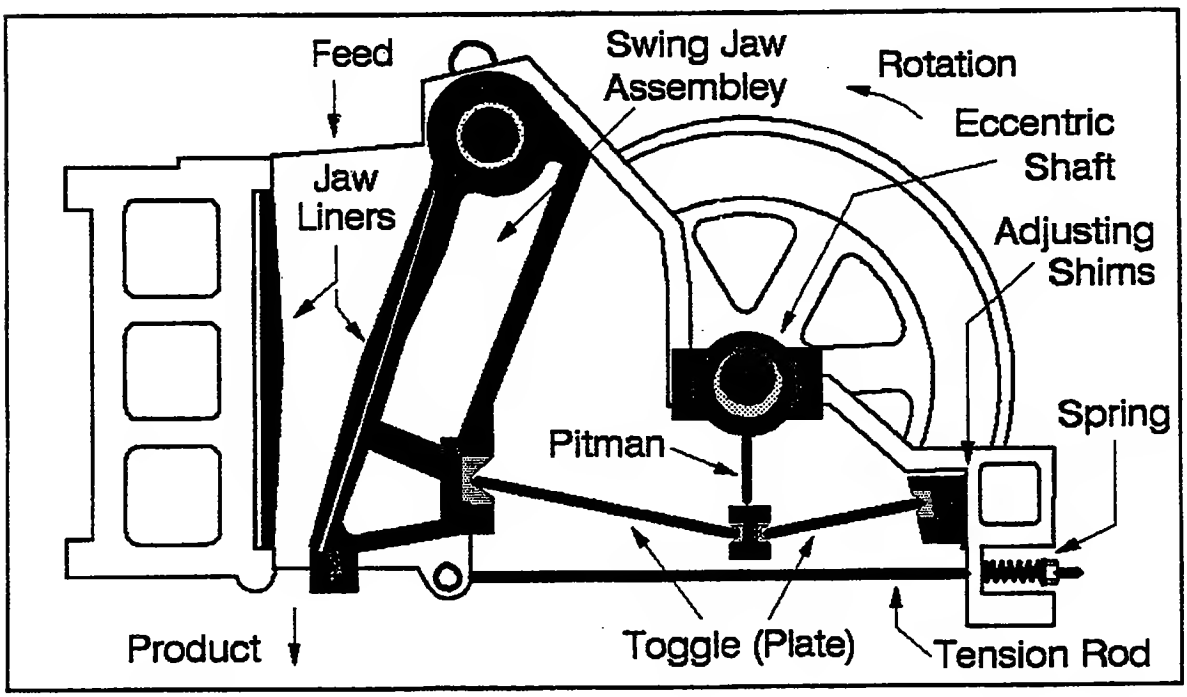


Figure 9 Double Toggle Jaw Crusher



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4.3.1 Single Toggle

The moveable crushing member is hung from an eccentric shaft the lower end being held in position by one toggle plate. The rear end of the toggle rests in a groove on the rigid back of the frame and the other end in the similar groove on the back of the swing jaw. The motion is the result of the circular action of the eccentric shaft at the top of the swing jaw combined with the rocker action of the toggle plate at the bottom of the jaw.

When the swing jaw rises it is opening at the top in the first half of the stroke and closing during the second half. The bottom of the jaw is closing during the whole up stroke. Opposite motion of the swing jaw takes place during the downstroke.

4.3.2 Double Toggle

The moving jaw is freely suspended from a shaft at its top edge and the jaw movement is transmitted through toggles near the lower edge. Maximum crushing force is applied at the inlet to the crushing chamber where the largest particles are found.

The moving jaw is suspended from a cross shaft at its upper end, this shaft in turn being supported at each end in bearings at the top of the two sides of the crusher frame. The actuating mechanism consists of the eccentric shaft, supported in bearings, the pitman and the pair of toggles. The motion of the eccentric shaft is transmitted through the pitman to the inner ends of the toggle pair. The setting of the toggle plates alternatively increases and decreases the effective length of the toggles, imparting the horizontal crushing motion to the jaw. The motion is greatest at the discharge opening.

In both machines the toggles are clamped in position by spring loaded bolts attached to the moving jaw, taking up the motion of the jaw. The crushing action is intermittent allowing the rock to fall further down the crushing chamber before being crushed again. This action can cause choking if overfeeding takes place and it is best to control the feed to this type of crusher.

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The single toggle crusher will not take such hard rock as the double toggle unit. With a rock of medium hardness the single unit will have a larger throughput, due to the travel of the jaw. Where the rock to be crushed is highly laminated the jaw crusher tends to produce slabs. This can make handling of feed material to secondary crushers difficult.

All crushing in jaw crushers is done by compression.

4.4 Roller

A roller crusher consists of a heavy cast frame supporting one or two rolls. Each roll comprises a shaft, a roll center made of cast steel and a shell. This shell can either be smooth or corrugated or have teeth of a wear resistant metal inserted into slots on the shell. Diameters of rolls can vary from 350 mm to 2.0 m and in width up to 2.0 m. Rolls are independently driven by a flywheel type multiple V-belt sheave. One set of bearings is made to slide horizontally on the frames. These moveable bearings are spring loaded to provide a safety relief for excessive pressures. Springs are pre-set to a working pressure for which the machine is designed. The bearings are drawn up against locating shims to establish the gap between, either, a roll and breaking plate or two rolls.

Speeds of roller crushers vary according to the material being crushed. Medium rotor speeds, a tip speed of 8.0 to 10.0 m/s, are used with materials that are difficult to comminute. For materials of a plastic consistency a lower rotor speed preferred, 4.0 to 8.0 m/s. Clay crushers, no longer in general use, had a tip speed of only 1 to 2 meters per second.

Metal used for the teeth can be manganese steel or a boron alloyed carbon steel as they are subject to high wear rates. Teeth have to be profiled by means of special welding materials at intervals depending on the abrasiveness of the material and how the maintenance is carried out. The interval can be six weeks or six months.

Number and size of teeth on toothed roller crushers depend on the crushing requirements. A primary roller crusher will have larger teeth with fewer per row and less rows than a secondary crusher. Teeth of different heights are often used on primary crushers. The higher teeth are known as 'slugger' teeth and the shorter ones as 'regular' teeth. The slugger teeth are designed to grip the larger rocks dragging them between the crushing faces where they are sheared by the regular teeth.

The crushing action of slugger teeth is termed sledging. Sledging is the breaking of material as a sharp stroke is delivered to a particle while it is prevented from moving by being in contact with an opposing surface either fixed or moving.

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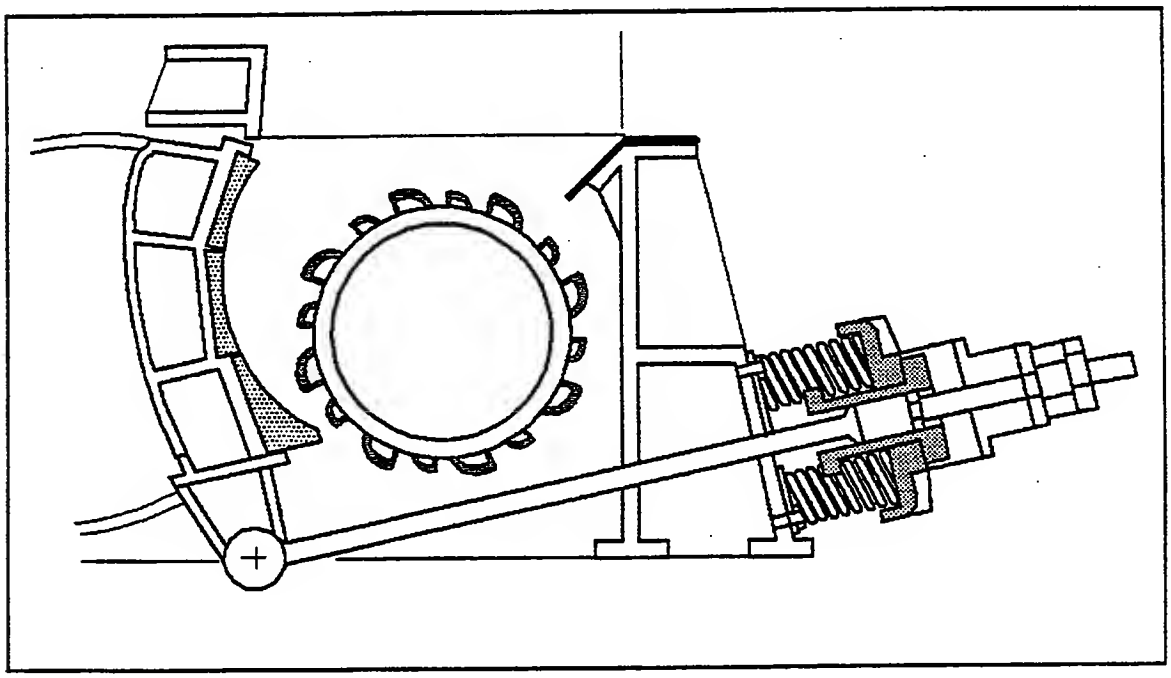
4.4.1 Single Roll

The crushing is done between the roll and the back plate or anvils which are pivoted at the top edge and adjustable at the point of discharge. The adjusting bolts can be spring loaded if the roll is fixed so as to permit the passing of tramp metal, see figure 10.

Figure 10 Single Roll Crusher



It is an effective breaker of soft to medium hard limestones and shales, but is not suited to blocky feeds unless of a friable nature. During the crushing process material will ride on the roll and there will be considerable slip between the material and the top of the teeth. This will lead to excessive wear if the material has other than a low percentage silica.

The machine will turn out a product with less percentage fines than either the jaw or gyratory crushers. Size of product is governed by the distance between the roll and the tip of the lower end of the anvil.

This machine is essentially a primary crusher and works most efficiently when set for a medium or coarse product.

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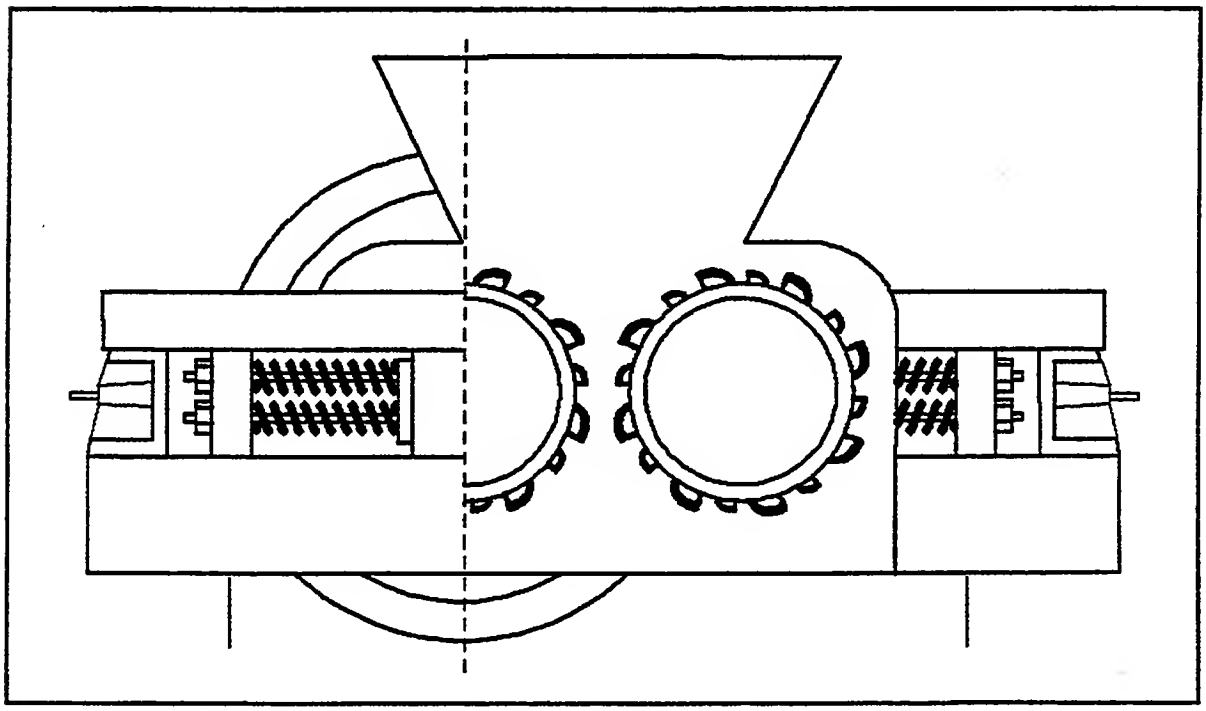
4.4.2 Double Roll

The crushing action takes place between the two rolls, figure 11. On large machines both rolls can be mounted on slides which can be adjusted to give the correct size product. In special cases these machines can be supplied with rotors running at different circumferential speeds. The shearing force due to this increases the comminuting effect. Crushers with medium rotor speeds have proved suitable for dealing with materials that are difficult to comminute, having compressive strengths up to 1500 kg/cm^2.

Figure 11 Double Roll Crusher



These crushers are successful in the reduction of moist sticky materials. They are especially suitable for crushing limestone, marl, clay and shale. Roller crushers are nearly always the first choice when there is a need to reduce wet and very sticky or plastic materials.

The term 'Crushing Rolls' applies to the double roll machine with smooth or corrugated shells. Crushing is entirely by compression of material between the surfaces of the roll faces. Although crushing rolls fitted with corrugated shells have been used in special applications they remain, essentially, a fine reduction crusher used in the secondary or tertiary reduction stage.

One disadvantage of the double roll is that one set of rollers delivers product to the conveyor in the opposite direction to the belt travel. A curved chute can be fitted to reduce the effect but if the feed material is sticky or contains a percentage of clay then a build up of material in the chute can occur leading to clogging. The sticky material can also form boulders in the dead space between the rollers which 'ride' on them and become such a size that when eventually discharged cause damage to the conveyor belts.

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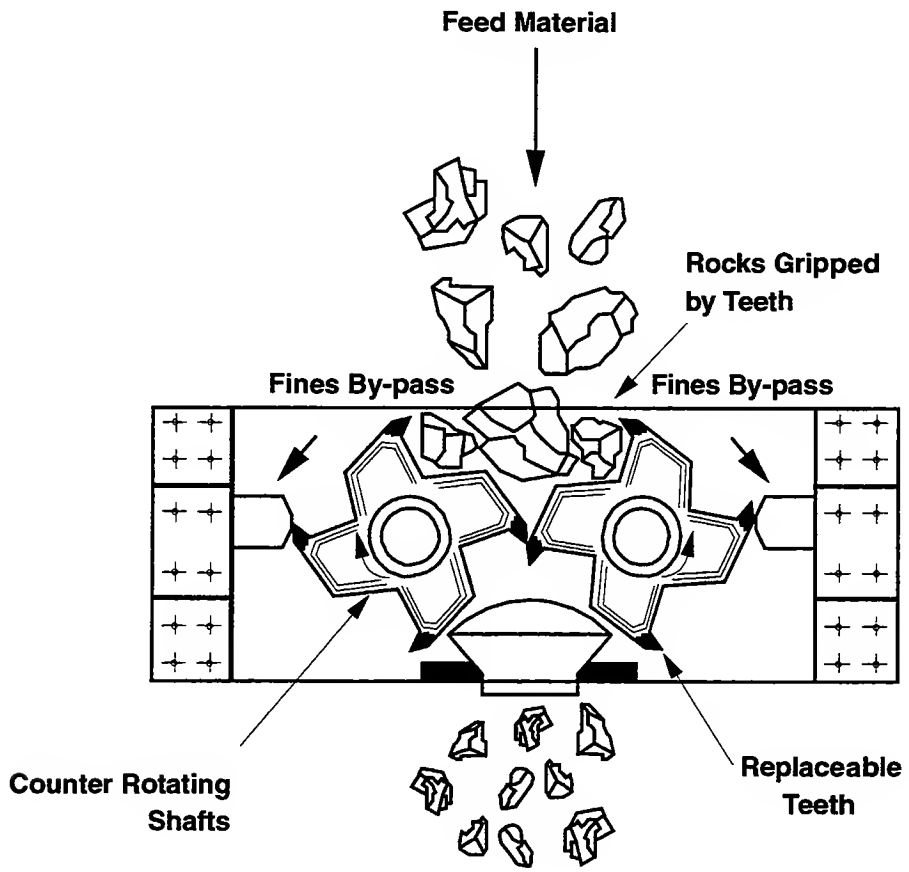
4.4.3 Roll Size

In general the roll size of a crusher is governed by the feed size, crushing strength and friability of the feed rock. In choosing the appropriate crusher size the aim should be to have a one to one ratio between the maximum feed size and the roll diameter. Crusher throughput and product particle size are mainly governed by the distance between the rolls and their width.

4.4.4 Mineral Sizers

The twin roll mineral sizer is a development of the on-belt coal crushers and Stammler sizers. The unit consists of two counter rotating shafts mounted in a frame. On the shafts are mounted several rows of sizing teeth which draw the rock down into the crusher. The shafts are driven by an electric or hydraulic motor through a gearbox and fluid couplings. For large capacity crushers, up to about 8'00t/h, there is a motor to drive each shaft. The units are of modular construction which allows each machine to be made up to the purchaser's requirements. The number of rings and the number of teeth per ring can be varied to suit. Detailed specifications of each machine are decided upon by its specific duty. Figure 12 sketches a cross sectional view of this type of crusher.

Figure 12 Section through the shafts of an MMD Mineral Sizer



A wide range of raw materials required for the making of cement are suitable to be crushed in a sizer. These crushers perform well with very sticky clay and also marl with a high moisture content and are to a large extent self-cleaning. Such a machine will also crush medium hard limestone having brittle characteristics successfully. Material with a high abrasive index, however, will accelerate wear on the pick heads and can result in them being replaced at frequent intervals. The machine is slow running and with few moving parts which minimises wear and replacement of parts. If a hard rock or piece of tramp iron enters the crusher the rotating shafts will stall when the teeth grip it rather than force such material through the crusher. The shafts are then counter rotated and the offending piece can be removed. Similarly if bridging takes place above the shafts they can be counter rotated until the teeth grip and the material is dislodged. Units have been installed at an angle to assist very hard material to run off over the shafts rather than be crushed and damaging the shafts or picks.

The sizer accepts material in through the top of the unit. The two inward rotating shafts with their rows of teeth draw the oversize down into the crusher while permitting the controlled passage of undersize without further reduction in size. This eliminates the need for a scalper in front of such a primary crusher. Material is broken in a sizer principally by tensile stress and shear. This reduces the quantity of fines produced by the crushing action. All breaking forces are contained in the machine. This has the effect of reducing the foundations necessary to support the crusher. It is very well suited to mobile mounting on crawlers, skids or even wheels.

These crushers are compact the space and particularly the vertical dimensions are by far smaller than for the equivalent capacity jaw or hammer crusher. Their vertical dimension is small enough that they have been installed at transfer points between conveyors. It does not require a feeder; material can be loaded directly into the crusher chamber. However wear rates and the incident of broken teeth can be considerably increased by this action and it is not to be considered "good practice".

The sizer does not have a large reduction ratio and in projects where such a crusher has been considered then thought has also to be given to having a secondary unit. Due to its small size the space taken by two such crushers is minimal; especially when compared to other types of crusher. It does, however, make the cost of the crushing system equal or higher to the more conventional single stage systems that are in use at many of our plants.

4.4.5 Reduction

In these crushers three methods of reduction occur. Entering the crusher chamber the material is impacted by the teeth of the revolving roll. As the turning roll compresses the material against the breaker plates or other roll the teeth shear the material. Finally as the teeth of the roll come together the rock is compressed.

4.5 Impact Crushers

Both swing hammer and fixed breaker bar machines may be classified under this heading. A characteristic of these breakers is the large quantity of fines produced and the high reduction ratios obtainable, up to 30 to 1. Run of quarry material can be reduced in one pass of the crusher and the product will be mill feed size. A percentage of slab oversize may, however, be present in the product which can be reduced by passing it in closed circuit back to the crusher.

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4.5.1 Single Roller Hammer Crusher

This machine consists of a base frame and drum with a set of breaker plates or flanges mounted in one half, a horizontal rotating shaft on which the hammers are mounted, and a set of breaker bars or grates situated around the lower circumference of the frame.

The rotor consists of a shaft carried in bearings on either side of the housing at the centre of the multi-flanged drum. The flanges are drilled near their outer edges for hinge pins to which the hammers can be attached. The speed or the rotor is such as to keep the hammers in position by centrifugal force. Rock entering the crusher is struck by the hammers and cast against the breaking plates. This action can be repeated several times in one pass through the crusher. Figure 13 shows the operation of a hammer crusher.

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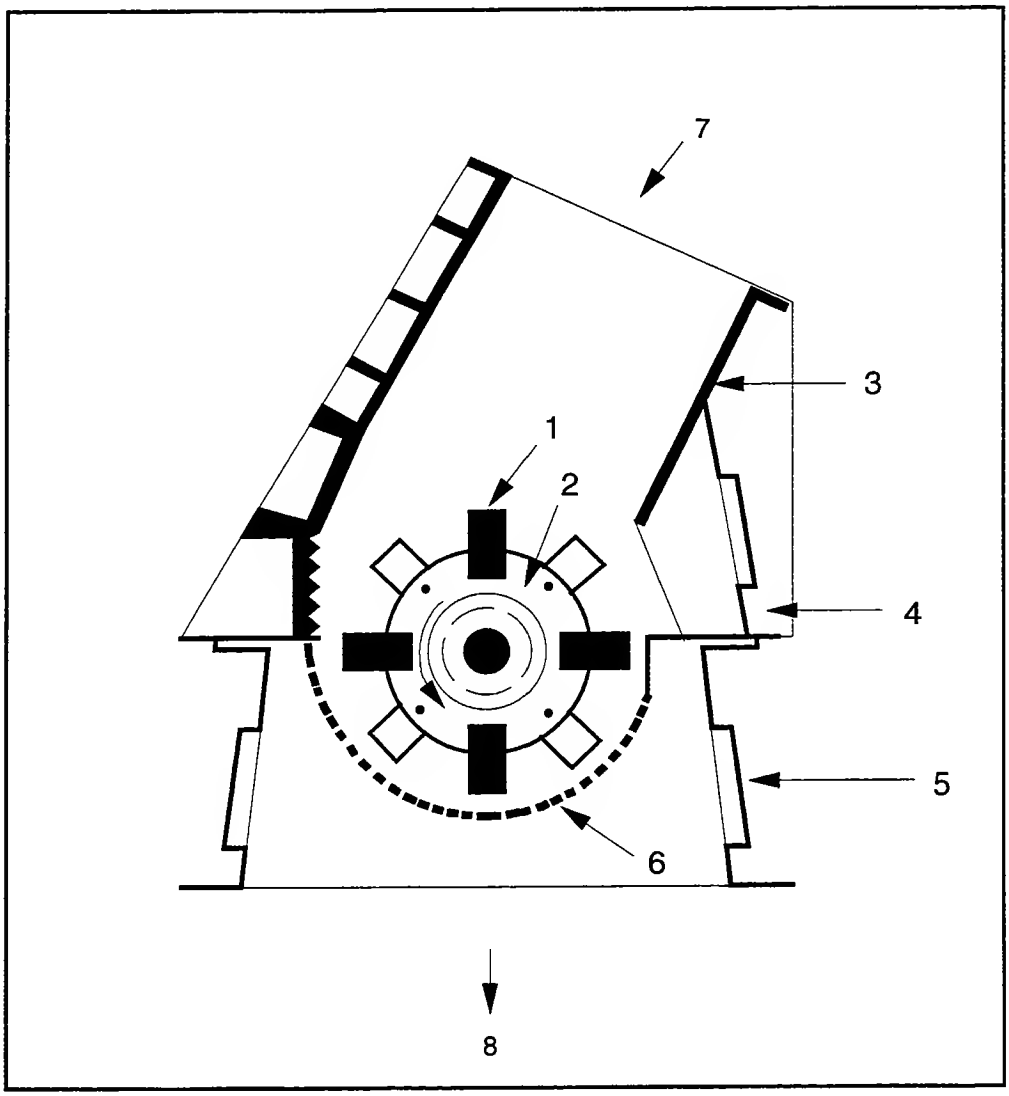
Figure 13 Single Shaft Hammer Crusher with Grates



1 Hammers	5 Lower Casing
2 Rotor	6 Grates
3 Feed Plate	7 Feed
4 Upper Casing	8 Product

A series of transversely arranged, tapered, wear resisting steel bars form a circular cross section just below the hammers path. The spacing of these bars are varied to suit the required product size. When a sticky feed is crushed in this type of breaker the grates soon become blocked and the crusher can eventually stall. The problem then arises that the product will leave the crusher at a high velocity and protection of the conveyor belt removing the product may be needed.

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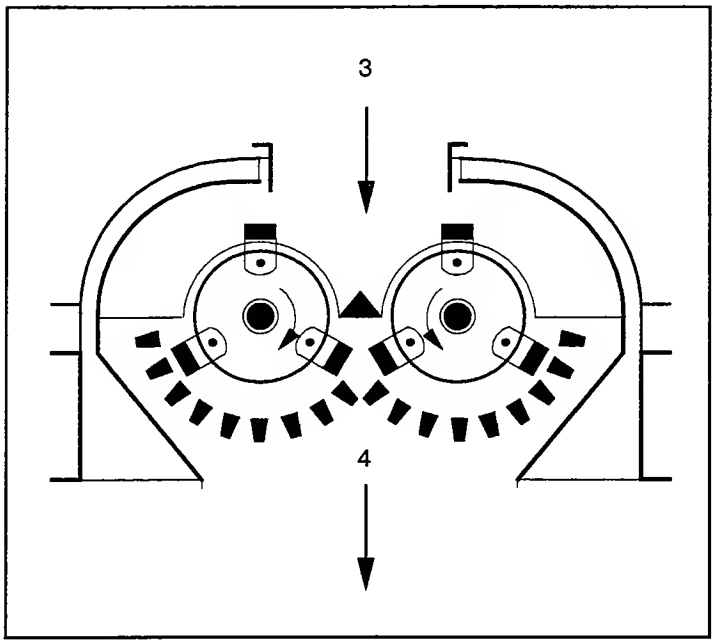
4.5.2 Double Roller Hammer Crusher

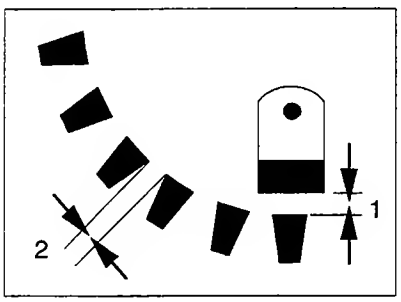
Material is fed, by gravity, into the center of a double rotor machine and is hit by the hammers. This impacting of feed material against two counter rotating sets of hammers is the primary means of size reduction. Other impactions further break the material on an anvil block situated between the rollers. A grid of bars, parallel to the hammer's path can be placed above the rollers. These act as a screen preventing oversize material falling between the rotors. The hammers path is between the grid bars and reduces these blocks so that they can fall into the crusher.

Material is crushed in two ways. First by impact on the hammers and breaker plates. Then by attrition and shear where there is slight clearance between the hammers and the screen bars. This zone is the final sizing area for the product. Some makes of double roller hammer crushers use only the dynamic impacting action of one set of hammers to the other. The advantage of using this method is that in reducing wet sticky materials there is little for it to stick to in the crushing chamber and therefore power consumption will be relatively low. Figure 14 sketches the action of a double roller hammer crusher with a grate.

All the points listed concerning the single roller version apply to the double roller crusher as well.

Figure 14 Double Shaft Hammer Crusher with Grates



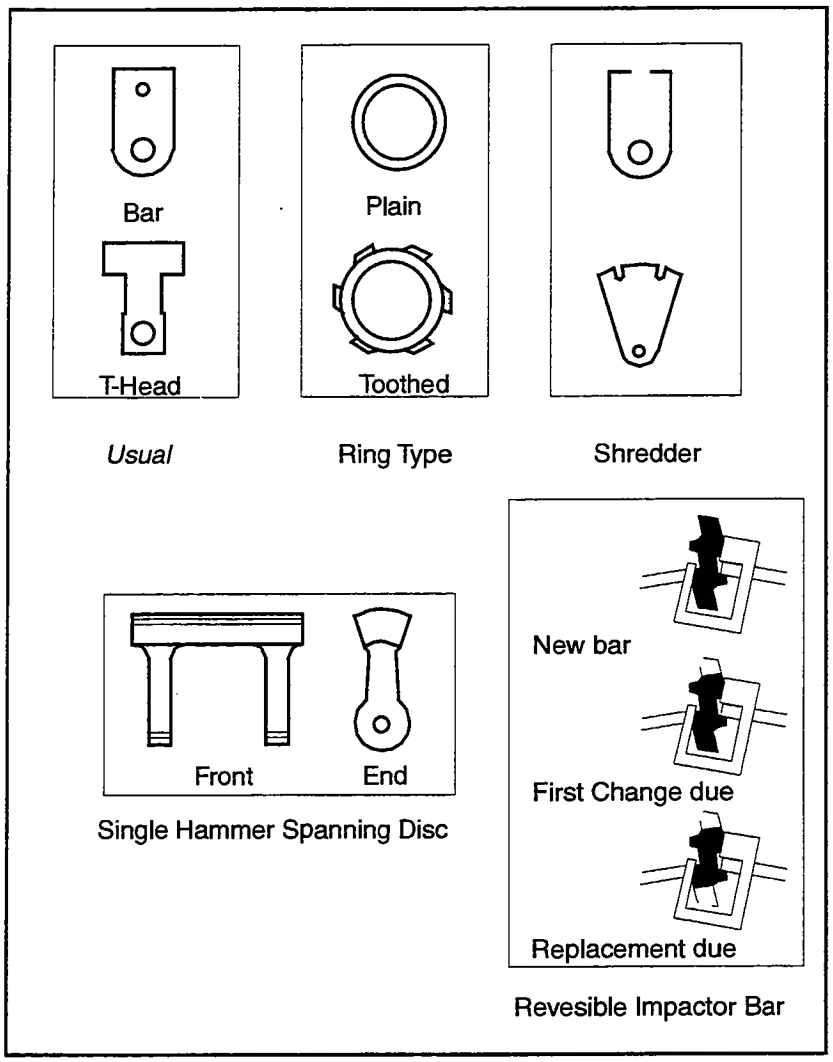


1 Grate Setting
2 Grate Bar Spacing
3 Feed
4 Product

4.5.3 Swing Hammers and Fixed Impact Bars

There are a variety of shapes of hammers, swing for hammer crushers and fixed for impact crushers. Some examples are shown in figure 15. They can be made to span the whole width of the machine hinged on two flanges or be installed individually on separate flanges in a row. There are usually two circles of these flange holes one a smaller radius than the other. When the hammers are new, and their maximum length, the spindles are placed in the inner row of holes. When worn and shortened, they are then transferred to the outer row to restore the gap to the original size. Hammers are normally made out of manganese steel and are reversible to maximize their use before new ones have to be purchased.

Figure 15 Types of Hammers and Bars



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Speed has a definite influence on the reduction ratio of hammer crushers. The effect of a speed of about 60 meters per second on friable material is almost explosive the rock being pulverized. This results in a very high percentage of fines. A low speed crusher can result in a fairly uniform product with low percentage fines providing the feed is not struck too may times in the crushing chamber. The design of the crushing chamber will also effect the fineness of the product. Machines which perform the majority of the work by plain impact action will have a more uniform product size with less fines.

With all hammer crushers a means of controlling the feed rate is necessary and of screening out under size material before it enters the crusher. A constant degree of crushing can be maintained by having moveable breaker plates so that as the hammers wear the plates are moved inwards. A good practice, which helps to reduce costs, is to preserve hammer profile for as long as economically possible.

4.5.4 Single and Double Rotor Bar

These types of crushers have rigid breaker bars extending the full length of the rotor with a combination of fixed bars and moveable breaker plates to permit the passing of uncrushable material. Four or six rows of bars is the norm, spaced equidistantly around the roller. The top ends of these moveable plates are hinged and the lower end suspended on spring loaded adjustable bolts. This will allow the gap to be varied to determine a particular product size. Alternatively breaker plates can be fixed in one position arranged to give the rock particles maximum impact when cast against them.

4.5.5 Reversible Impactors

Feed is dropped into the central feed chute and cast by the hammers against breaker plates or blocks and shatters under the repeated impact. The symmetrical construction of the crusher permits the unit to work equally well with the rotor revolving in either direction. The advantages of this option are that wear can be evenly distributed over both sets of breaker plates and that hammers do not have to be turned during their working life. Figure 16 shows the outline of such a machine.

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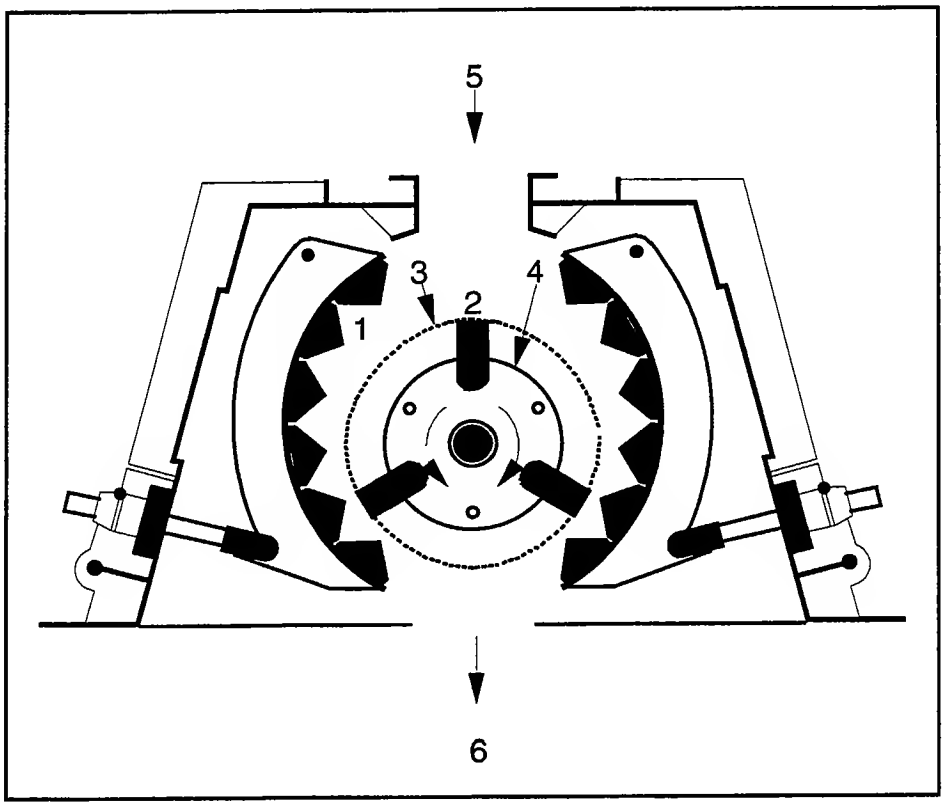
Figure 16 Reversible Single Shaft Hammer Crusher



1 Breaker Block (Impact Area)
2 Hammers
3 Rotor Diameter
4 Rotor Assembley
5 Feed
6 Product

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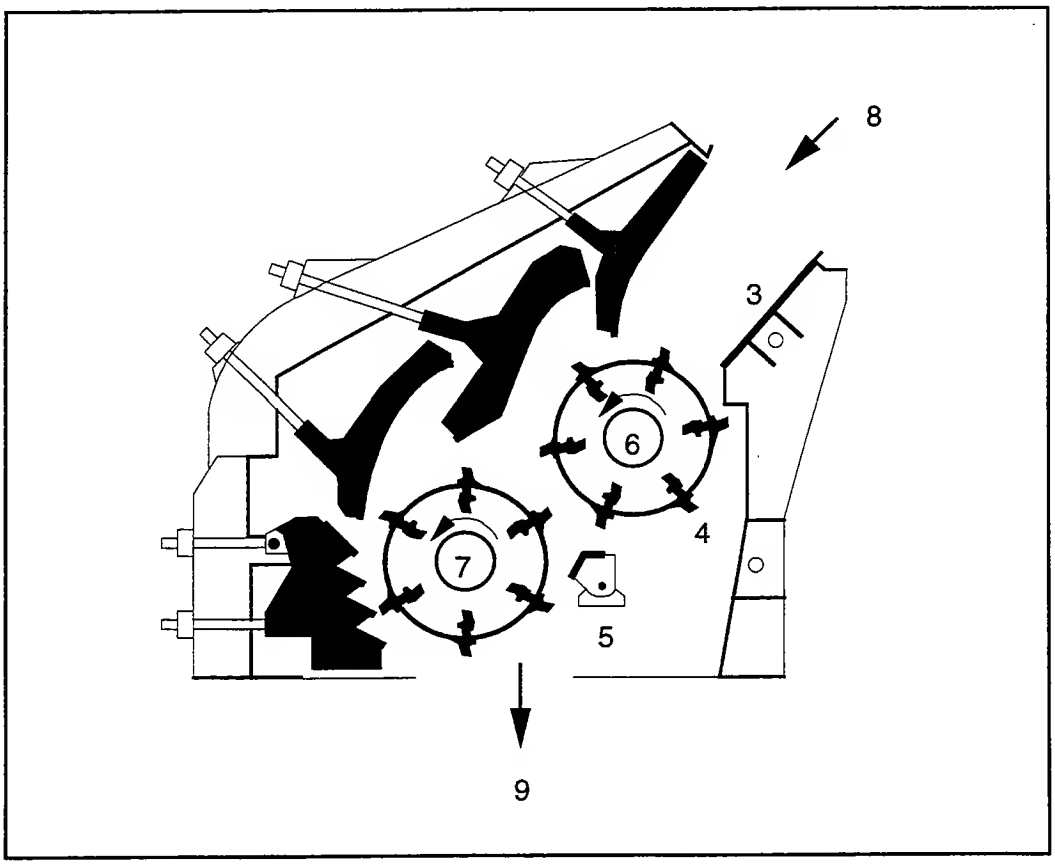
4.5.6 Compound

This design eliminates the need to have several crushing stages by accepting Run of Quarry (RQQ) feed and producing 95 % less than 25 mm. The compound crusher contains in one housing both primary and secondary crushers. Feed enters through the primary inlet into the primary crushing chamber where it is reduced and passed immediately to the secondary crusher. After further reduction the material moves through a grinding path before exiting the housing. The sectional sketch in figure 17 shows the two rotors working in conjunction with three impact aprons and a grinding path in the rear.

Figure 17 Compound Crusher with Grinding Path



1 Impact Plates
2 Grinding Path
3 Feed Plate
4 Impact Bars
5 Anvil
6 First Rotor
7 Second Rotor
8 Feed
9 Product

The compound crusher consists of two rotors turning in the same direction. Each rotor contains three to six breaker bars depending on the hardness of the material to be crushed. The secondary rotor turns at a higher speed than the primary. An average figure is 30 to 33 meters per second for the tip speed of the primary and 37 to 40 for the secondary. A maximum tip speed for the primary crusher of between 40 and 45 meters per second can be realized but this results in increased wear on the breaker bars. The speed and the position of the rotors in relation to each other are the main factors affecting the reduction ratios. The secondary crusher has a lighter weight rotor than the primary and is commonly fitted with breaker bars that are of a softer steel.

A compound crusher will crush a wet sticky feed successfully without clogging the machine as there are no grates in the crusher housing and the striker plates can be heated. Compound crushers have high reduction ratios and throughput. They reduce the need for belts and feeders between primary and secondary crushers. Their main field of application is in the limestone and gypsum industry.

5. MOBILE CRUSHING PLANTS

In a shallow quarry the benches will be advanced over distances of hundreds of meters. If trucks are used to move broken material then provision must be made to increase the fleet in line with the extra distance to ensure that production is maintained. An alternative, with considerable cost savings, is to erect the primary crusher in the quarry and transport the product from it by conveyer belt. To maximize the reductions in the fleet of trucks the crusher can be mobile or semi mobile. It can be advanced with the quarry faces maintaining a fixed tramming distance.

5.1 Mobile

A mobile plant is one that can move itself without assistance from outside forces. There are a variety of moving mechanisms available to suit different applications. Hydraulic walking pads are favored in quarries with reasonably smooth floors. Their advantage is that the can turn on the spot. Crawler tracks can follow a face working over floors with poor load bearing qualities. Pneumatic tyred crushers move quickly, compared to the other mechanisms, and are suited to frequent movement. When in operation all traveling mechanisms, except crawler mounted, have to be freed from load. Table 1 compares the various mechanisms.

Table 1 Comparison of mobile mechanisms see figure 18

Traveling mechanism	Speed (m/h)	Ground pressure (kPa)	Travel	Quarry floor	Cost (% of crusher)
Tracks	400	80	medium	muddy	260
Pads	72	120	short	level	130
Wheels	800	800	frequent	good	190

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Figure 18 Application Criteria for Types of Transportation Systems

Transportation Equipment / Operating Conditions		Rail Mounted Wheels	Raft Type Walking Mechanism	Pneumatic Wheels	Crawler Mounted	Fork-lift Car	Transport Crawler
Load	Low	●	●	●	●	●	●
	Medium	●	●	●	●	●	●
	High	○	●	●	●	●	●
Travel Speed	Low	●	●	●	●	●	●
	Medium	●	○	●	●	●	●
	High	●	○	●	●	●	●
Climbing Capacity	Low	●	●	●	●	●	●
	Medium	○	●	●	●	●	●
	High	○	●	○	●	○	●
Ground Pressure	Short	●	●	○	●	○	●
	Medium	●	●	●	●	●	●
	High	●	●	●	●	●	●
Travel Distance	Short	●	●	●	●	●	●
	Long	●	●	●	●	●	●
Mobility in all directions		○	●	●	●	●	●
Applicable for several machines		○	●	●	○	●	●
Overall Weight	Low	●	●	●	●	●	○
	Medium	●	●	●	●	●	●
	High	●	●	●	●	●	●
Price Relation		1.0	1.3	1.9	2.6	2.2	3.5

● Most Suitable ● Suitable ○ Not Suitable

5.2 Semi-Mobile

These plants have no independent means of movement and either have to be towed or lifted by special transporters. They are selected when a change of location is only necessary at long intervals. If the working face is straight and proceeds in one direction the crusher chassis can be rail mounted to ease movement. This is accomplished by means of a towing vehicle. Wheels are retracted when the crusher is in operation.

6. FACTORS AFFECTING POWER REQUIREMENTS

Although no hard and fast rules can be applied to the determination of power requirements extensive field experience has led to the compiling of some average values for various crushers. There are many factors affecting consumption, the major two are the nature of the raw material being fed and the specifications and features of the crusher. In general terms and as a rough guide for comparison the main factors are:

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6.1 Raw Material

6.1.1 Type of Rock

The difference between the energy required to crush the strongest and the weakest rocks is about 50 %.

6.1.2 Moisture Content

A water saturated feed requires twice the energy to crush it as for a dry feed.

6.1.3 Feed Size

The maximum size of feed material rocks should be regulated so that the crusher feed does not have to be stopped and started because of oversize in the crusher bowl. This will result in overall low rates of feed and higher power consumption.

6.2 Crusher

6.2.1 Reduction Ratio

When crushing to the maximum ratio possible and having a choke or maximum regulated feed the energy consumed is about 30 % higher than when the reduction ratio is 80 to 85 % of maximum.

6.2.2 Rate of Feed

Low rates of feed give high energy consumption values especially when using jaw, gyratory or roller crushers. With jaw crushers lowest energy consumption is achieved when feed rate is between 50 and 75 % of choke feed.

6.2.3 Type of Crusher

Impact breakers have marginally lower power consumption than other types of crushers.

6.2.4 Speed of Crusher

Energy consumption of impact breakers and rolls is less at higher speeds.

6.2.5 Crushing Strength

The power required for crushing is almost directly proportional to the crushing strength of the material.

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6.3 Work Index

This is defined as the power per unit mass required to break an infinite particle size down to 80 % passing 100 microns. Units of measurement are usually kwh per ton. The higher the work index of a material the higher will be the required input work to obtain a given reduction ratio. Table 2 lists the average work indices for materials associated with the cement industry.

Table 2 Average work indices

Material	SG	Work Index
Chert	2.65	26.16
Silica Sand	2.65	16.46
Shale	2.58	16.40
Clinker	3.09	13.49
Limestone	2.69	11.61
Dolomite	2.82	11.31
Clay	2.23	7.10

6.4 Typical Power Requirements

A primary single roll crusher reducing run of quarry medium hard limestone to a 165 mm product will require about 400 watts per ton per hour. When this type of crusher is used to crush medium hard bituminous coal to the same specifications only about 65 watts per ton per hour is needed. Even with the hardest minerals single rolls use about 750 watts per ton per hour of power.

Impact crushers, used as primary crushers, require about 760 watts per ton per hour of the gross load to drive them. This only increases slightly if the rotor speed is very high.

A jaw crusher needs approximately 250 watts per ton per hour for its maximum size reduction. If the ratio of feed to product size is reduced then this will decrease.

6.5 Closed Circuit Crushing

This is a means of controlling product top size by screening and returning oversize to the crusher. To control top size with an open circuit, material must remain in the crushing chamber until it is reduced to the required size. The result is that a proportion of the material is overground and the percentage fines in the product is high. Efficiencies are lowered with a corresponding increase in power costs. In a multi stage crushing plant with the final stage in closed circuit the primary crusher operates at a setting which produces a satisfactory feed size for the secondary so that a balance exists for the work done by each crusher and power consumption is minimized. Figure 19 is a schematic drawing of a typical closed circuit crusher system. The argument against a closed circuit system is that it reduces plant capacity, if the recirculated load gets too large efficiencies are markedly decreased. With both, jaw and gyratory crushers, there will be no increase in the product quantity produced since the increased portion will be offset exactly by the reduced final output. Impact breakers will show an increase as there will be a smaller average feed size.

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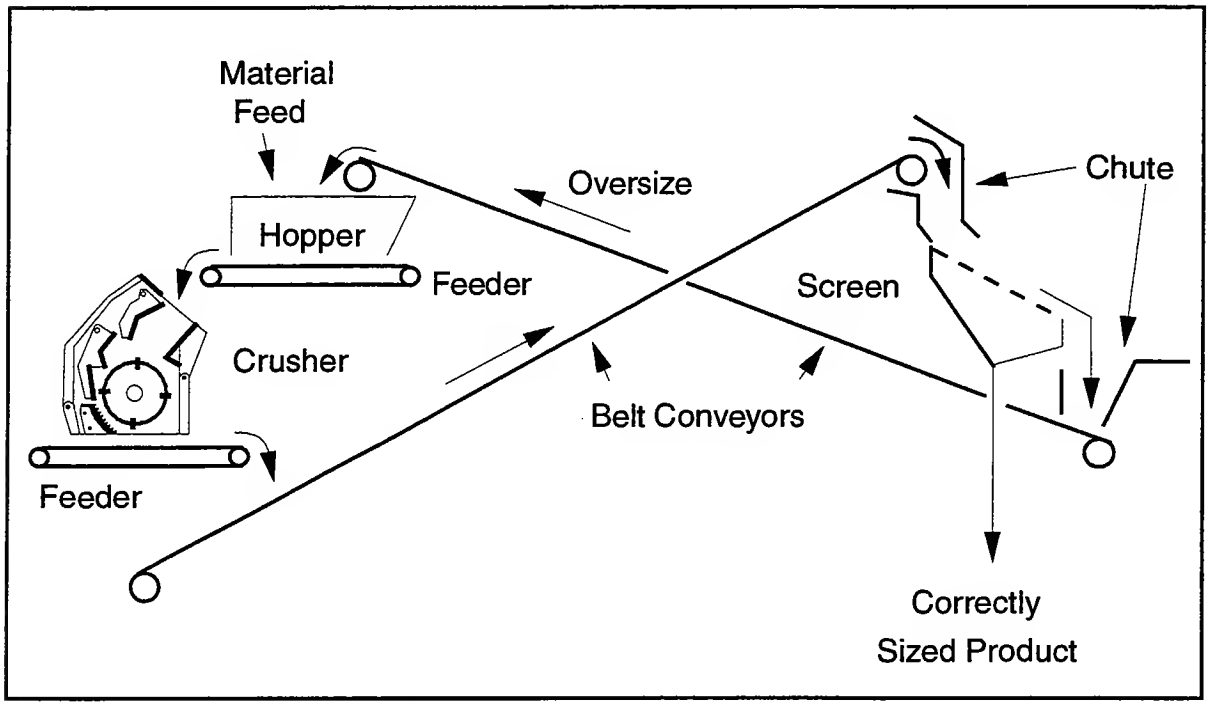
Figure 19 A Closed Circuit System



6.6 Choke or Regulated Feed

A completely filled crusher chamber, or as full as the design will permit? Is there a sufficient head of material above the receiving opening to keep the crusher continuously full? This is particularly important with cone crushers as choke feeding gives higher reduction ratios and reduces wear on the crusher parts. Regulated feed is the throttling of the supply of material to the crusher to keep it below the capacity of the machine.

7. DRYING

In compound and impact crushers the contact plates can be heated to about 120 degrees Celsius. Moisture contained in the sticky material evaporates when it comes into contact with these plates and falls off. The purpose of this devise is not to dry the feed but to prevent build-up of material on the contact plates and reduce the crushers ability to crush.

An alternative method of drying the feed is to blow hot air through the crusher unit as it is working. These machines are capable of drying a feed containing 20 % moisture to a product containing 0.5 % moisture whilst maintaining a high reduction ratio. An example of such a machine is shown on figure 20. The inlet slide plate and impact aprons of crushers can be designed for heating of the material that comes into contact with these surfaces. Feeds with a moisture content of 15 % can be crushed with few problems.

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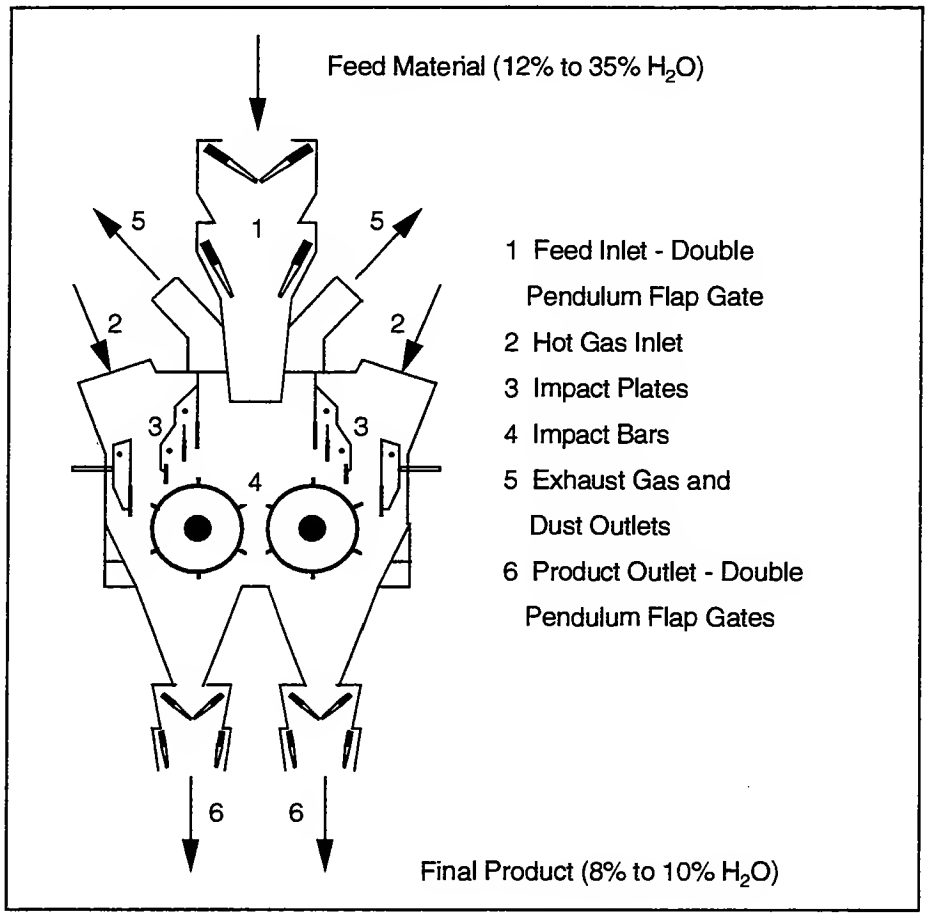
Figure 20 Crusher with Dryer



8. FEED CONTROL

A suitable feeder is required in order to present feed to a crusher to achieve the greatest efficiency. All crushers, with the exception of the gyratory which can be flooded, require constant feed at near maximum levels to achieve this. Where the screening off of undersize is necessary, as in clay and sticky feeds it is possible to use a type of feeder that will remove most of the undersize as well as regulate the feed rate.

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9. HOPPERS

Reception hoppers work under severe conditions, particularly the primary feed hoppers which take run of quarry material. They are subject to severe shocks when trucks or dumpers tip into them loads of 20 to 50 tons from heights of 6.0 meters.

The average hopper has a capacity of about 50 tons and is made of steel plates supported by a frame of Rolled Steel Joists (RSJ). Inside they should be lined with wearer plates or heavy rails laid in the direction of flow. The hopper must retain material at all times so that as much of the wear and shock is taken up by material on material when tipping into it.

10. SELECTING A CRUSHER SYSTEM

Primary crusher systems are usually the biggest and most expensive items of equipment for reducing raw material to an acceptable size. The whole of the crusher house arrangement is designed around the primary crusher and a mistake in the initial choice can be very costly. Major types of crusher and their characteristics are summarized in figure 21 and 22.

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Figure 21 Summary of the Major Types of Crusher

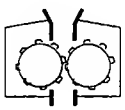
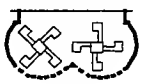
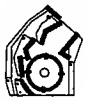
Crusher type		Reduction Ratio	Application	Operation
Jaw Single Toggle		Primary 5 to 1 - 7 to 1	1 st. or 2 ndary. for very hard abrasive material with low moisture. Good for inclusions of sticky material higher wear than 2-toggle	Compression
Double Toggle		2 ndary 3 to 1 - 5 to 1	1 st. or 2 nd. for very hard abrasive material with low moisture. Capacity small.	Compression
Gyratory Gates System		5 to 1 - 7 to 1	Primary - for very hard, abrasive material with low moisture	Compression
Symons System		5 to 1	Secondary - for hard abrasive materials	Compression
Hydrocone		5 to 1	Secondary - for hard abrasive materials with low moisture	Compression
Roller Single		4 to 1 - 6 to 1	Soft materials	Shear, gravity impact & compression
Double		4 to 1 - 6 to 1	1st. choice for sticky materials with inclusions of flints, pebbles etc	Shear, gravity impact & compression
Impact or Hammer Double Hammer		up to 50 to 1	Soft to medium hard not for abrasive or sticky materials H_2O	Impact & attrition
Impact		up to 50 to 1	Soft to medium hard not for abrasive or sticky materials H_2O	Impact
Impact Hammer		up to 50 to 1	Soft to medium hard not for abrasive or sticky materials H_2O	Impact & attrition

Figure 22 Application of Crusher Types

Material Conditions Demands / Crusher Type	Material Classification			Crushing Ratio				Material Abrasivity			Material Moisture		
	soft	medium	hard	1:5	1:15	1:30	1:50	low	med	high	dry	moist	wet
Single Toggle Jaw	●	●	●	●	○	○	○	●	●	○	●	●	○
Double Toggle Jaw	●	●	●	●	○	○	○	●	●	●	●	●	○
Gyratory/Cone	●	●	●	●	○	○	○	●	●	●	●	●	○
Roll	●	●	○	●	○	○	○	●	●	○	●	●	●
Single Shaft Hammer	●	●	●	●	●	●	○	●	●	○	●	●	○
with Grate	●	●	●	●	●	●	●	●	●	○	●	○	○
Double Shaft Hammer	●	●	●	●	●	●	○	●	●	○	●	●	●
with Grate	●	●	●	●	●	●	●	●	●	○	●	●	○
Single Shaft Impact	●	●	●	●	●	●	○	●	●	○	●	●	○
with Grate	●	●	●	●	●	●	●	●	●	○	●	●	○
Double Shaft Impact	●	●	●	●	●	●	●	●	●	○	●	●	○
with Grate	●	●	●	●	●	●	●	●	●	○	●	●	○

● Most Suitable ● Suitable ○ Not Suitable

The following have a bearing on the choice of crusher:

10.1 Characteristics of the Material

This includes the geological classification of the rock, physical structure, chemical analysis and an evaluation of its resistance to crushing. Such information may be obtained from neighboring quarries or from laboratory testing of actual samples. Many deposits of limestone are thinly stratified and can be blasted into fragments that can easily be handled by a medium size crusher, a gyratory or sledge roll.

A jaw crusher may be necessary if the rock formation is massive and tends to be blasted into big blocks bounded by jointing. If the rock to be processed contains high amounts of silica or other abrasive material roll and hammer crushers are not a suitable selection unless the host rock is very soft and friable. Even then they are unlikely to be an economic proposition.

10.2 Capacity

A crusher cannot be selected solely on the basis of capacity and should never be selected to just meet the capacity. In the average quarry operation there is little or no surge capacity. In an operation delay were to occur in loading then the primary crusher will in short order stop through lack of feed. All crushers, primary or secondary, will bridge or block at some stage. The crusher capacity must be high enough to overcome these shortfalls in production. Minimum reserve capacity should be 25 % and up to 50 % in critical cases. Care should be taken when selecting secondary or tertiary crusher systems that their combined

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feed rate will be capable of accepting all the product, requiring further reduction, from the primary.

10.3 Product Size

A primary crusher should give the highest reduction ratio as is economically feasible. The crusher will be set at, or near its minimum setting. This will reduce the throughput and the figures quoted for crusher performance should reflect this. Table 3 lists various types of crusher and the product and feed sizes obtained for different materials. Reduction ratio depends on the tip speed of the crusher, if a rotary for 95 % of the measure of reduction. The higher the tip speed the greater the reduction ratio. However increased speed results in increased wear on the crusher and maintenance costs rise steeply.

Table 3 Examples of reduction ratios

Crusher	Material	Feed Size (max. mm)	Product Size (mm)
Single Rotor Hammer	Limestone	1800	0 - 80
Double Roll	Limestone	1000	0 - 150
Gyratory	Basalt	1500	0 - 230
Double Rotor Hammer	Limestone	1000	0 - 30
Jaw	Iron Ore	850	0 - 130
Impact	Limestone	1500	0 - 300
Impact	Limestone	1500	0 - 150

10.4 Quarry Equipment

Face loading and transporting equipment are all included in this. Usually the only surge capacity in the quarry is that loaded into the trucks. The distance that the material has to be moved as well as the number of trucks available will determine the actual peak capacity needed to maintain throughput and therefore the size of the primary crusher.

All types of crusher work at their best efficiency if continuously fed at a rate close to designed throughput. A short transporting distance with relatively small trucks will approximate this condition. A surge bin ahead of the primary crusher capable of holding three or four trucks should take up any minor delays in feed supply.

10.5 Drilling and Blasting

The size of the drillhole should be suitable for the physical properties of the material. If not then the average rock size to the crusher may be large. This becomes important when a new primary crusher is to be installed on an existing operation and the drilling equipment is not being replaced. Different types of explosives result in different muckpile characteristics and fragmentation analyze. ANFO is a common explosive found in quarries, it is cheap and easy to make on site. When it detonates there is a higher proportion of heave energy released. This results in a coarser muckpile than if an emulsion explosive was used. The number of holes and rows fired in a blast will also effect the muckpile fragmentation. Oversize is generated from the collar of holes and the backbreak from the final row. If large blasts are set off and a hang up occurs then it is possible that the crusher could stand for some considerable time.

10.6 Feeder Design

At most plants there will be an intermediate stockpile between the primary and secondary crusher systems. A majority of quarries will be able to produce the required tonnage on one

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shift. The plant process is continuous and therefore enough material must be available to permit the operation to continue on the other shifts.

11. DAY TO DAY RUNNING

When a crushing system has been selected there are points to be considered with regard to installation and operation. Below is a list of the main ones:

11.1 Operation

11.1.1 Fine Material Scalping and Screening

This is the removal of the fine material that can reduce the efficiency of the crushing process before it passes through the crusher. Good screening performance is especially necessary for fine crushers. The removal of this material, not requiring further crushing, reduces wear on the manganese and the total percentage fines in the product. In addition moisture and contamination in feed material is usually found in the finer fraction. Moisture and clay can cause abnormally high stresses in crushers, by packing, and should be removed before crushed. It is sometimes necessary to have a scalping screen ahead of the primary crusher to remove the undersize portion of a sticky, wet feed. A grizzly feeder, combining both feeding and removal of fines, is one of the more popular methods of accomplishing this.

Screening accuracy depends to a high degree on the material bed depth on the screen deck which is, in turn, dependent on screen width, a wider screen giving better results.

11.1.2 Even Feed to the Primary

An even and regular feed to the primary will increase output. A crusher waiting for material will not perform at its optimum. The situation can be improved by the installation of a hopper in front of the primary crusher capable of taking two to three truck loads of material. Even with this addition if the truck fleet is not reliable throughput rates will suffer.

11.1.3 Surge Storage

Where secondary crushing is necessary a decision has to be taken as to whether the secondary plant is to be operated as an extension of the plant, on a continuous process, or as part of the quarry. Run as part of the plant its size can be reduced but a surge stockpile is required between the primary and the secondary. The alternative means that the stockpile can be within the plant. With a wet feed or climate the stockpile should be under cover. If all crushing stages are direct feed from the previous the stage with the lowest capacity will govern the throughput and all other parts of the plant will operate at below optimum. Providing surge capacity between crushing stages eliminates this problem. The size of the surge stockpile should be such that the operation of the plant is not affected by breakdowns or stoppages further up the line, a truck breakdown or a blockage in the primary crusher, for instance. A stockpile will ensure a regular feed to the next crusher with the benefits of maximum utilization.

11.1.4 Overload Protection

All machines driven by electric motors should have overload protection. In crushers it is either the transmission vee belts or the springs which take the initial shock of an overload. Hydraulic systems have an advantage in that the crusher can open instantaneously if an overload occurs but with a steady return to normal operation without the 'chattering' inherent in the steel spring system.

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11.1.5 Setting Adjustment

Optimum performance is obtained by optimum setting of the crusher action. Adjustment must be made for manganese wear on a regular basis, all the more frequent for abrasive materials. A few millimeters too large a setting can reduce undersize in the product by 10 to 20 percent.

11.1.6 Metal Detectors or Magnets

Occasional tramp metal is found in the material fed to the crusher. It will pass the primary easily but can cause damage to the finer crushing stages. Magnets, sited on the product belt from the primary crusher, remove the metal. They are usually on a small belt which will lift the tramp metal clear and deposit it in a bin. Detectors, on discerning tramp metal present, will stop the belt so that it can be lifted off.

11.1.7 Automation

There can be no doubt that automation of a plant increases throughput. The reaction of an automated crusher to changing feed rates is much more precise and quicker than that of a manual system. With the aid of electronic circuits to measure amperes on motors and mechanical loads, information and settings can be frequently adjusted to suit the incoming feed. If a hydraulic system is in use on the crusher the pressure in it can be measured to give the level of the crushing forces present in the machine. Manually controlled crushers under the control of a skilled operator seldom use more than 65 % of the permissible power. An automated feeder / crusher system can easily reach 90 %.

Level monitors provide a basic form of automation. Such a monitor in a cone crusher provides an automatic choke feed to the machine. Monitors placed in the bottom and top of a surge bin can stop the crusher when there is insufficient feed or provide protection from overfilling the bin.
Interlocking of the crushing system keeps spillage to a minimum by guaranteeing that the system starts in the right order. If a conveyor trips interlocking will stop all forgoing machines to prevent excessive spillage.

11.2 Maintenance

Efficient maintenance is the best way of maximizing the return on capital from a crushing plant. The following points list the basics of good maintenance:

11.2.1 Access and Lifting Gear

Provide sufficient space to work around crushers and to use the large tools. All inspection and lubrication points should be easily accessible. Plant should be designed with suitable lifting gear included or structures capable of supporting mobile lifting machines.

11.2.2 Manuals

Personnel should understand and be familiar with all the manuals supplied for the particular crushers purchased.

11.2.3 Training

All maintenance personnel must be qualified and properly trained.

11.2.4 Cleanliness

Dust can be hazardous to health. Piles of spillage can obscure areas of machines so that incipient failures cannot be seen. The most profitable crushing plants are the cleanest.

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11.2.5 Oil Analysis

Most lubricant suppliers provide an oil analysis service. Used regularly it will indicate trends and show up any lubrication problems which may be developing. It can give advance warning of accelerated wear.

11.2.6 Pre-Planned Manganese Replacement

A set of crusher liners will give full capacity output for a period of time and then wear will reduce output. At some stage in the wear it will be more economical to change plates than to operate at an increasingly reduced capacity.

11.2.7 Spare Parts

Where possible the use of standard basic machines for a new plant is recommended as this will reduce the spare part inventory. A spare parts stock at the plant is necessary to minimize downtime in the event of a breakdown.

11.2.8 Operation and Maintenance Log

Written records of operation and maintenance for each crusher will make it easier to spot repetitive problems and may give clues to an otherwise inexplicable failure. A regular log will soon indicate when liner changes are due.

11.3 Safety

Rules of procedure should be written down. High voltages and high pressures can be dangerous. Safety guards covering moving machinery will prevent accidents. Lockable isolating switches should be provided for all motors preventing unauthorized starting.

Rocks in a crusher are subjected to large forces and can shatter propelling splinters out of an open crusher mouth with great force. These will penetrate a body like a bullet. Personnel working in a crusher plant should be provided with adequate safety gear and warnings of the sources of danger.

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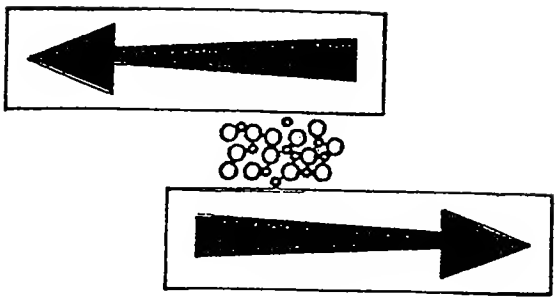
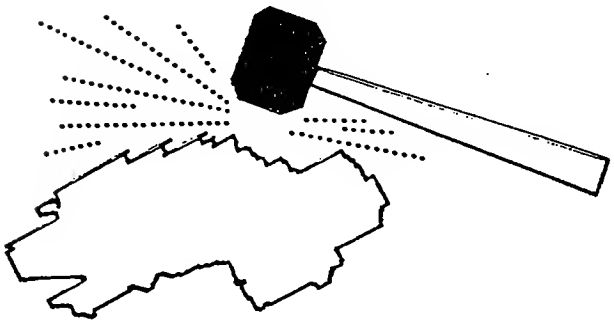
Purpose of Raw Material Crushing

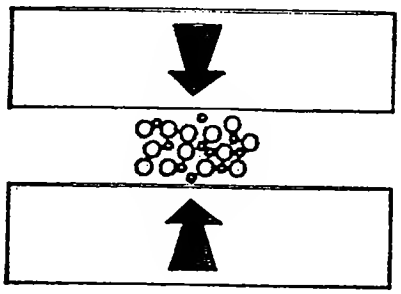
- Suitable Size for Belt Transport
- Possibility for Prehomogenisation
- Feed Granulometry of Raw Mill
- Mixing of different Raw Materials



Attrition



Impact



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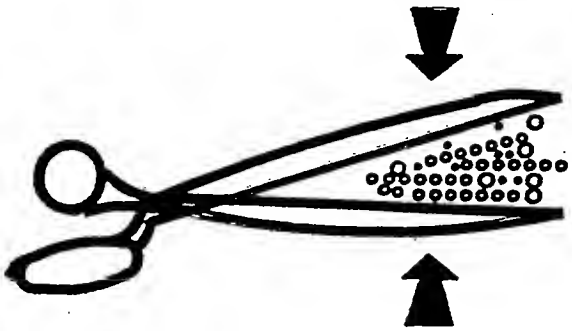
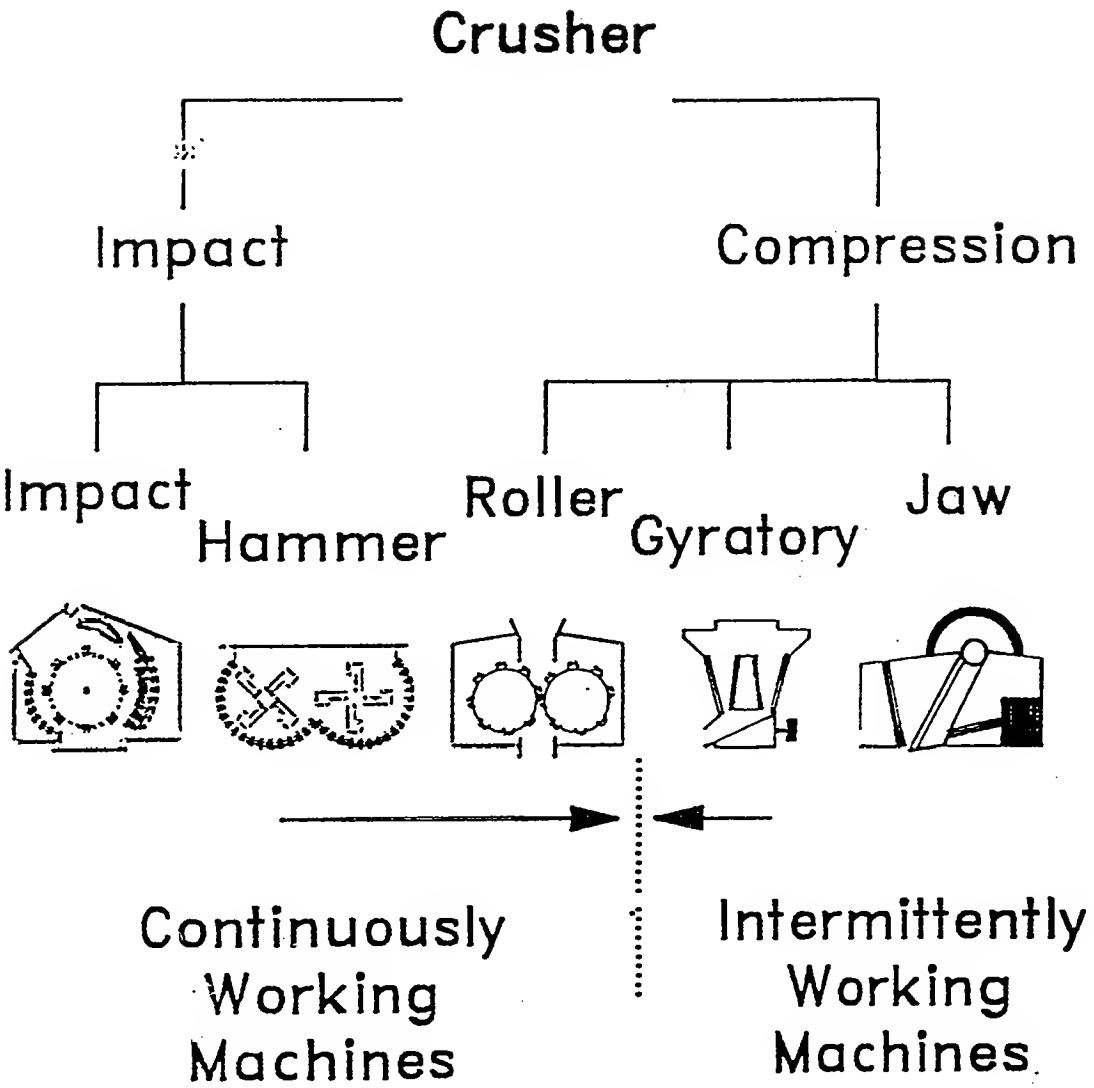
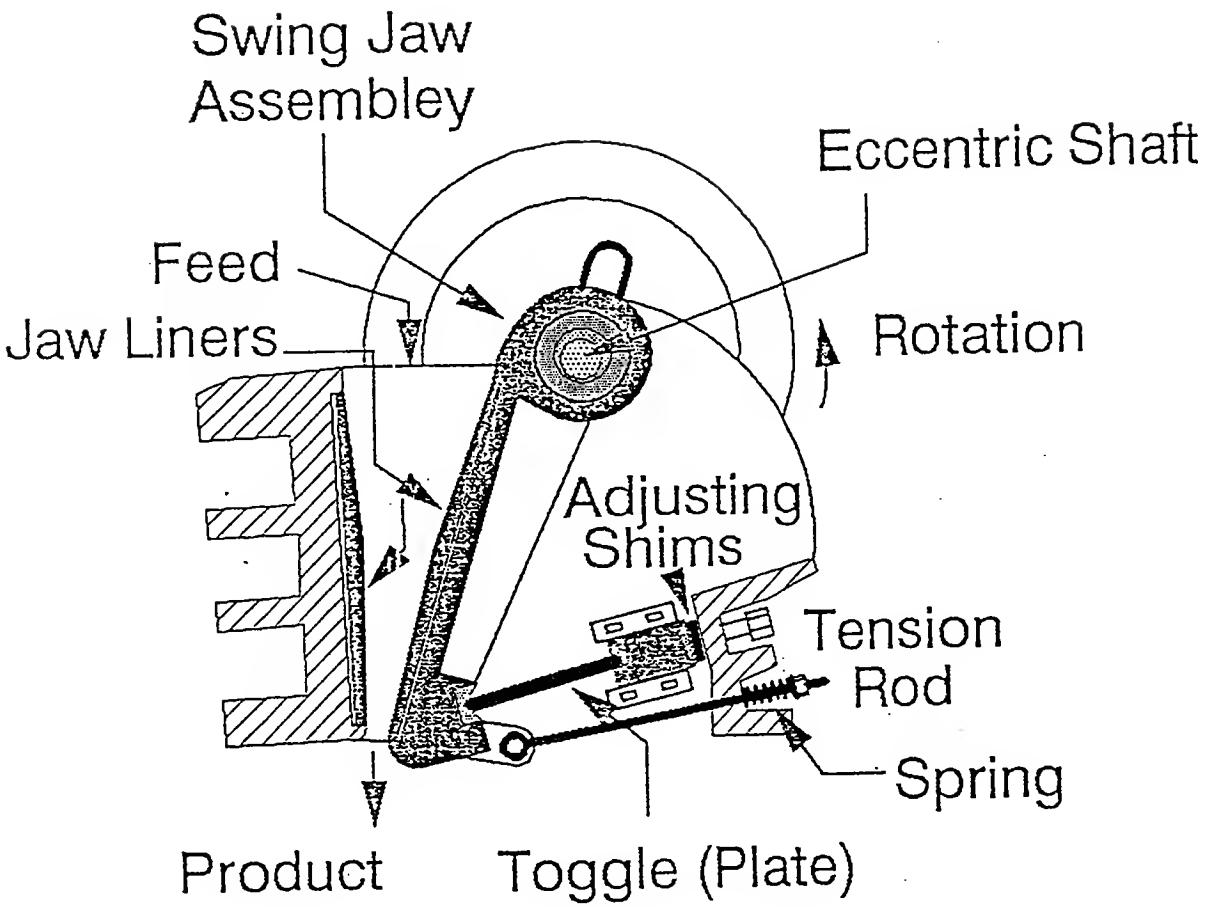


Shear

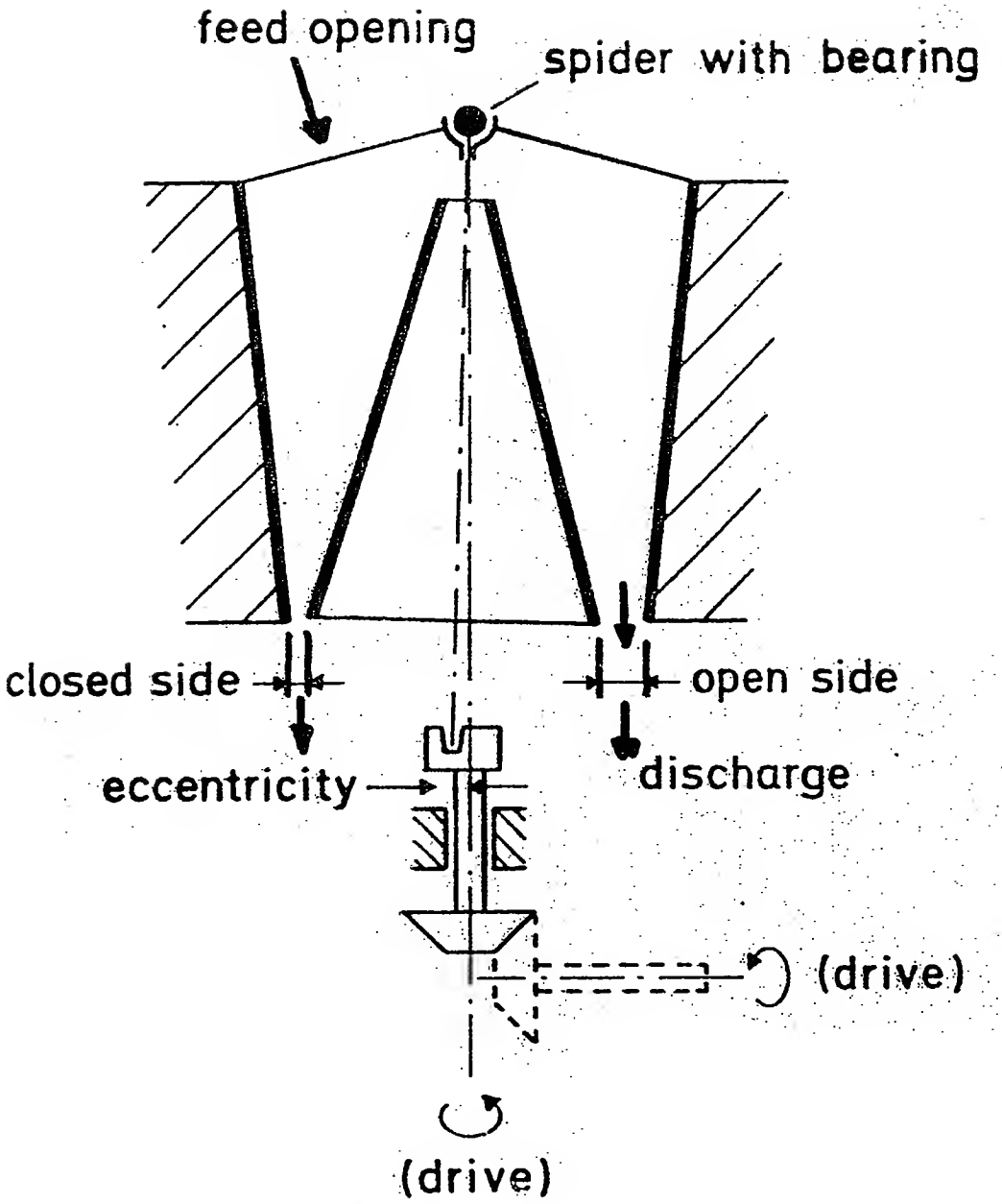
Mechanical Reduction Methods



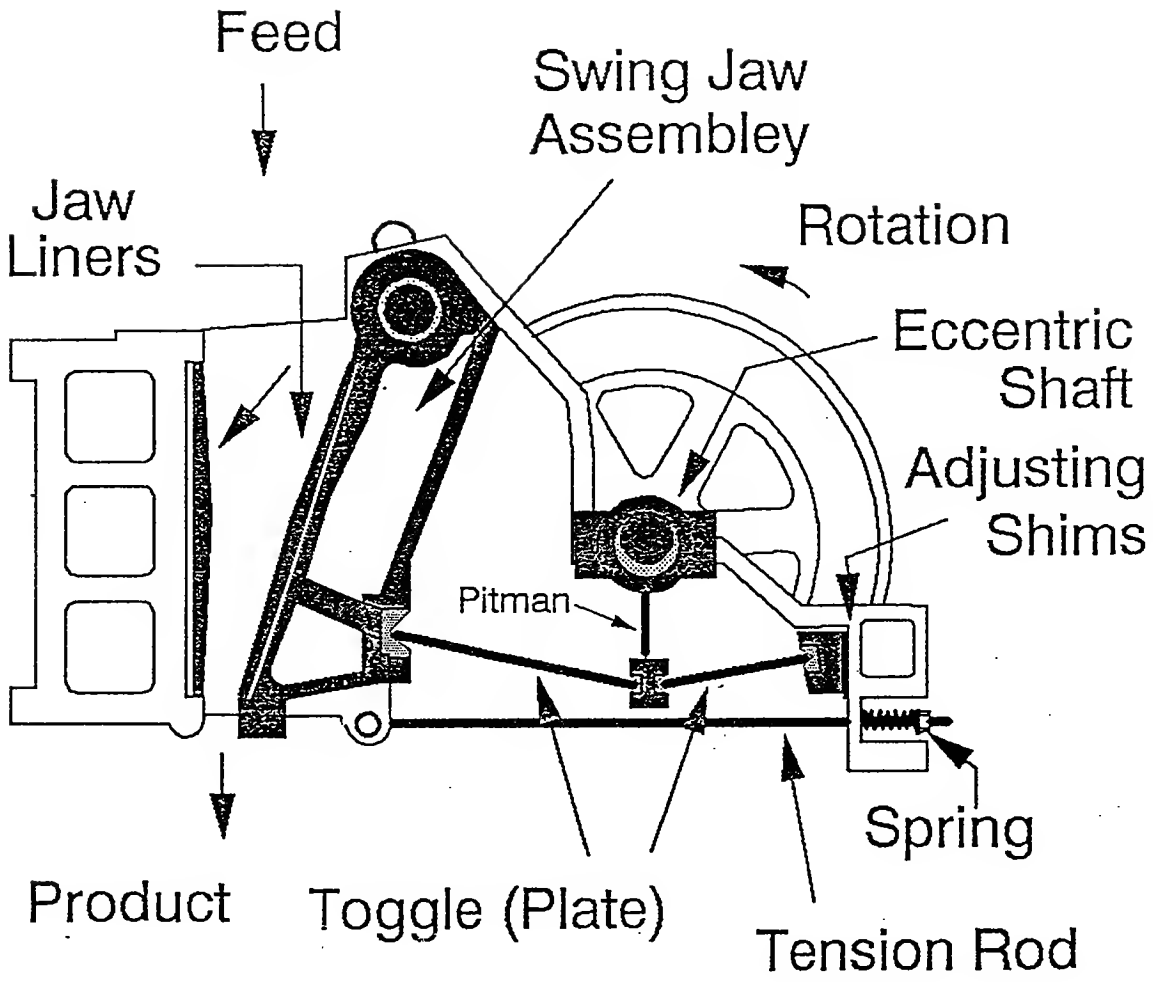
Crusher Types



Single Toggle Jaw Crusher



Gyratory Crusher



Double Toggle Jaw Crusher

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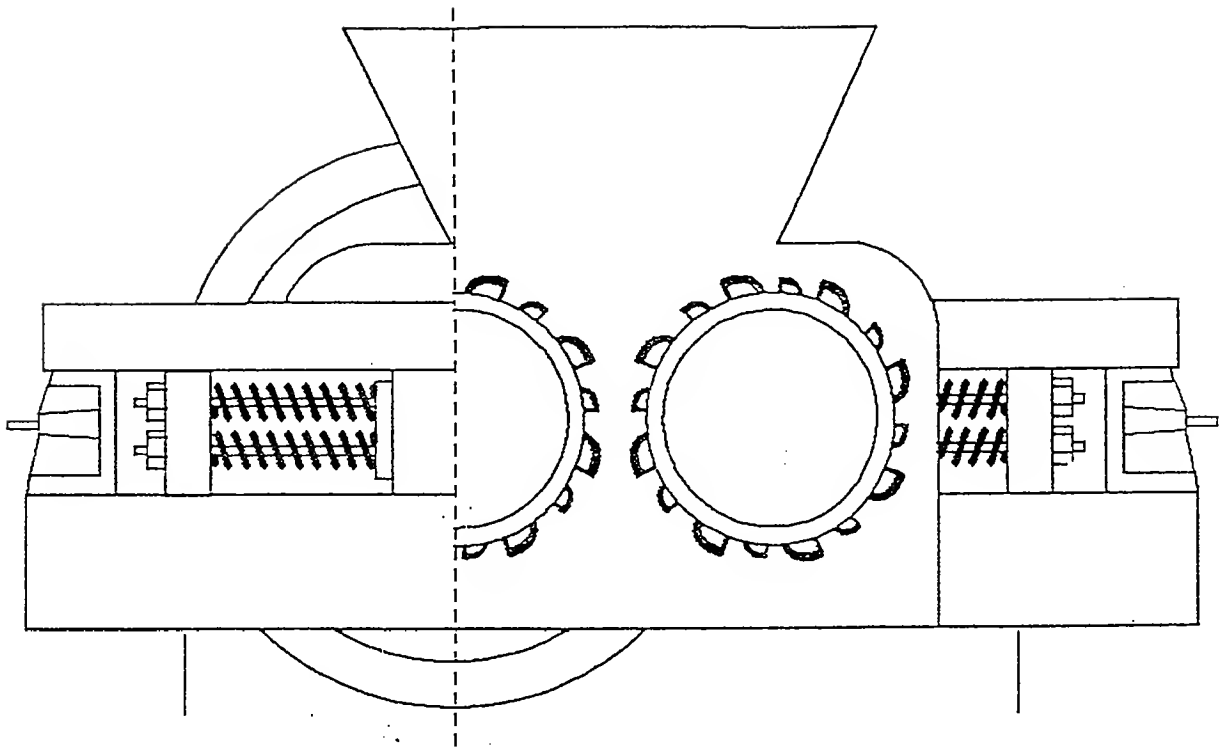
CRUSH004



Double Roll Crusher

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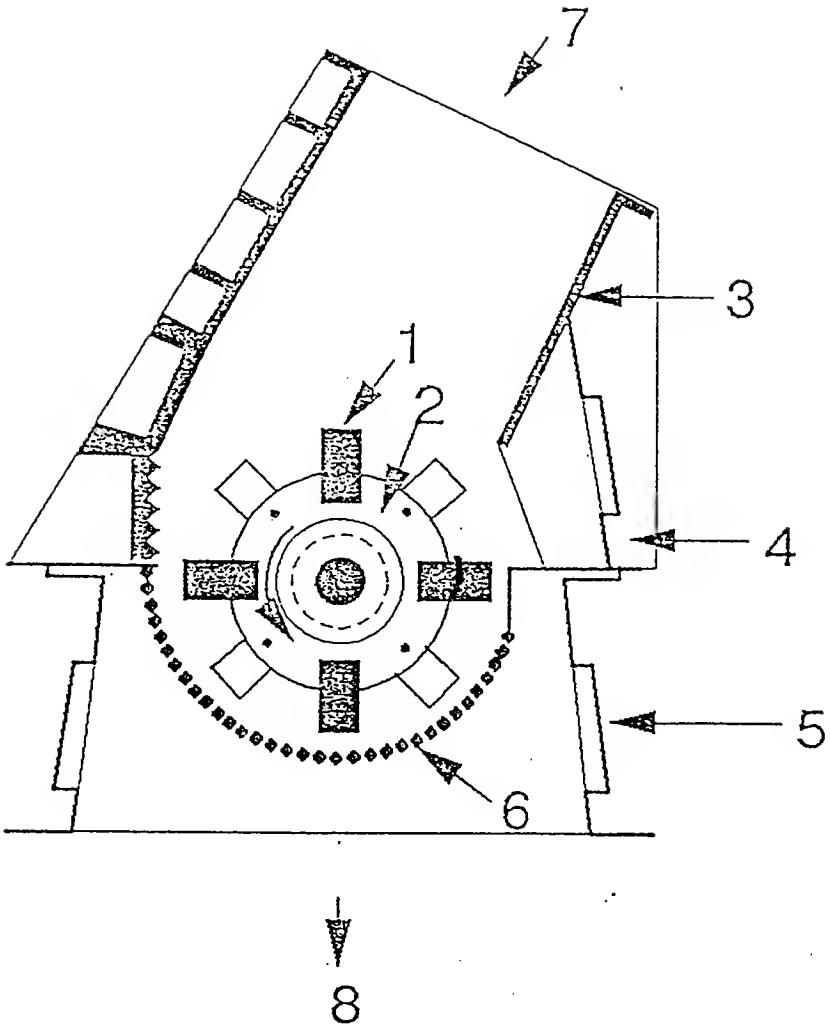
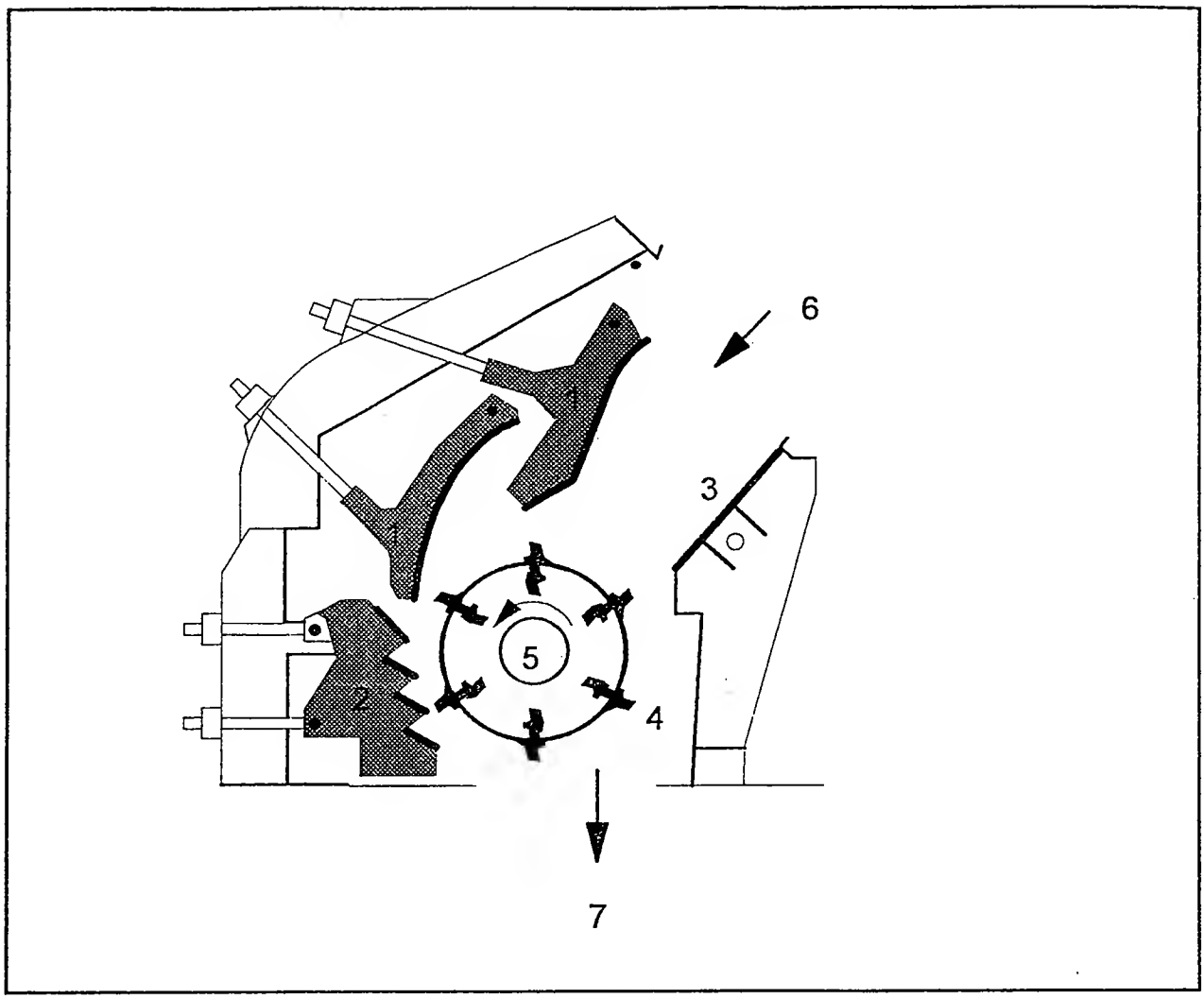
CRUSA02R



1 Hammers	5 Lower Casing
2 Rotor	6 Grates
3 Feed Plate	7 Feed
4 Upper Casing	8 Product

Single Shaft Hammer Crusher with grates

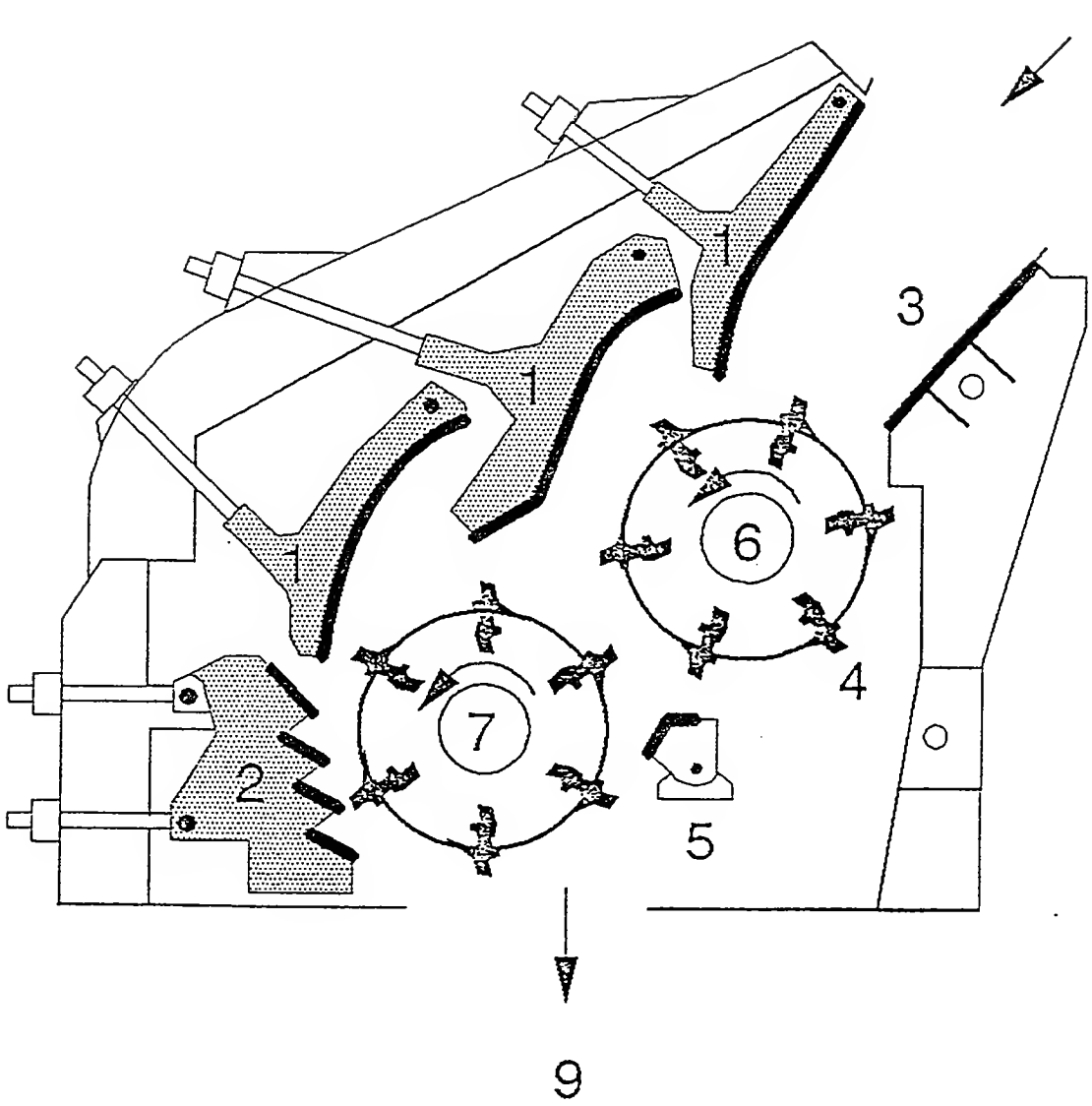
RF:92:04
CRUSA003



1 Impact Plates
2 Grinding Path
3 Feed Plate
4 Impact Bars
5 Rotor
6 Feed
7 Product

Impact Crusher

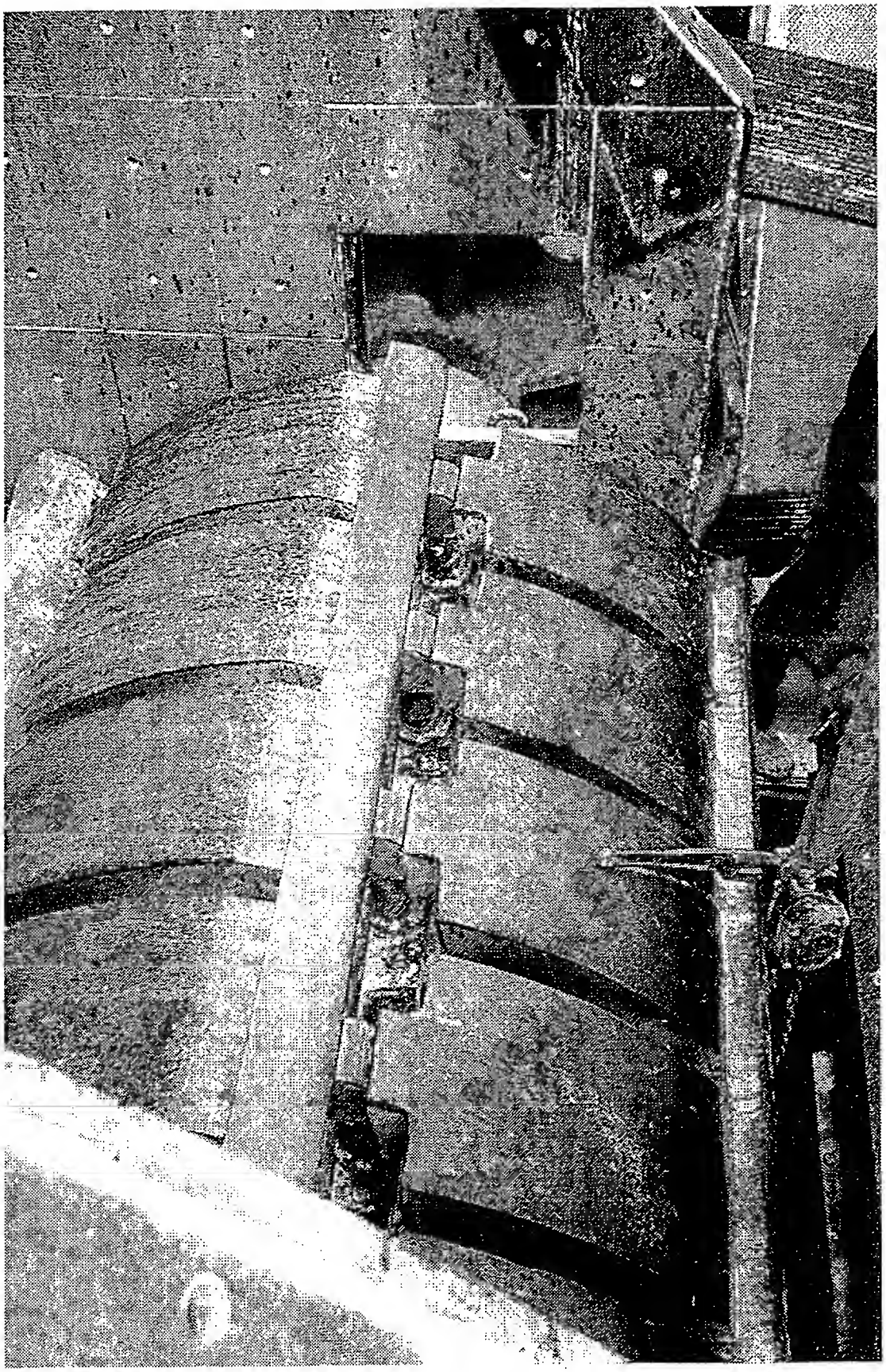
RP:92:04
CRUSH12A

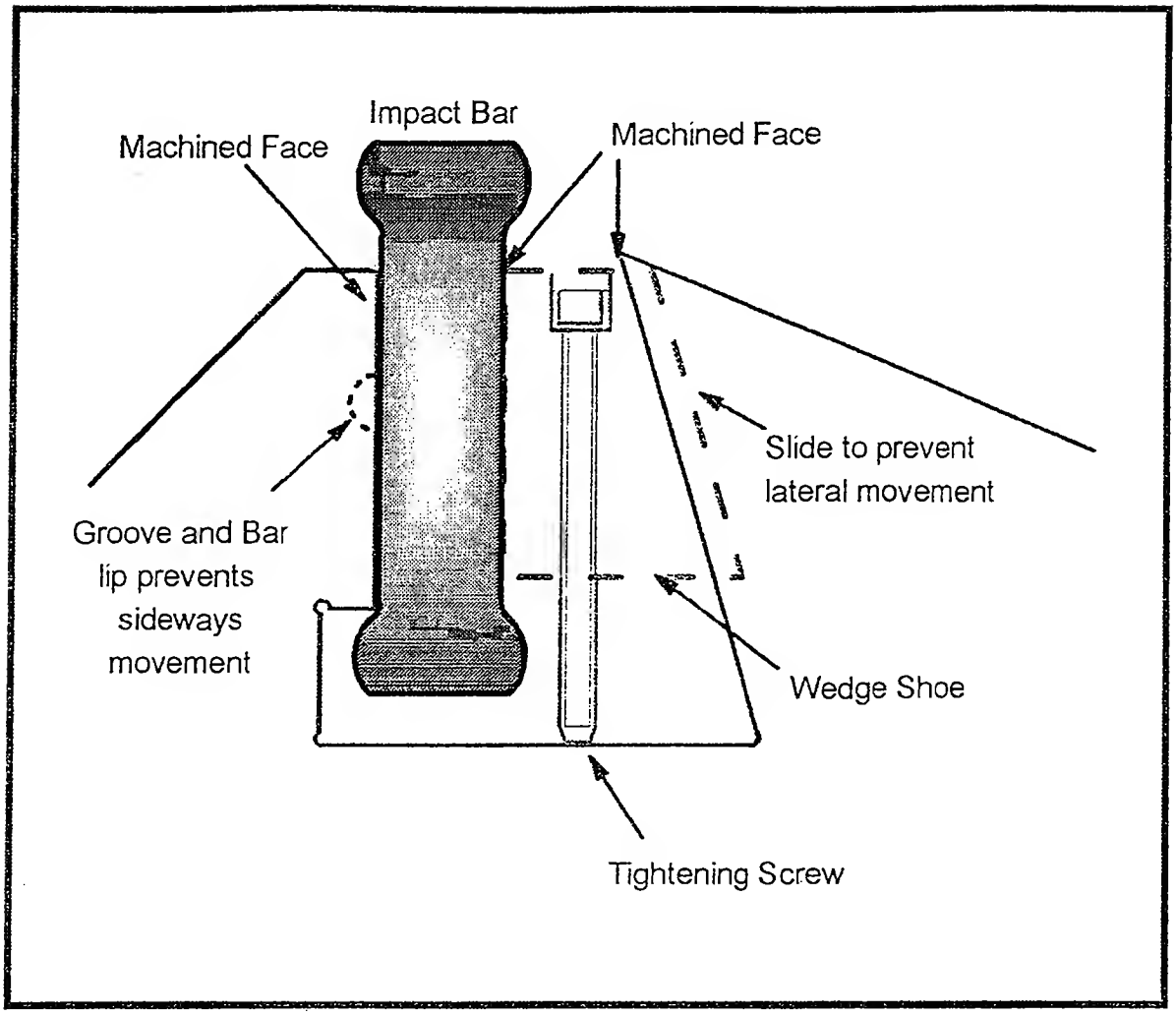


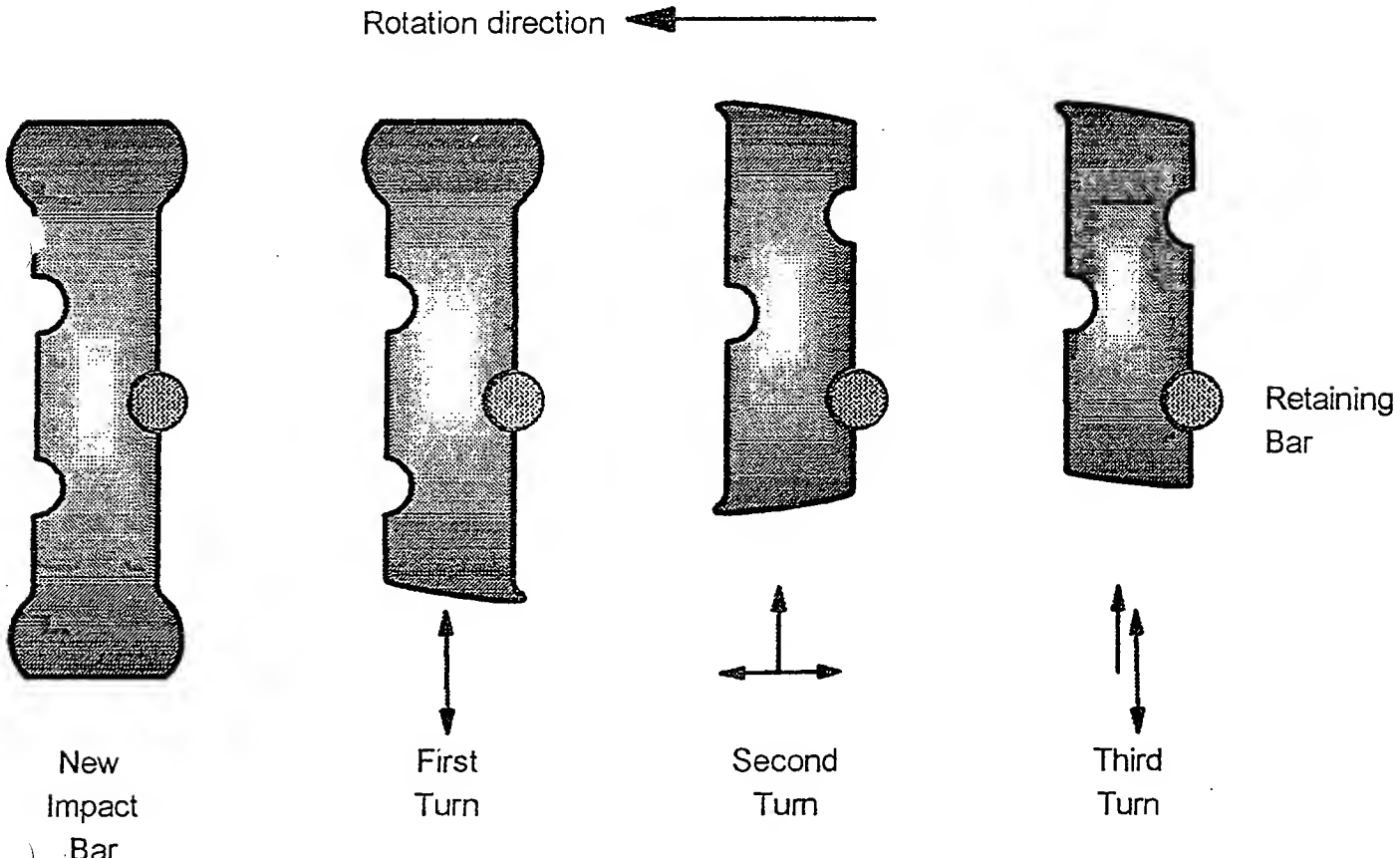
1 Impact Plates
2 Grinding Path
3 Feed Plate
4 Impact Bars
5 Anvil
6 First Rotor
7 Second Rotor
8 Feed
9 Product

Compound Crusher with grinding path

RF:92:04
CRUSA012





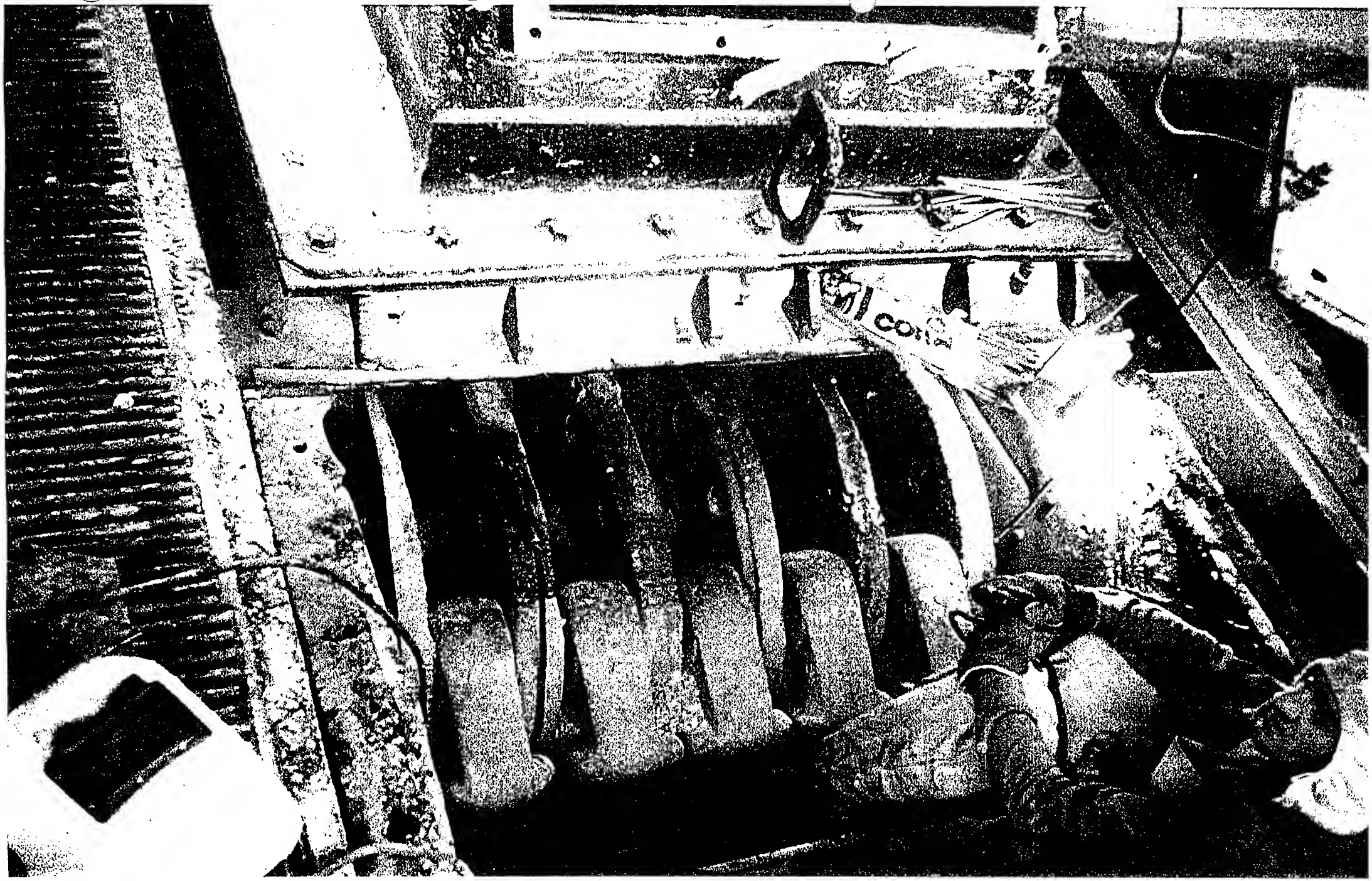


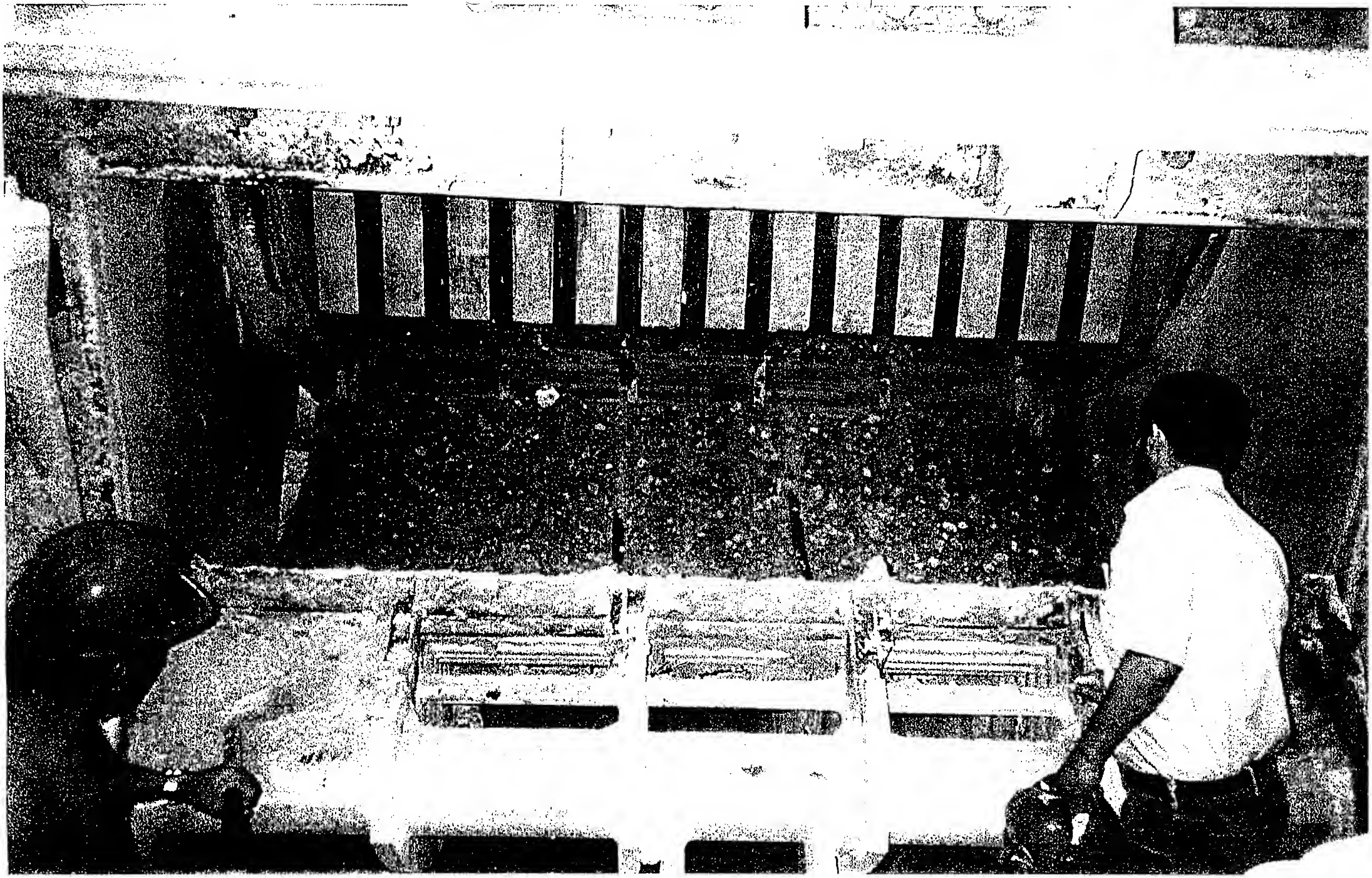
Bar Rotation

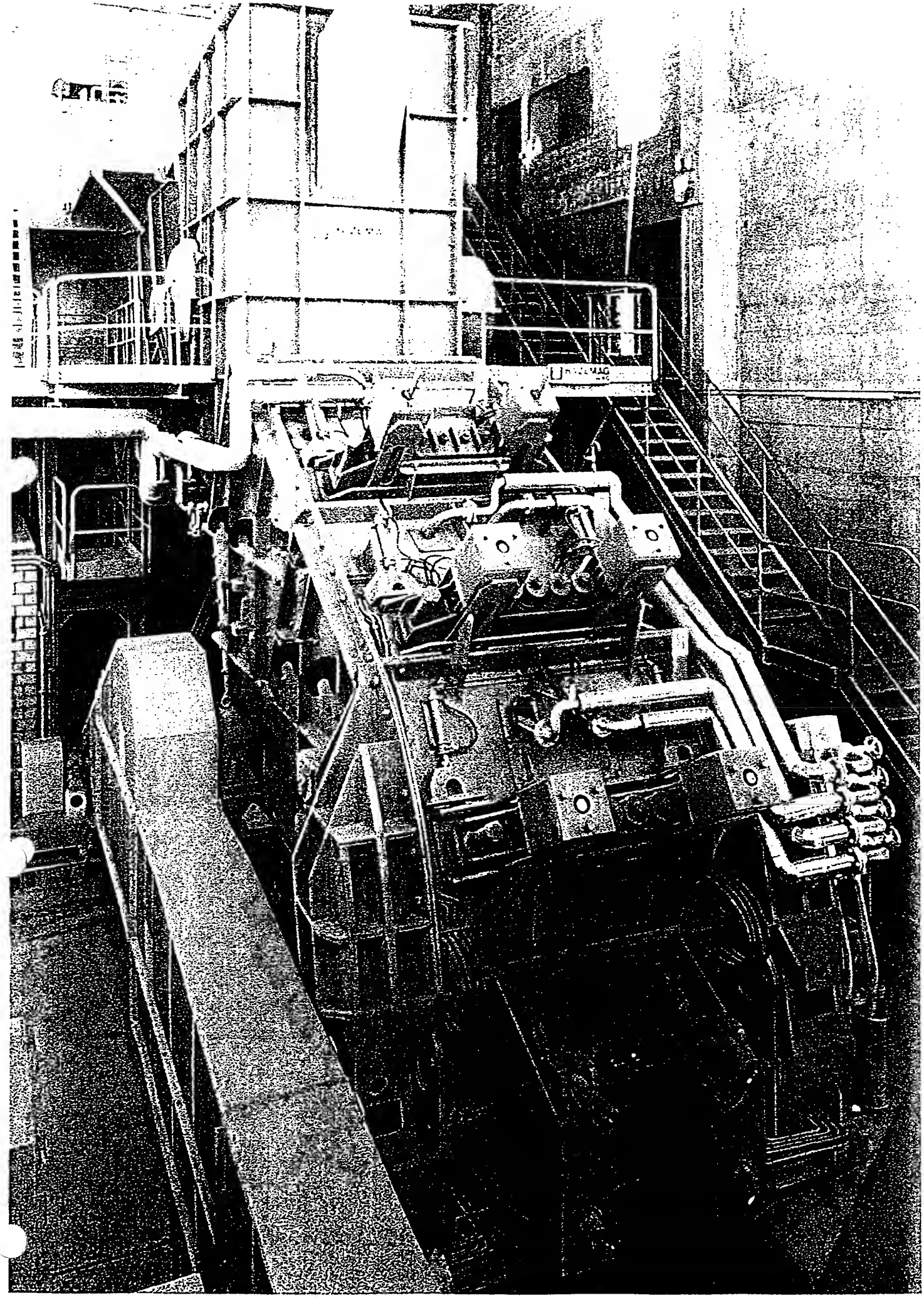
Application of Crusher Types

Material Conditions Demands / Crusher Type	Material Classification			Crushing Ratio				Material Abrasivity			Material Moisture		
	soft	med.	hard	1:5	1:15	1:30	1:50	low	med.	high	dry	moist	wet
Single Toggle Jaw	✓	✓	○	✓	✗	✗	✗	✓	○	✗	✓	○	✗
Double Toggle Jaw	○	✓	✓	✓	✗	✗	✗	✓	✓	✓	✓	○	✗
Gyratory/Cone	○	✓	✓	✓	✗	✗	✗	✓	✓	✓	✓	○	✗
Roll	✓	○	✗	✓	✗	✗	✗	✓	○	✗	✓	✓	✓
Single Shaft Hammer with Grate	✓	✓	○	✓	✓	✓	○	✓	○	✗	✓	✗	✗
Single Shaft Impact	✓	✓	✓	✓	✓	○	✗	✓	○	✗	✓	○	✗
Double Shaft Impact (Compound)	✓	✓	✓	✓	✓	✓	○	✓	○	✗	✓	○	✗

✓ Most Suitable ○ Suitable ✗ Not Suitable

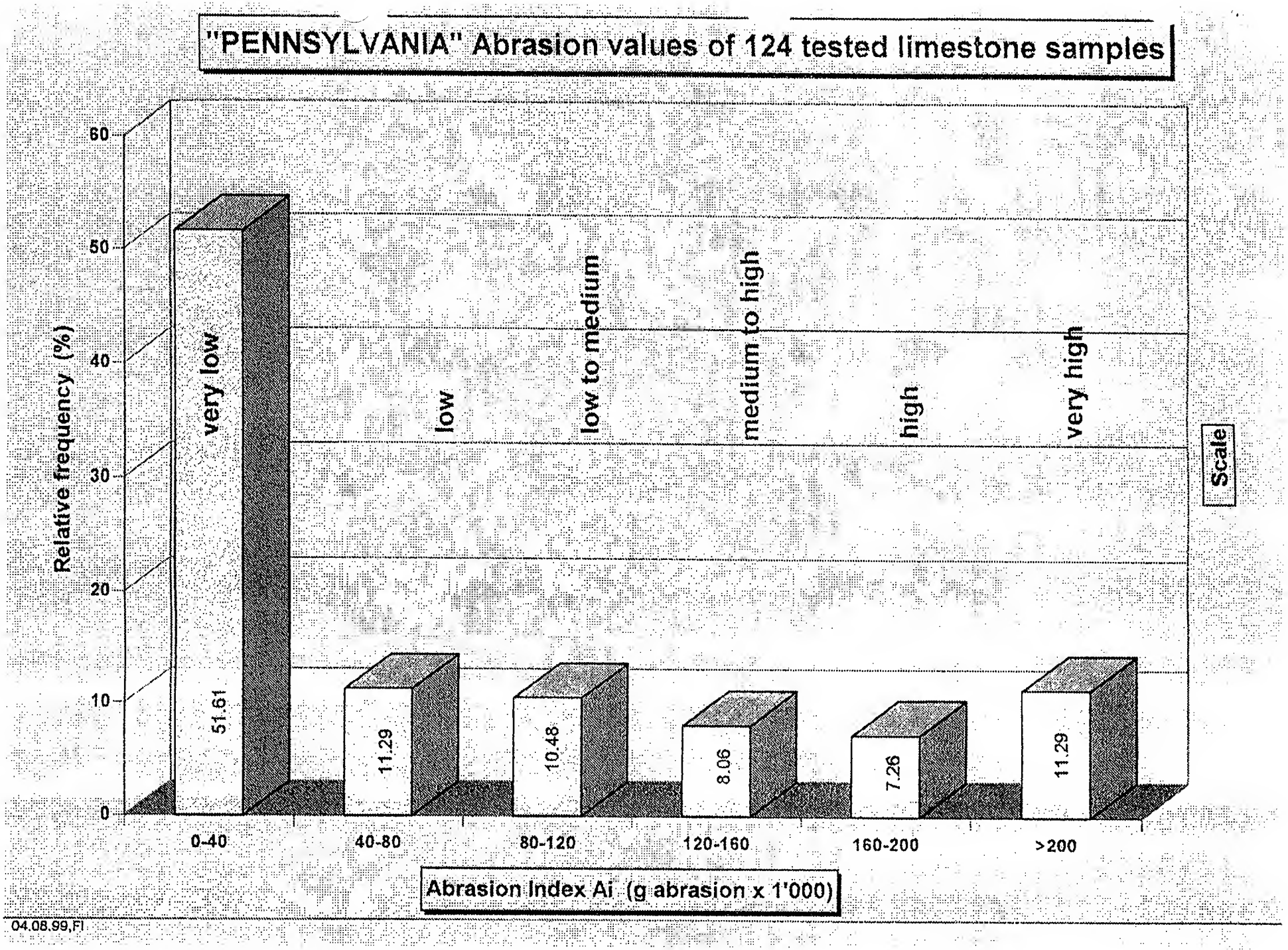






Crusher Selection Criteria

- **Work Index**
  - Power per unit mass required to break an infinite particle size down to 80% passing 100 μm (Wi of Limestone: 9 - 15 kWh/t).
- **Abrasiveness**
  - Pennsylvania abrasion test index Ai.
- **Stickiness**
- **Tons per Hour**
  - crusher is interface between continuous and discontinuous operation
  - consider: surge, blocking, shortfalls in production
  - minimum reserve capacity = 25%
- **Size of Feed**
- **Reduction Ratio**
- **Power Consumption**


"PENNSYLVANIA" Abrasion values of 124 tested limestone samples
Relative frequency (%)
60
50
40
30
20
10
0
very low
low
low to medium
medium to high
high
very high
Scale
51.61
11.29
10.48
8.06
7.26
11.29
0-40
40-80
80-120
120-160
160-200
>200
Abrasion Index Ai (g abrasion x 1'000)
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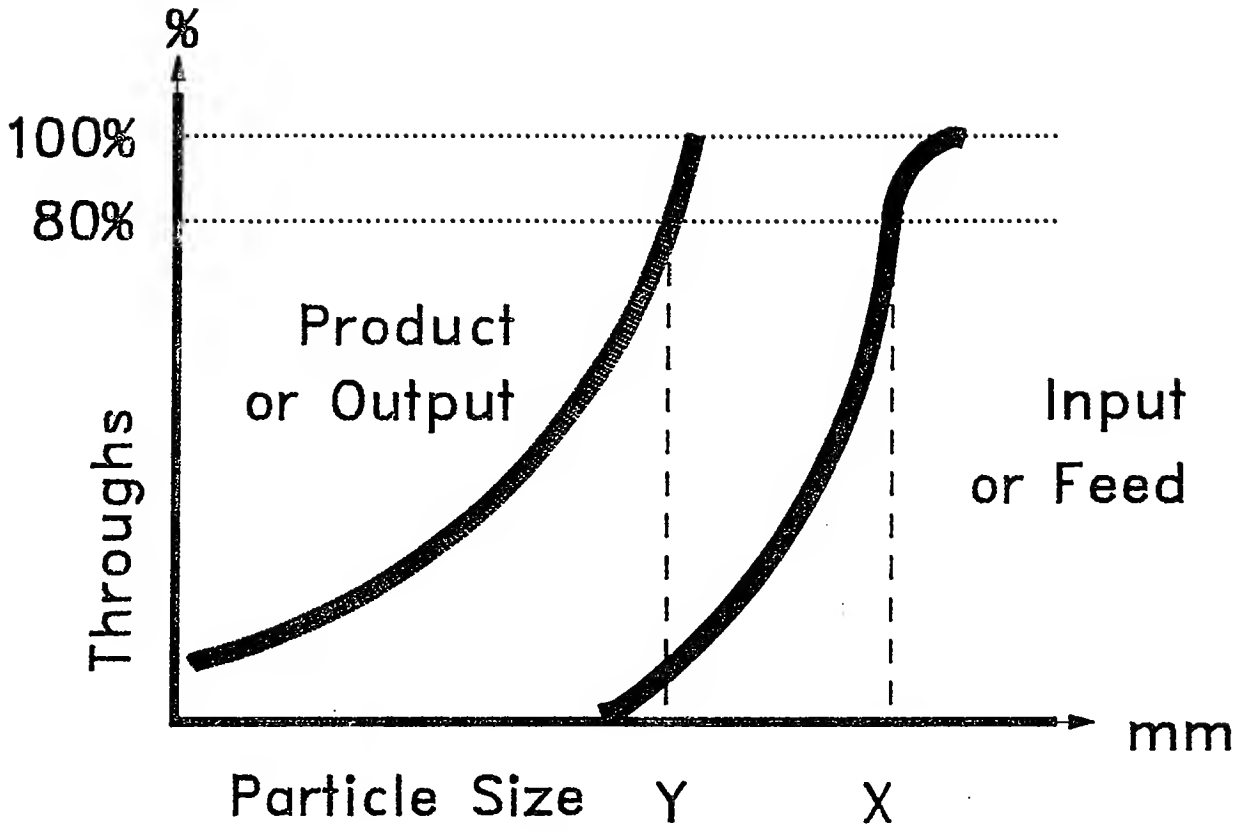
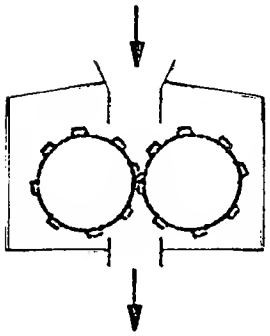
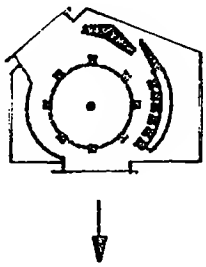


Reduction Ratio = X/Y

Reduction Ratio

Power Consumption

Medium Hard Limestone

Primary	Secondary
ROQ	165 mm
	
165 mm	12-15 mm
600 to 800 w/t/h	1000 w/t/h

Crusher	w/t/h
Roll	500
Jaw	250
Hammer	1000

09:93:RP

Energy Consumtion in Crushers

Reduction Ratio

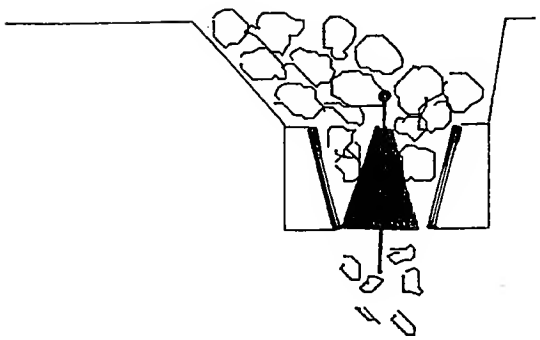
to crush to the maximum ratio uses 30%
more energy than crushing to
80% to 85% of maximum

Moisture Content

water saturated feed uses 2 * the energy
of dry feed

Strength

the strongest rocks consume 50% more
energy than the weakest



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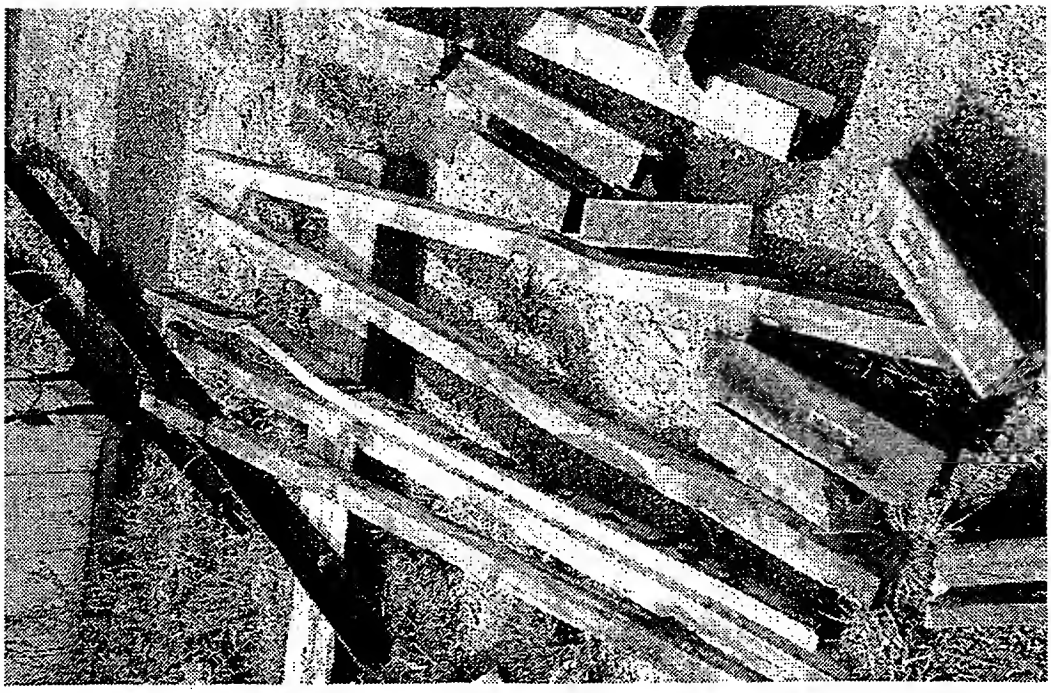
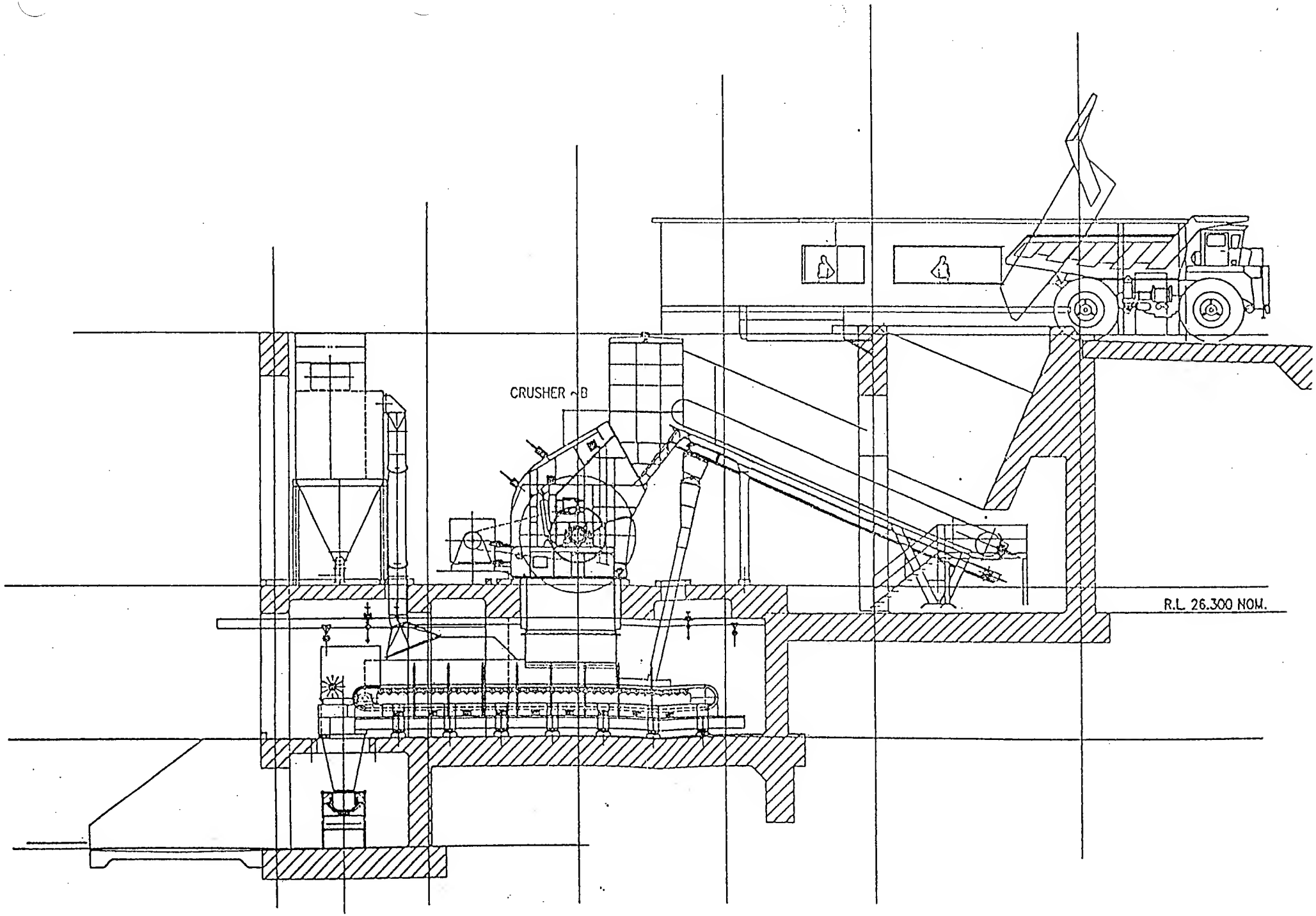
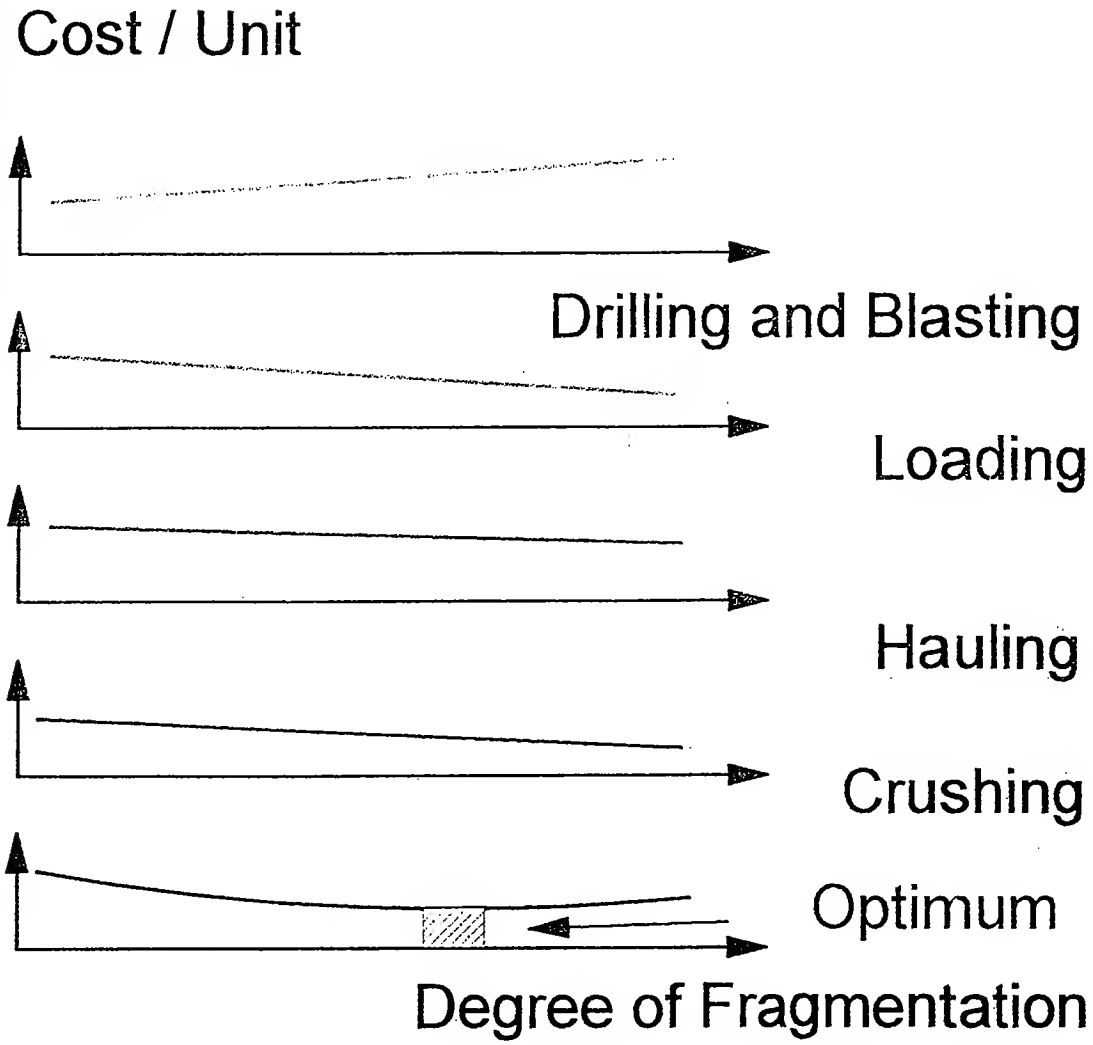



CRUSHER - B
R.L. 26.300 NOM.

Quarry

Cost as a function of Degree of Fragmentation



Process Technolog,

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Chapter 3

Design of Bins and Feeders

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2. Coal Storage and Proportioning 163

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Design of Bins and Feeders

F. Bucher
PT 97/14241/E

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1. INTRODUCTION

Looking at the flow sheet of the cement process (Fig. 1) the importance of bulk solid storage in bins and silos becomes evident. Bins and silos are installed either

- as *intermediate stores* as to compensate for differences in material demand of subsequent processes (e.g. clinker and cement silos) or
- as *feed bins* installed for feed rate control (e.g. feed bins for raw, coal and cement mills)

Bins have been considered for too a long time as unimportant and cheep equipment when compared to mill and kiln systems. They have been considered to be simple containments only which are required to be fed with product for storage with the aim that this product may be reclaimed later at a given time at a consistent rate. The physical product characteristics have typically been ignored in bin design with exception of the angle of repose which may have been considered by using slightly steeper wall inclination for a bin's discharge hopper. Making optimum use of a given area was the prime design criteria. So it is not surprising that operation of many bin installations is still impaired by too flat concrete discharge hoppers designed with a too small outlet and equipped with inappropriate feeders. Operation problems as

- erratic or even interrupted product flow out of bins with cohesive bulk solids,
- flushing of the feeders with powders,
- product segregation,
- incomplete emptying resulting in a reduced bin live capacity

are common experience and very often accepted to be inevitable. The use of a sledge hammer and poke rods are well known but ineffective means to compensate for the adverse consequences of a careless and incompetent bin design.

It has to be mentioned that the attitude of the cement industry's towards bin design has changed fundamentally in the past fifteen years. Modern test procedures (as the shear test) and design concepts (mass-flow) are no longer ignored but accepted.

In this paper an attempt is made to give a survey on 'bulk solid storage in bins'. The following topics will be dealt with:

- description of the different bin design concepts and the resulting flow patterns,
- description of the bulk solid characteristics which are relevant in bin design and their experimental determination,
- analysis for product flow in bins,
- criteria for the design of bin/feeder systems,
- possible measures for improving the flow pattern for existing bins.

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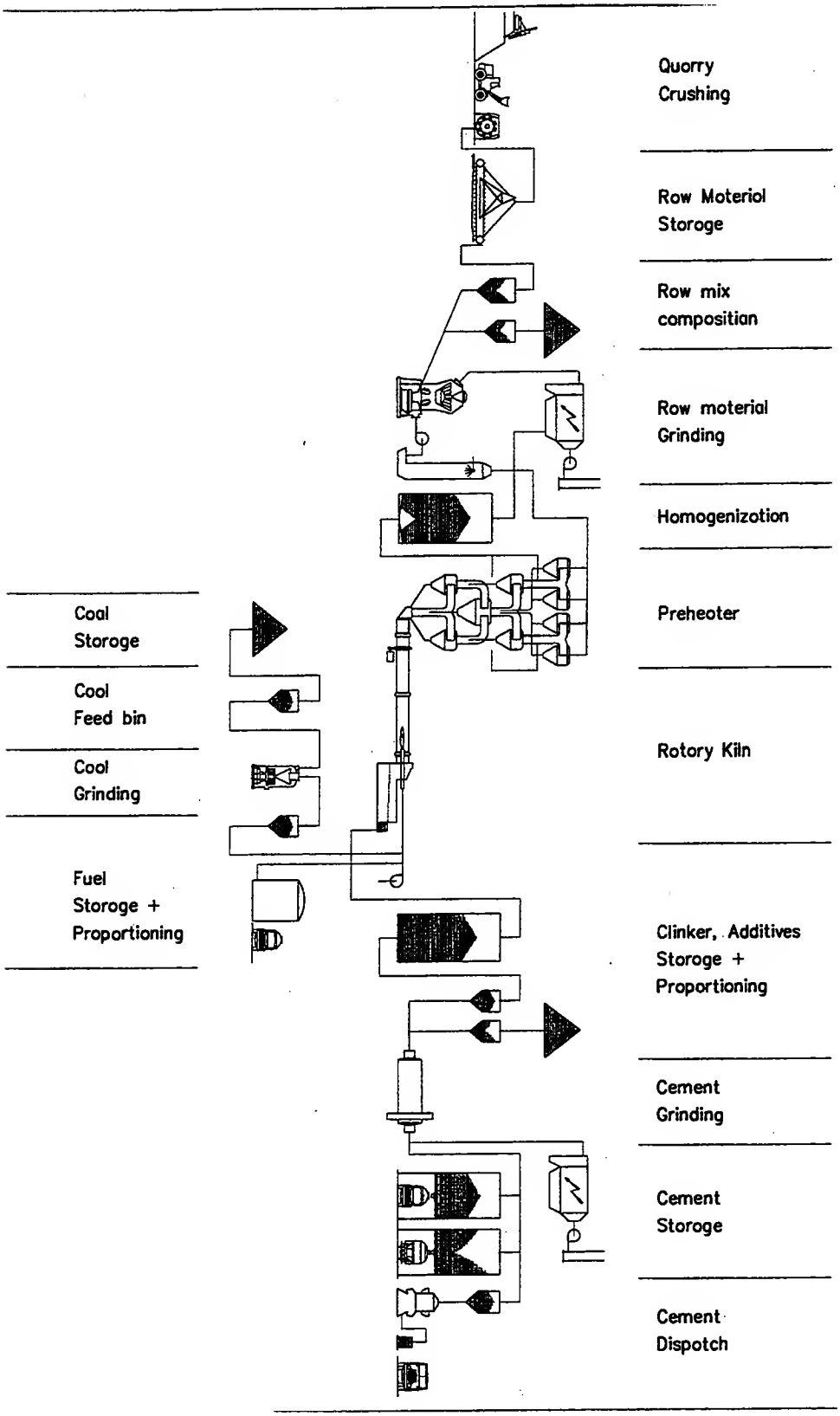
Figure 1 Cement Process Flow Sheet



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2. BIN DESIGN CONCEPTS

The flow pattern that develops in bins as the stored bulk product discharges under gravity is commonly demonstrated by means of transparent bins filled with layers of differently colored product. Such a set-up allows to observe the changes in stratification during flow, helped to give an insight to the nature of gravity flow of free-flowing and cohesive bulk products and allowed to understand the influence of wall inclination and outlet size on the flow pattern. It became evident the flow pattern can be classified into two

- funnel-flow, and
- mass-flow.

2.1 The Funnel-Flow Concept

2.1.1 Bin Shapes

The funnel-flow bin type is the bin type with still the widest application in cement industry. Such bins may either be flat-bottomed or equipped with discharge hoppers. In case of the latter the flat hopper wall inclinations are the typical bin characteristic. Typical funnel-flow type bin shapes are shown in Fig 3.

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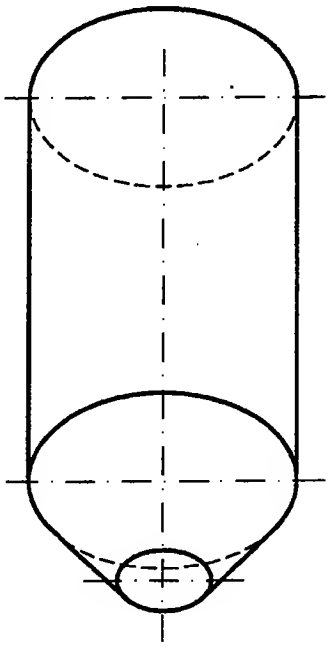
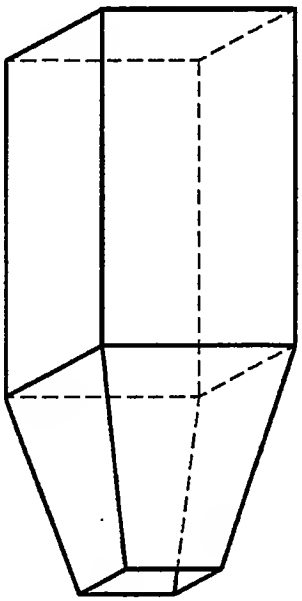
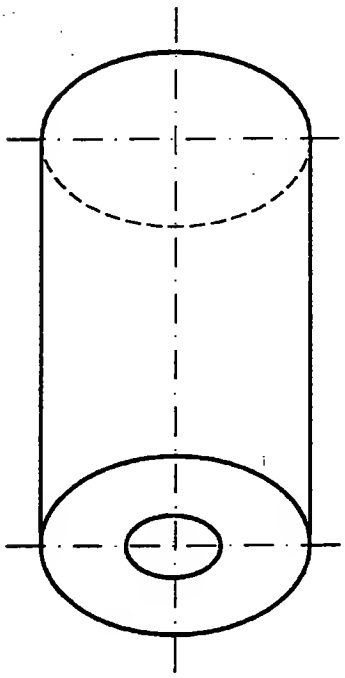
Figure 3 Funnel-Flow typical bin shapes



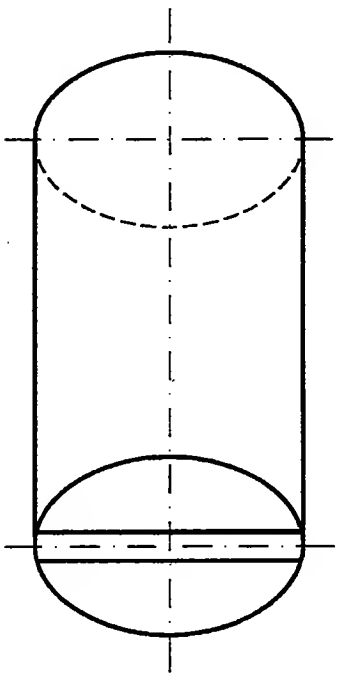
conical hopper
circular outlet



pyramid hopper
square outlet



flat-bottomed
circular outlet



flat-bottomed
slot outlet

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2.1.2 Flow Pattern

When filling a bin, product will always segregate: The coarse grains will roll down the slope of the surface pile to the bin's circumference while the fines will predominantly be accumulated in the bin's center. (Fig 4.1)

Development of the flow pattern in funnel-flow type bins depends on the product's flow characteristics, the relation of feed rate versus discharge rate and the arrangement of the feed relative to the discharge opening. When opening the outlet of a funnel-flow type bin, product will flow in a narrow core only that forms just above the bin's outlet while the surrounding product remains at rest. Bulk solid discharge is essentially irregular regarding rate and density. Funnel-flow type bins can typically not be emptied completely what results in a reduced live capacity. The discharge order is inverted compared to the loading order, i.e. the bin shows a first-in last-out flow sequence.

- For *no feed conditions* (Fig. 4.2) the product that will be discharged first is the fine product from the bin center zone. Product surface will then be inverted from a pile to a funnel shape, product level in the bin will fall. With the coarse particles from the bin's circumference now rolling down the slope to the center, granulometry of the discharged product changes from fine to coarse. All the product surrounding the center core remains at rest. The discharge order is inverted compared to the loading order, i.e. the bin shows a first-in last-out flow sequence.
- In case the *feed rate is larger than the discharge rate* (Fig. 4.3) product level in the bin is rising, the product surface keeps a pile shape and the product's fine portion is predominately discharged as all the coarse particles roll down the pile slope. All the product surrounding the center core remains at rest.
- In cases that *feed rate and discharge rate are similar* (Fig. 4.4) product level in the bin and shape of product surface will remain constant. Product do not segregate on feeding as it passes directly to the bin outlet via the activated center core. Again all the product surrounding the center core remains at rest.

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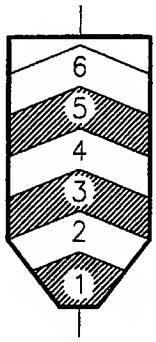
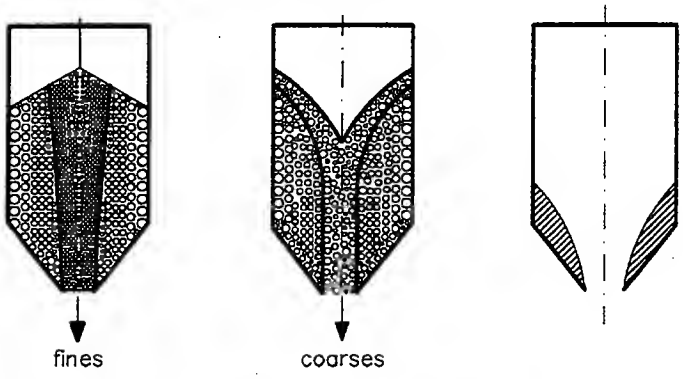
Figure 4 Flow sequence in funnel-flow type bins

FIG. 4.1 Feed no discharge



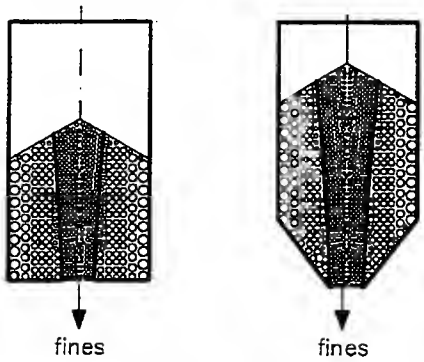
- product segregation

FIG. 4.2 Discharge no feed



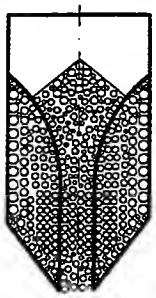
- falling product level
- prefered discharge of coarse fraction

FIG. 4.3 Discharge rate ≪ Feed rate



- rising product level
- prefered discharge of fine fraction

FIG. 4.4 Discharge rate = Feed rate



- constant product level
- no segregation

2.1.3 Operation Problems (Fig. 5)

Erratic flow and Flushing

Bulk solids do not flow consistently in vertical or very steep channels and have a tendency to form voids topped by product arches. As such an arch breaks, the product falls into a void. Coarse product tends to pack under the impact on the feeder and to form another arch again, discharge rate will be erratic. Fine powders will get aerated when falling into the void and may then behave as a liquid flushing the feeder.

Interrupted Flow: Arching and Rat-holing

Funnel-flow type bins are subject to arching and rat-holing. Product discharge may be interrupted when product consolidation in the bin is that important that formation of a stable cohesive arch or rat-hole becomes possible.

Product degradation

Product may degrade due to segregation or excessive storage at rest.

Product segregation when filling bins can not be avoided. The funnel-flow sequence do not correct the adverse effect of segregation.

Extensive storage at rest of the product in a bin's dead zones may result

- in lump formation with the possibility that such lumps plug later the bin outlet,
- in hang-ups on the bin walls,
- in a loss of bin live capacity due to incomplete emptying.

2.1.4 Evaluation of the Funnel-Flow Concept

\+ maximized storage volume for a given headroom but only in case the bin volume may be cleared completely on a batchwise basis

\+ low investment per m^3 storage volume

\- tends to erratic flow and an inconsistent discharge rate (inconsistent regarding flow rate and bulk density)

\- first-in last-out flow sequence

\- varying product quality at bin discharge as product segregation on feeding is not corrected

\- non-flowing zones favoring product degradation with time resulting in spoiling, lump formation, wall hang-ups and loss of live capacity due to incomplete emptying

\- subject to flow stoppages if the product consolidates (packs) sufficiently to form stable arches or rat-holes

→ *Funnel-flow may be acceptable in cases*

- where segregation is unimportant,
- where product degradation is not likely to be a problem,
- where feed rate consistency is not of prime importance

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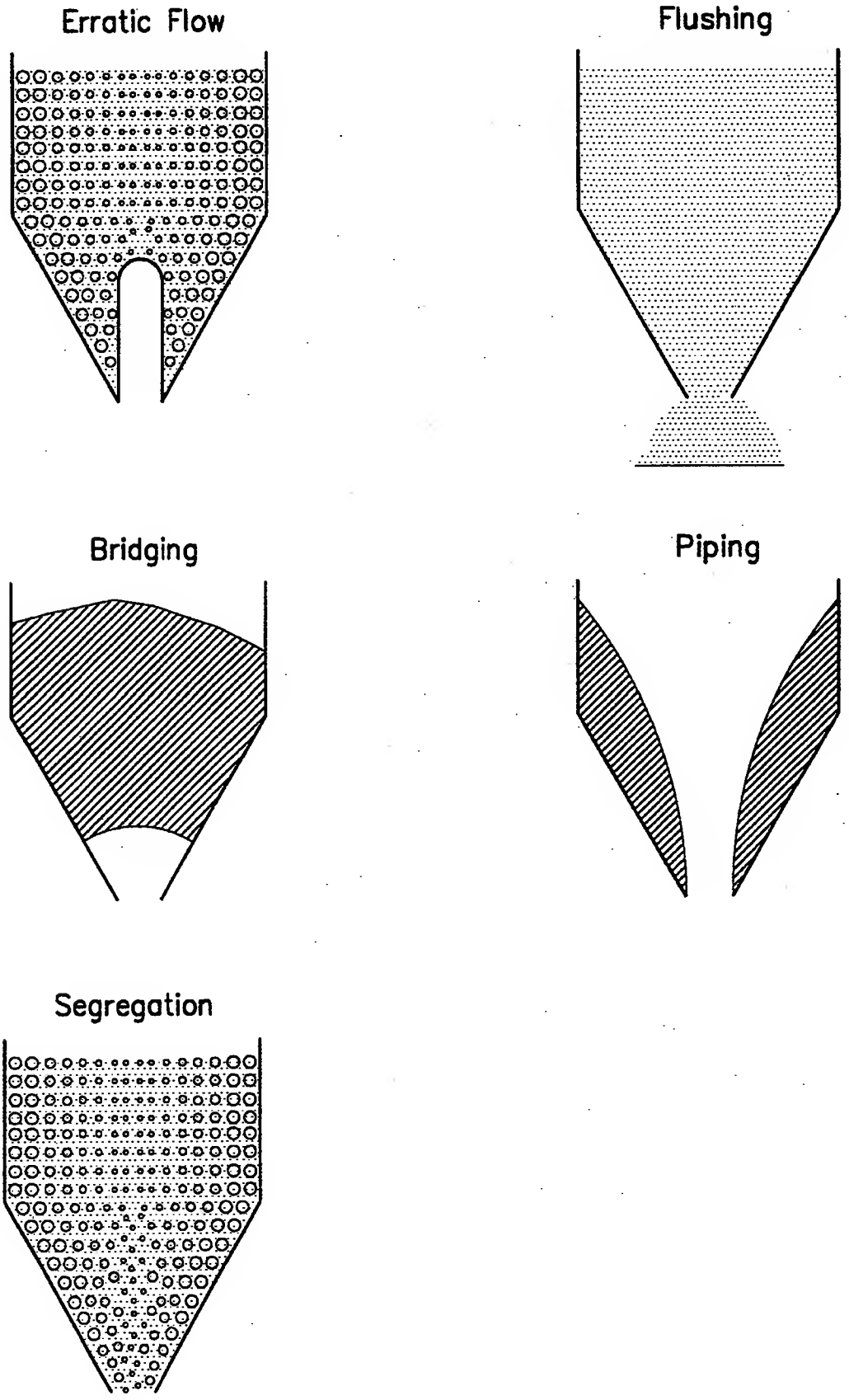
Figure 5 Operation problems with funnel-flow type bins



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2.2 The Mass-Flow Concept

2.2.1 Bin Shapes

Typical characteristics of mass-flow bins are a slender shape, steep wall slopes of the discharge hopper and relatively large outlets. Typical mass-flow type bin shapes are shown in Fig 6.

2.2.2 Flow Pattern

Product segregation when filling a mass-flow type bin can not be prevented with the coarse grains rolling down the piles slope to the bin circumference while the fines will predominately be accumulated in the bin center.

The flow pattern that develops in a mass-flow type bin differs from that of a funnel-flow type bin considerably. In mass-flow type bins all the stored product is activated and moves towards the bin outlet whenever product is discharged. This results in a first-in first-out flow pattern (Fig. 7). Mass-flow type bins can typically be emptied completely at uniform discharge rate.

2.2.3 Operation Problems

Consistency of Product Discharge

Product discharge out of mass-flow type bins is consistent in rate and bulk density thus not problematic.

Interrupted Flow: Arching

A mass-flow type bin only looses its beneficial properties in case the outlet is insufficiently sized or the outlet is not fully effective. In such cases arching and rat-holing become possible with the flow pattern changing to funnel-flow.

Product degradation

Zones of stagnant product will not form in mass-flow type bins at least as long as the bin outlet is fully active i.e. as long as all the stored product gets activated whenever any of it is reclaimed. The product sliding downwards along the bin walls keeps them clean and prevents formation of product lumps and wall hang-ups.

The discharge hopper acts as a mixer correcting product segregation that took place while filling the bin. This due to the fact that when emptying a mass-flow type bin the product surface keeps its shape down to the transition.

2.2.4 Evaluation of the Mass-Flow Concept

\+ product flow out of the bin is uniform independent of the head of product in the bin, erratic flow and flushing are absent

\+ first-in first-out flow sequence preventing product deterioration due to excessive storage at rest periods

\+ remixing effect for products which segregate on feeding

\+ non-flowing zones are absent as are formation of product lumps and wall hang -ups

\- limited storage volume for a given headroom

\- high investment per m^3 storage volume

→ Mass-flow is the preferred flow pattern. Mass-flow type bins should always be selected in cases

- where segregation is important,
- where product degradation is a problem,
- where feed rate consistency is of prime importance (e.g. for feed bins).

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Figure 6 Mass-flow : typical bin shapes

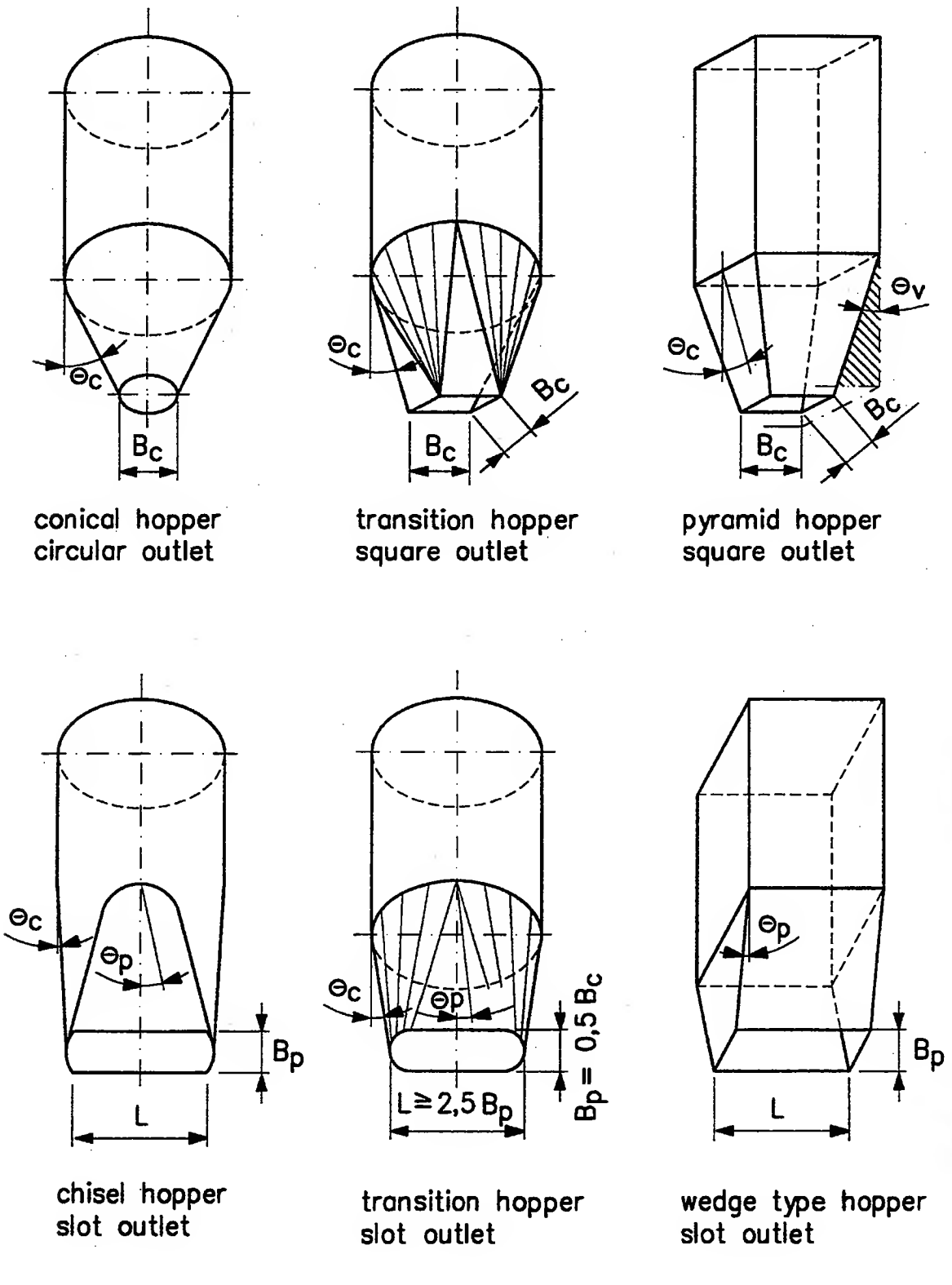
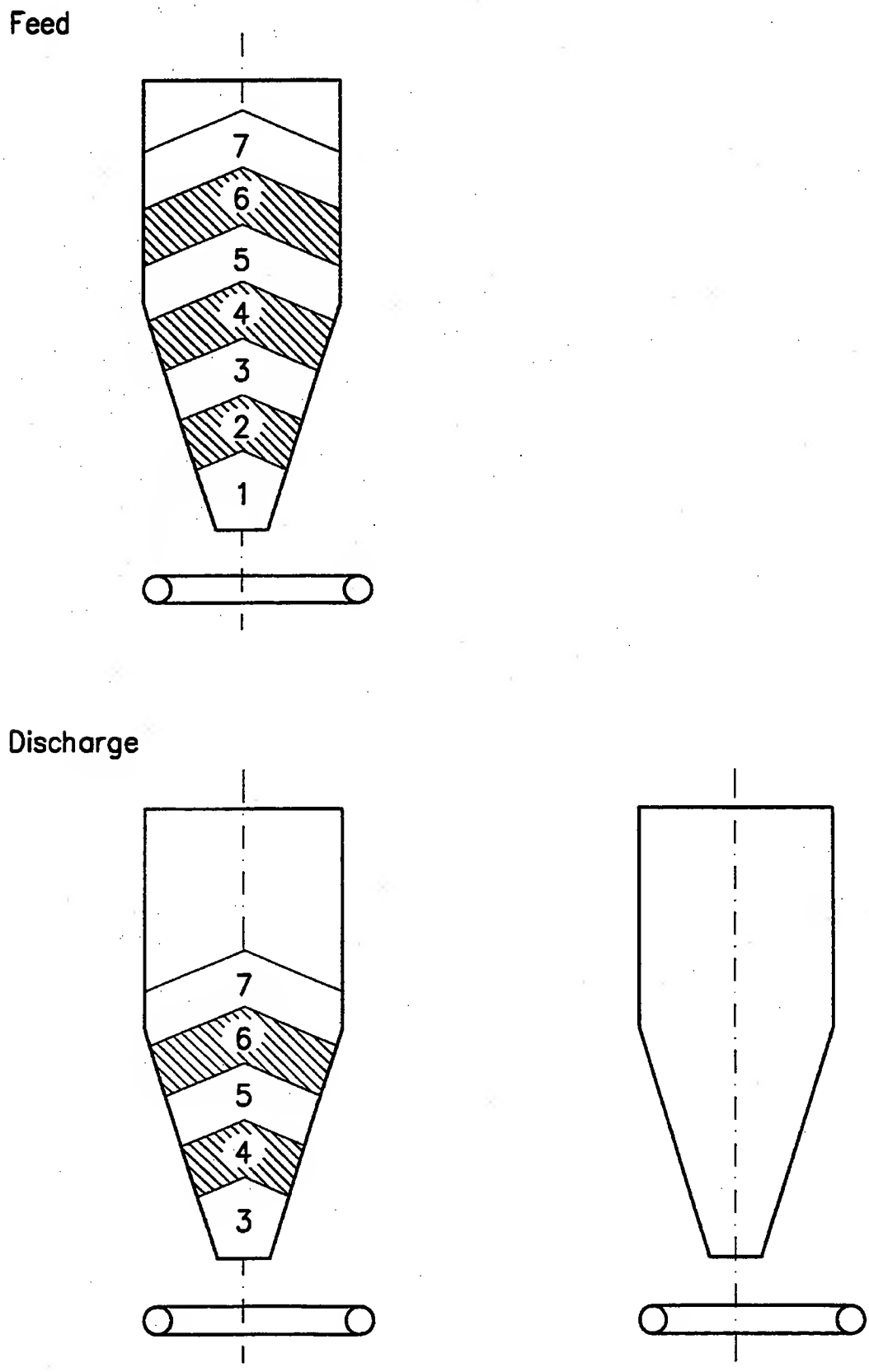


Figure 7 Flow sequence in mass-flow type bins



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2.3 The Expanded Flow Concept

2.3.1 Bin Shapes

The typical shape of an expanded-flow type bin is shown in Fig. 9. From this figure the concept becomes evident: The selected bin discharge configuration is that of a composite hopper with the upper hopper section designed for funnel-flow and the lower section for mass-flow.

2.3.2 Flow Pattern

It is evident that the expanded-flow concept combines both the funnel-flow and the mass-flow pattern. This is a creative way of increasing a bin's storage capacity while ensuring consistent product discharge.

2.3.3 Operation Problems

Consistency of Discharge Rate

Product discharge out of expanded-flow type bins will be consistent in rate and density provided the lower hopper is properly sized for mass-flow.

Interruption of flow: Arching and Rat-holing

Arching and rat-holing will be absent with bins of the expanded flow type provided the lower hopper is properly sized for mass-flow.

Product Degradation

In an expanded-flow type bin's funnel-flow section a core of activated product will form but the surrounding product will remain at rest. With products that tend to consolidation (packing) expanded-flow type bins may not be cleared completely, the formation of product lumps and of wall hang-ups can not be excluded.

Segregation that took place while filling the bin can only partly be corrected flowing through the mass-flow hopper section.

2.3.4 Evaluation of the Expanded-Flow Concept

\+ optimized storage volume for a given headroom

\+ relatively low investment per m^3 storage volume

\+ product flow out of the bins is uniform

\+ partial remixing effect for products which segregate on feeding

\- last-in first-out flow sequence

\- non-flowing zones favoring product degradation with time

→ *Expanded-flow may be useful*

- *where segregation is unimportant,*
- where product degradation is not likely to be a problem,
- as a means to improve the insufficient flow pattern of a given bin.

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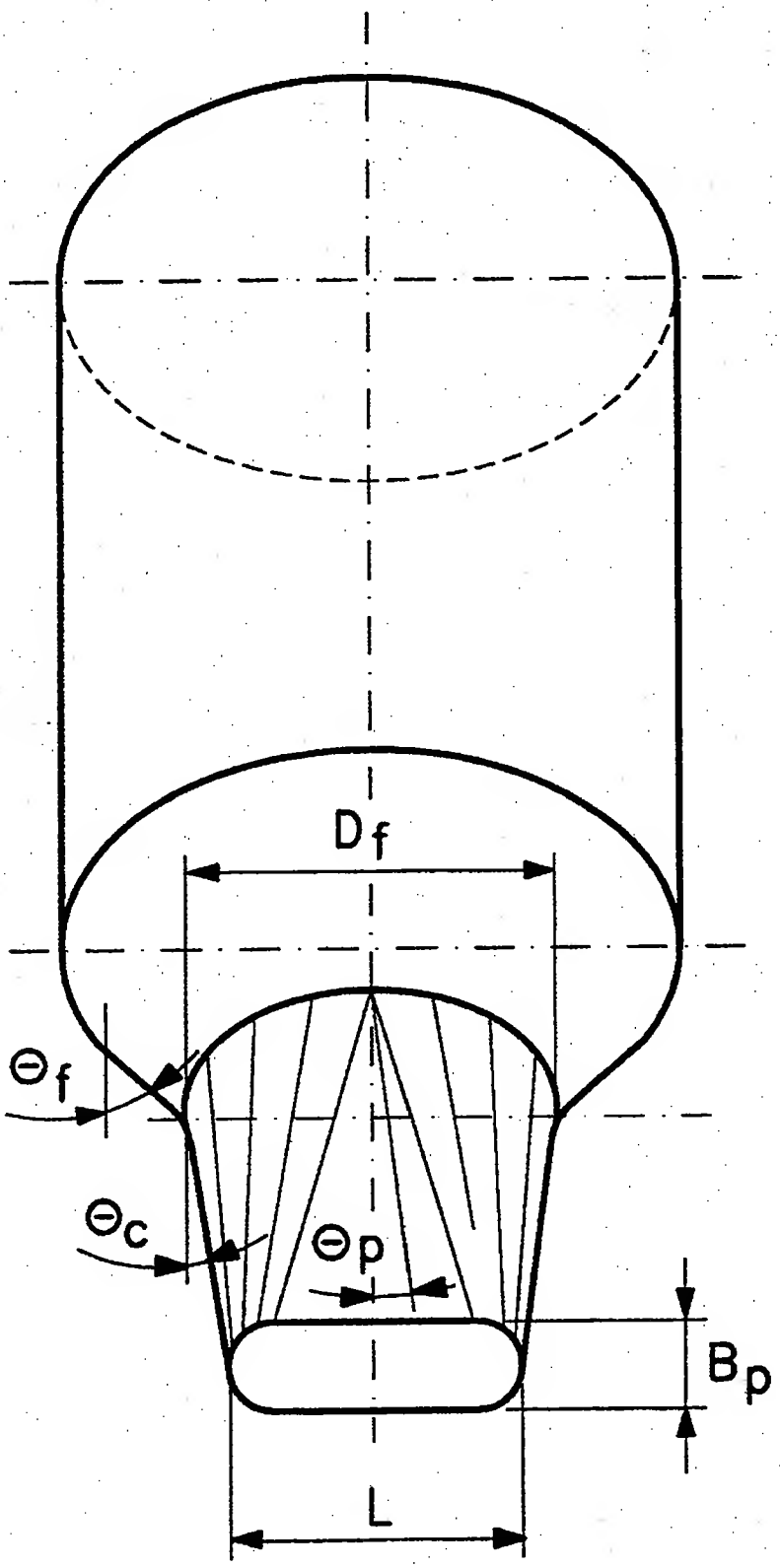
Figure 9 Expanded-Flow : typical bin shape



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3. THE NATURE OF BULK SOLIDS

3.1 Introduction

A bulk solid consists essentially of a multitude of granules (particles) which are randomly grouped to form of a bulk. These granules differ in size, in shape, in hardness, in surface texture, in chemical composition, etc.
The nature of a bulk solid not only depends on the characteristics of its constituent particles but also on factors as its way to compact and get strength (i.e. its cohesion), its angle of repose, its tendency to adhesion, it wall friction, the influence of moisture and storage at rest.
Hence it follows that any characterization of bulk solids must consider two levels

- the *features of the constituent particles*, and
- the *features describing a products bulk form.*

Unfortunately nowadays level of knowledge is still not sufficient as to allow for a reliable prediction of a bulk solids behavior based solely on the characteristics of its constituent particles.

A first step in the characterization of bulk solids is always a qualitative description of its behavior, of its 'personality'. Questions to be asked are

- does it flow easily or is it sticky;
- is it fragile, abrasive, corrosive, explosive, etc.

Such descriptive terms are essential when communicating information on bulk solids. But they are unlikely sufficient as to allow for the design of handling systems which are matched to the product. For this purpose measurable numerical parameters are required which characterize a bulk solid in a quantitative manner. The most important of these parameters will be discussed in more detail in the following.

3.2 Relevant Characteristics of Bulk Solids

3.2.1 Particle Size and Particle Size Distribution (Granulometry)

Qualitative terms commonly used to describe the granulometry of a bulk solid are given in below table.

Table: Descriptive terms for bulk solid granulometry

Descriptive term	Particle Size Range	Example
coarse (crushed) solid	5-300 mm	limestone
granular solid	0.3 - 5 mm	sand
particulate solid:		
- coarse powder	100 - 300 µm	separator grits
- fine powder	10 - 100 µm	cement
- superfine powder	1 - 10 µm	dust collector product
- ultrafine powder	< 1 µm	paint pigments

Above qualitative terms are not sufficiently precise for the description of conglomerates of irregularly shaped and sized particles. Quantitative parameters are required for the description of particle size, size distribution and shape.

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Monosized spherical particles are easy to be described. One single dimension is required, the particle diameter.

For *a conglomerate of spherical particles* of varying sizes the situation is more complex. For the characterization of particle size distribution the definition of an average particle diameter is required together with some information on particle size distribution.

The situation gets worse in case of *industrial, non-spherical bulk solids*. The parameters used for the description of particle size, size distribution and shape need careful definition. It is common industrial practice to determine the granulometry of bulk solids by using one of the sizing techniques given in below table. Selection of the sizing technique depends on the product's particle size range.

Table: Sizing techniques and useful range

Sizing Technique	Useful Range
• dry sieve analysis	50 µm - 100 mm
• wet sieve analysis	10 µm - 100 mm
• particle sedimentation	2 µm - 75 µm
• laser granulometry	1 µm - 600 µm

The results of a sieving test can easily be represented in a *histogram* (Fig. 10.1), i.e. by drawing rectangles over the selected class intervals. The area of each rectangle is proportional to the percentage of particles in that class. A smooth curve through the intervals results in a grain size distribution curve.

An even more useful way for the representation of the analysis data is the *cumulative graph* (Fig. 10.2). In such a graph the particle size is represented along the horizontal axis whereas the cumulative percentage retained or passing is given on the vertical axis. The advantage of this representation is that grain sizes not determined experimentally are reliably predicted as is the *median particle size* (the 50% size).

The median particle size mentioned above is probably the most common term used for representing a bulk solids average particle size. But it has to be noted that there are other definitions for an equivalent diameter such as the mean volume diameter or the mean surface diameter.

- *mean volume diameter*

$$d_{pm} = \left(\frac{6V_{pm}}{\pi} \right)^{\frac{1}{3}} = \left(\frac{1}{N} \sum d_v^{\,3} \right)^{\frac{1}{3}}$$

- *mean surface diameter*

$$d_{sm} = \left(\frac{A_{spm}}{\pi} \right)^{\frac{1}{2}} = \left(\frac{1}{N} \sum d_s^{\,2} \right)^{\frac{1}{2}}$$

Which of above figure is used for characterization of a bulk solid's granulometry depends on the product's industrial use. The median is common in cement industry for characterization of e.g. raw meal and cements, whereas the mean surface diameter is relevant in chemical industry for the characterization of e.g. catalysts.

3.2.2 Particle Shape

The shape of the particles of a bulk solid has proven to be relevant for its packing and flow behavior. But establishing quantitative data on particle shape is discouraging as there is hardly an agreement on how to define shape factors. This above all as defining the shape of non-spherical particles in mathematical terms is not easy.

For the time being the effect of particle shape on a bulk solid's flow behavior can only be based on qualitative observations:

- a fibrous appearance may warn of a tendency to particles interlocking,
- a sharp angular particle shape may indicate that abrasive wear may be excessive.

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Figure 10 Graphical methods of presenting particle size distribution

FIG. 10.1 The Histogram

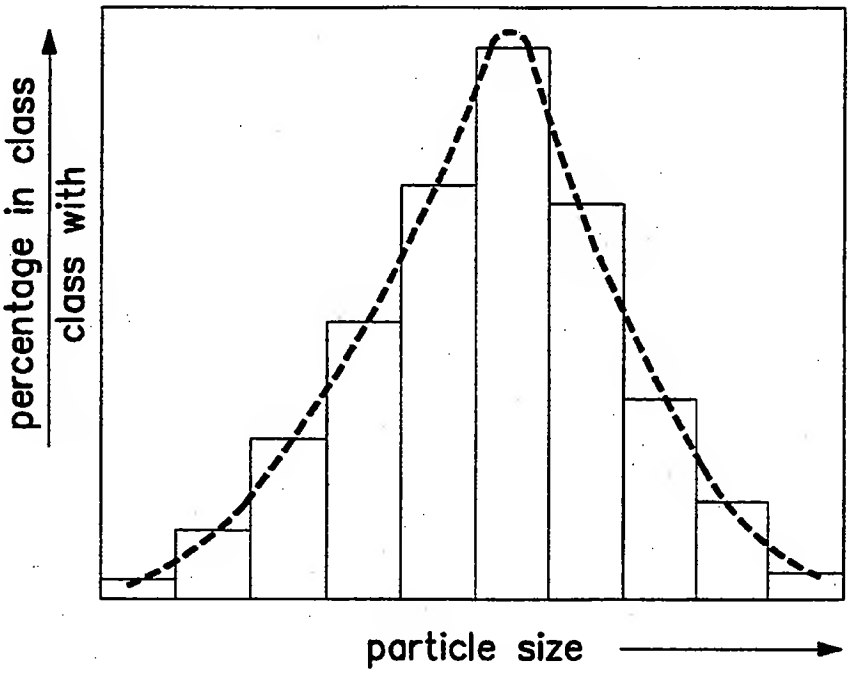
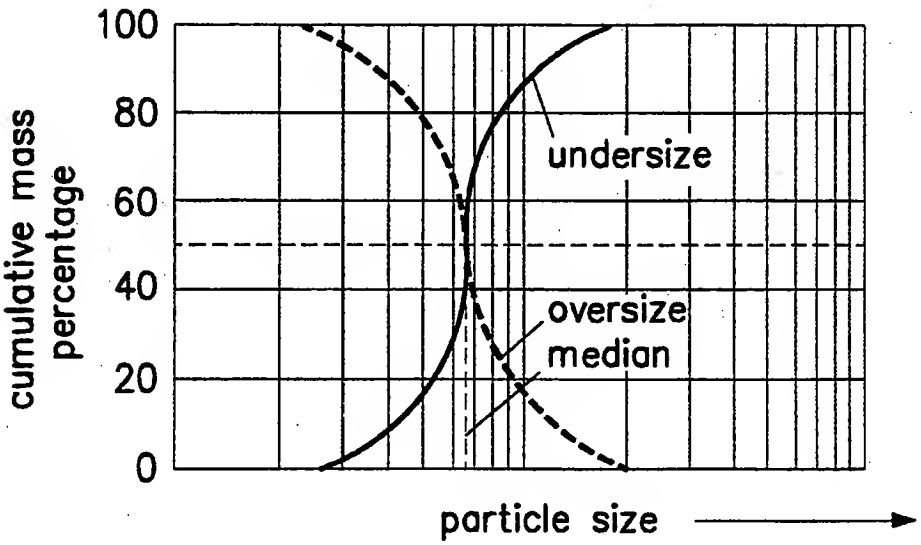


FIG. 10.2 The cumulative percentage graph



3.2.3 Particle Hardness

Knowledge regarding the hardness of the particles constituting a bulk solid is important in that respect that it gives an indication regarding the product's abrasivity. It is common experience that a product is the more abrasive the harder its particles are.

F. Mohs proposed already in 1822 a scale of hardness based on ten standard minerals grouped according a simple scratch test.

Table: Mohs scale of hardness

Mohs scale hardness	Material	Chemical formula	Scratch test
1	Talc	$Mg_3(OH)_2{\bullet}(Si_2O_5)_2$	Very soft, can be powdered with fingers
2	Gypsum	$CaSO_4{\bullet}2H_2O$	Moderately soft, can scratch lead
3	Calcite	$CaCO_3$	can scratch finger nail
4	Fluorite	CaF_2	can scratch a copper coin
5	Apatite	$Ca_5(PO_4)_3(Cl,F)$	can scratch a knife blade with difficulty
6	Feldspar	$KAlSi_3O_8$	can scratch a knife blade
7	Quartz	SiO_2	all products harder than 6 will scratch window glass
8	Topaz	$Al_2F_2SiO_4$	
9	Corundum	Al_2O_3	
10	Diamond	C	

The Mohs scale proofed to be too coarse and insufficiently reproducible. This resulted in the introduction of a number of *indentation tests* above all for metals. The most common hardness figures are

- the Vickers Pyramid Number VPN,
- the Brinell Hardness Number BHN,
- the Rockwell hardness HRC.

There is sufficient knowledge available as to allow for relating the different hardness numbers to the Mohs scale.

3.2.4 Voidage and Bulk Density

Bulk solids are constituted by a bulk of randomly packed particles which differ in size and shape. Bulk solids are thus combinations of particles and void space. The percentage of the volume not occupied by particles is referred to as the *voidage*.

$$\text{voidage, } \varepsilon = \frac{V_{voids}}{V_{particles} + V_{voids}} \times 100[\%]$$

Note that sometimes the term *porosity* is applied with bulk solids as a synonym for voidage. However, it is advisable to reserve this term for the description of the structure of individual particles.

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A quantity of a bulk solids has an apparent density, usually called *bulk density*, which is defined to be the mass of material divided by the bulk volume.

$$\text{bulk density, } \rho_b = \frac{m_{solid} + m_{voids}}{V_{solid} + V_{voids}} \left[t/m^3 \right]$$

Writing ρ_p the particle density and ρ_f as the density of the fluid in the voids it follows another expression for the bulk density:

$$\text{bulk density, } \rho_b = (\sigma_p - \rho_f)(1-\varepsilon) + \rho_f \left[t/m^3 \right]$$

It is important to make a clear distinction between bulk density and particle density. For a single particle the density is defined as its mass divided by its volume. Thus, determination of a particle's density involves the measurement its mass and the volume occupied. For large particles the density can easily be determined by measuring the volume of liquid being displaced in a measuring cylinder.

Determination of a product's bulk density is more tricky. For coarse products, again the liquid displacement method can be used whereas for powders the use of a specific-gravity bottle or of an air-compression pycnometer is required.

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3.2.5 Angle of repose

A bulk solid's angle of repose is defined to be the angle of an exhibited sloping product surface which forms e.g. when a product is piled or after slippage from a product pile. Quantitative determination of the angle of repose is discouraging in that respect that no unequivocal method is available. One has to accept that for the same product as many different angles of repose may be measured as measuring methods are used. Some of these methods are illustrated in Fig. 12 .

Besides the test procedure adopted and the skill of the operator the angle of repose depends upon the nature and condition of the bulk solid concerned (moisture, granulometry, electrostatic charging, storage at rest period).

Below table gives a qualitative evaluation of bulk solid flowability based on the angle of repose.

Angle of repose	**Flow behavior**
20 - 30 deg	free flowing
30 - 38 deg	easy flowing
38 - 45 deg	fair flowing
45 - 55 deg	cohesive
> 55 deg	very cohesive

The angle of repose should not be abused as a quantitative guide in bin design. But it may be a useful indicator for the contours of product storage piles, the cross-sectional area of a product bed on a belt conveyor, etc.

3.2.6 Cohesion and Adhesion

Flowability of a bulk solid is function of the forces of attraction or the 'cohesion' between its constituent particles.

In case the interparticle forces are low the bulk solid will easily flow with the single particles moving as individuals relative to one another. Dry sand or clinker are familiar examples from cement industry.

Interparticle cohesive forces are typically caused by moisture and electrostatic charging. The effect of such forces is especially pronounced with products containing a considerable portion of fines. Familiar examples for cohesive products in the cement industry are clay, natural and synthetic gypsum, fine coal, fly-ashes, etc. When cohesion is defined to be a bulk solid's resistance to shear under a given compressive (normal) stress quantitative measurement becomes possible. For details see paragraph 'Shear Strength'.

Flowability of bulk solids is not only concerned with cohesion but also with 'adhesion' on boundary surfaces such as the walls of a bin discharge hopper, the bottom and side surfaces of transfer chutes, etc. Whereas cohesion reflects the effects of interparticle attractive forces, adhesion describes a bulk solids tendency to stick on a boundary surface. The adhesions between a bulk solid and a type of surface material can quantitatively be measured using a test similar to the shear test mentioned above for cohesion. For details see paragraphs 'Shear Strength' and 'Wall Friction'.

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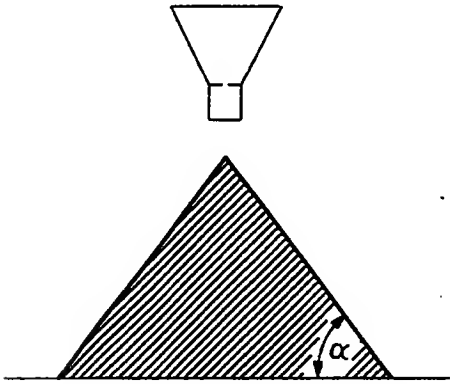
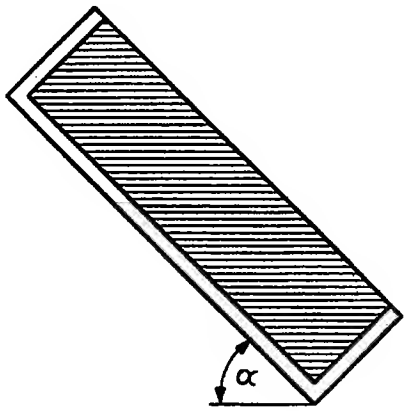
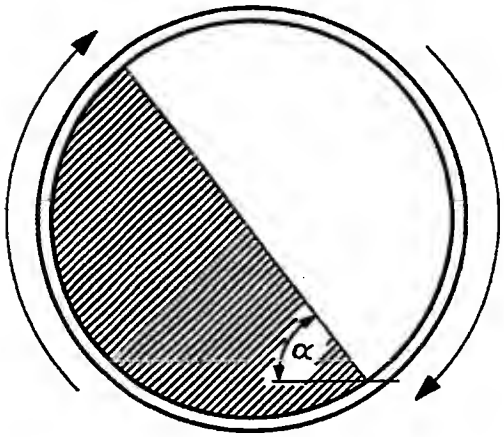
Figure 12 Methods of measurement of angle of repose



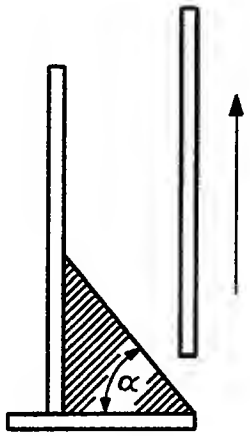
Heap poured on flat surface



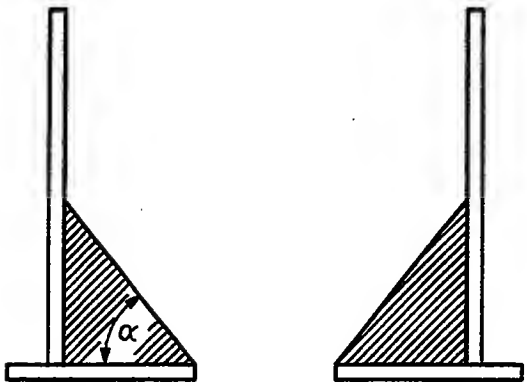
Tilting table



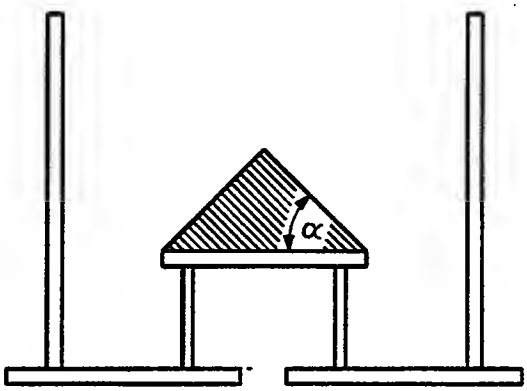
Rotating cylinder



Box with removable side



Cylinder with hole in base



Heap on circular platform

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3.2.7 Shear Strength

As with continuos material, the application of any force to a bulk solid tends to cause a shear deformation and will result in an opposing resistive force. With increasing magnitude of the applied force, a point will be reached where the bulk solid begins to deform with the constituent particles sliding relative to another.

For illustration let's consider the following simple experiment (Fig. 13) :

- Prepare a bulk solid specimen in a cylindrical mold (cross section A) by compacting the product applying a defined normal load F_N. The cylinder wall is assumed to be free of friction.
- Remove the cylindrical mold.
- Expose the specimen exposed to an increasing normal load (e.g. by filling a bottle of water). At a distinct load level which is termed *shear* or *product strength* F_C, the specimen will collapse and shear. The shear strength is determined just by weighing the bottle.

From this simple experiment it follows the existence of a *flow limit.* This flow limit is specific for each bulk solid and function mainly of the magnitude of the applied consolidating force. But it is also affected by factors as the particles nature (as e.g. their shape and packing arrangement), moisture content, thus the pre-history of the product.

Product strength can (unfortunately) not be determined directly with sufficient accuracy using above described simple experimental concept but rather needs to be derived from the results of so called shear tests. The concept of shear testing is to make specimens of a bulk solid shear under varying normal loads while measuring the appearing shear forces.

The result of such a test is commonly represented as a so called *yield locus* YL, i.e. as a plot of the shear strength S_f versus the compressive (normal) force F_N. A yield locus is thus a border line separating the range of stresses which cause no permanent deformation from the range of stresses causing failure and plastic flow.

Interpretation of a yield locus plot involves the use of Mohr circles of stress. From such a plot (Fig. 14) the following characteristics can be derived:

- drawing a Mohr circle tangential to a yield locus and passing through its end point gives the major principal stress σ_1, representing the initial state of sample consolidation;
- the Mohr circle drawn through the diagram's origin and tangential to a yield locus gives the unconfined yield stress σ_c, which represents a product strength at a free surface;

  (Note that the unconfined yields stress for non-cohesive, free flowing bulk solids is equal to zero. Hence it follows that the yield locus representing such products pass through the diagram's origin!)
- the slope of the yield locus at any point defines the *dynamic angle of internal friction* ϕ, which in practice is somewhat varying due to the typically slight curvature of the yield locus,
- the straight line tangential to the Mohr circle for the initial consolidating load and passing the diagram origin is termed the *effective yield locus* with its slope defining the *effective angle of internal friction* ϕ_e.

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Figure 13 Testing a bulk solid for flow

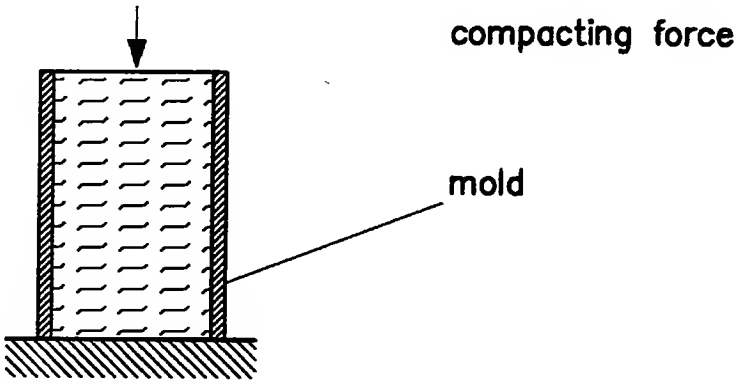

FIG. 13.1 Preparation of specimen

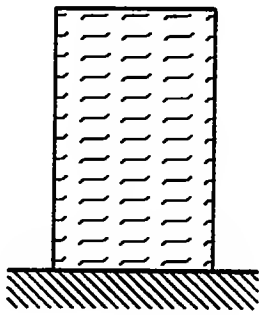

FIG. 13.2 Remove mold

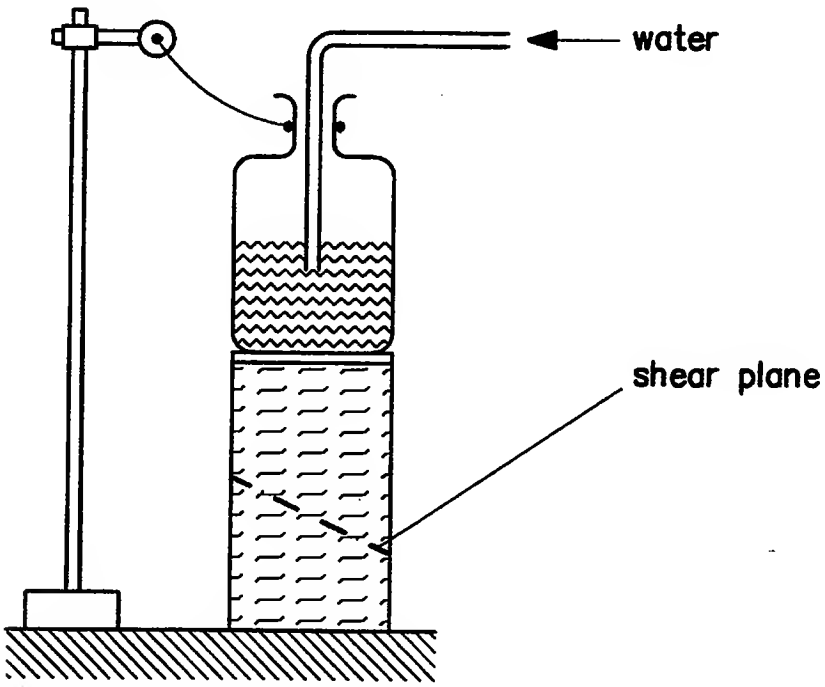

FIG. 13.3 Shear test

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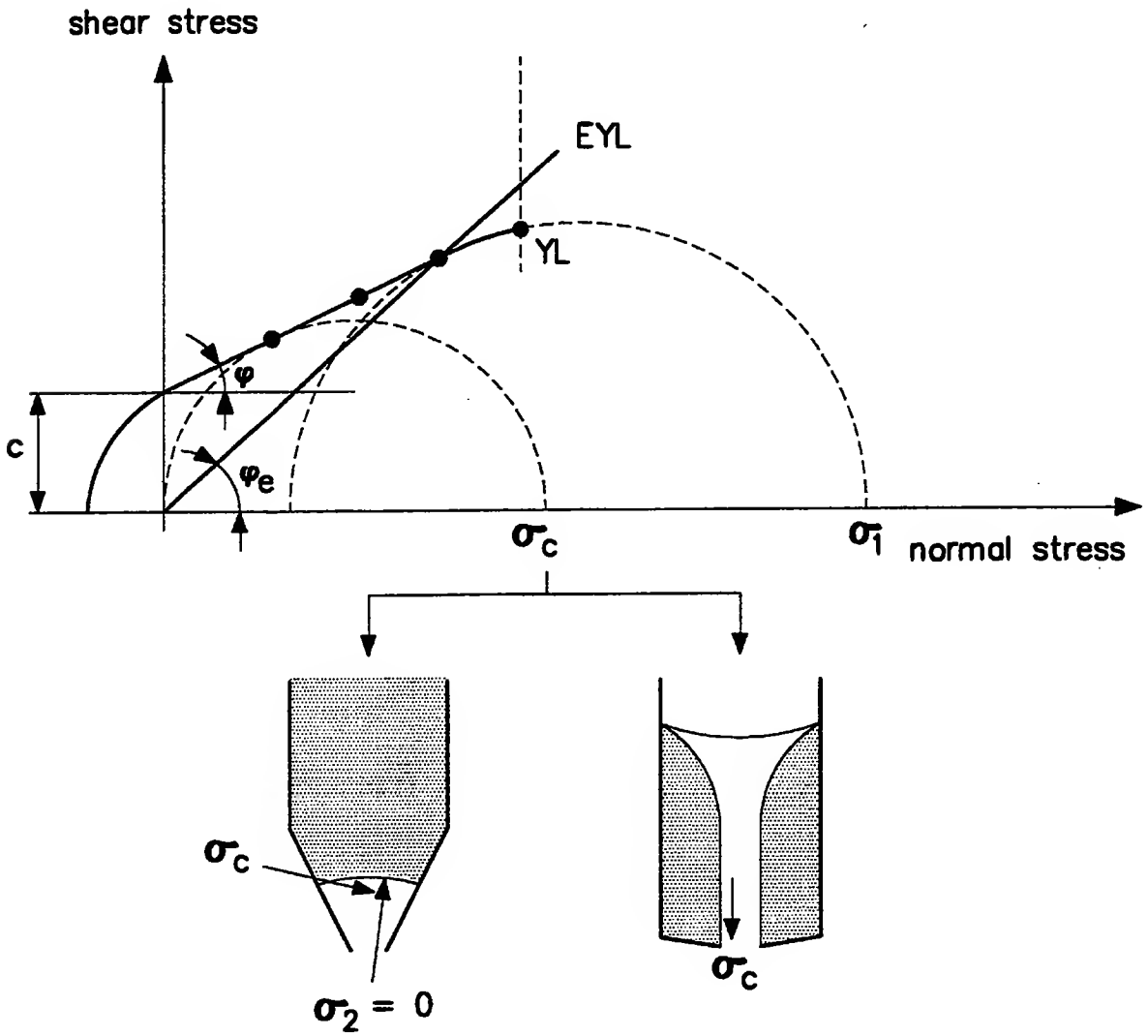
Figure 14 The yield locus



YL	measured yield locus
EYL	effective yield locus
σ_1	major compressive stress representing the samples state of consolidation
σ_c	unconfined yield stress representing product strength
φ	dynamic angle of internal friction
φ_e	effective angle of internal friction
c	cohesion

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It has already been mentioned that a bulk solid's shear strength is affected by its state of consolidation as well as its pre-history. As a consequence a single yield locus is not a sufficient characteristic to describe a bulk solid's flow behavior as it is for a Coulomb type solid. For a full investigation of its flow properties specimens must be tested for different levels of product consolidation, i.e. a *family of yield loci* YL_i (Fig. 15) must be produced.

As already mentioned the shear strength of bulk solid's may be determined by means of a shear-testers. More than one type of shear testers have been developed with probably the *Jenike translational shear tester* being the most familiar one. For any experimental arrangement designed for testing the flowability of bulk products it is a prerequisite that the complicated stress conditions can be simulated which are imposed on bulk products while being filled into bins, during storage at rest in bins and when being discharged from bins.

Jenike's experimental technique meets these requirements. The set-up of such a tester (Fig. 16) is very simple: Two rings of equal diameter form the shear cell. Samples of the product under investigation are poured into such cells, are precompacted to different degrees of consolidation and made shear under varying normal loads. It is important that a consistent procedure for sample preparation and testing is followed. Jenike's translational shear tester is subject to a number of limitations as its limitation to fine products or to a product's fine portion only (< 3 mm), its limited translational displacement as well as the complicated testing procedure.

The *torsional shear tester* arrangement as it is used at HMC's laboratory (Fig. 17) allows for a simplified testing procedures as it allows for unlimited strain. With such equipment the torques required to make the specimens shear are measured. The respective shear stresses required for the representation of the yield loci YL_i can be calculated.

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Figure 15 The flow function

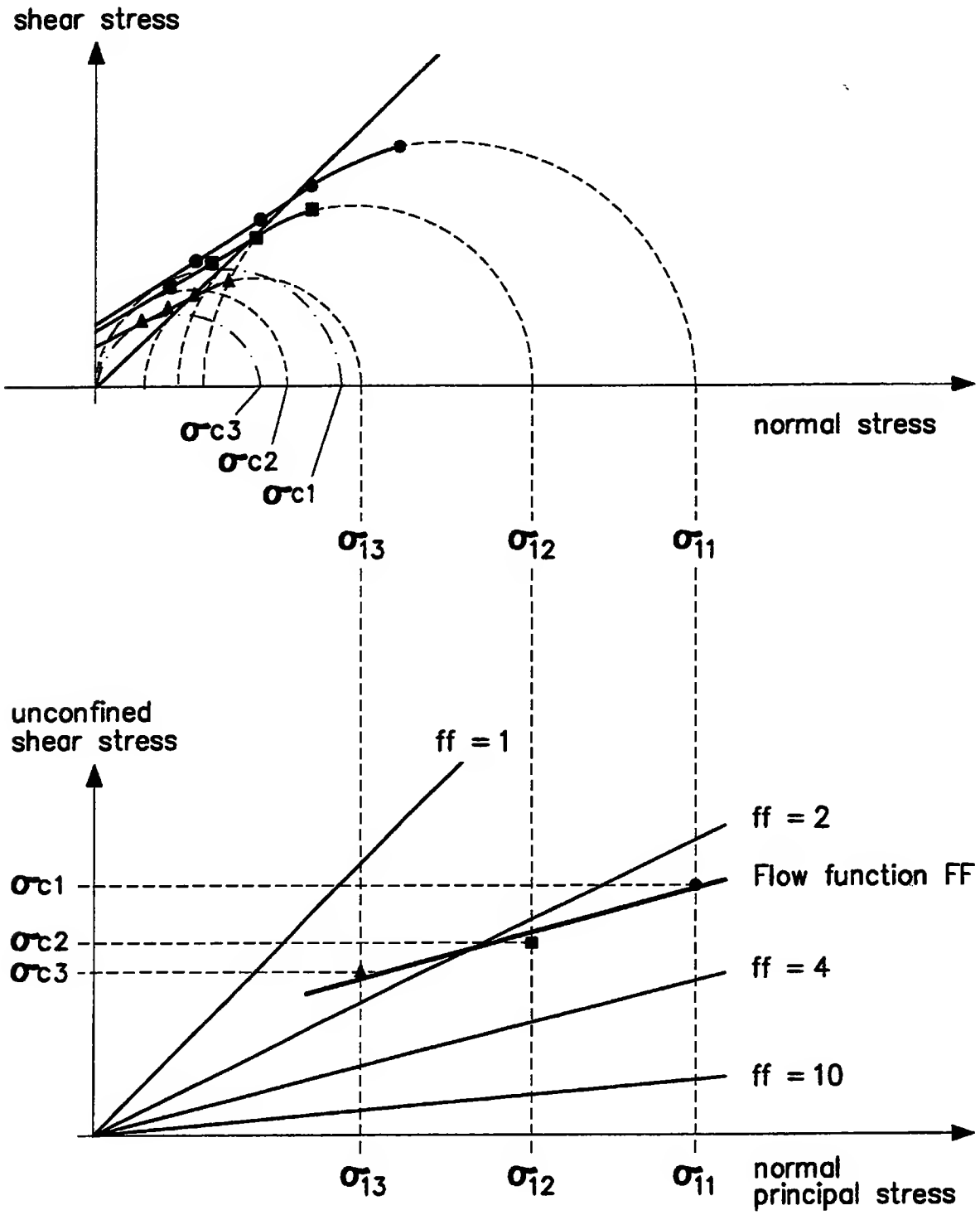

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Figure 16 Shear test concepts

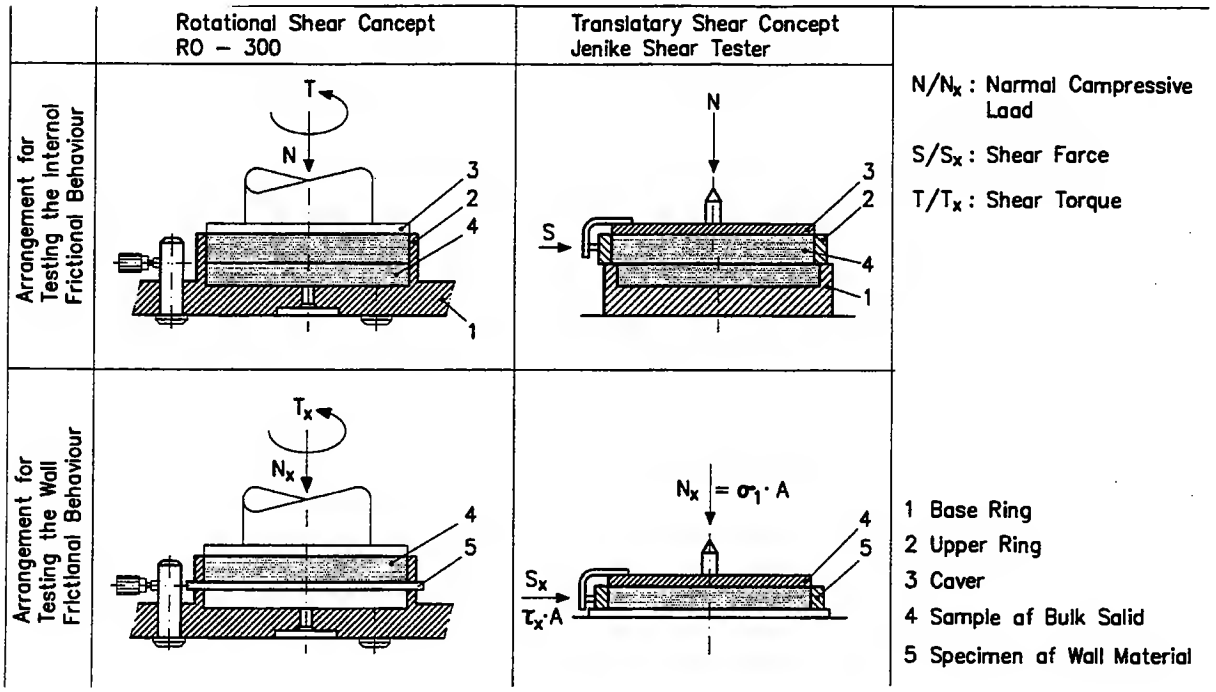
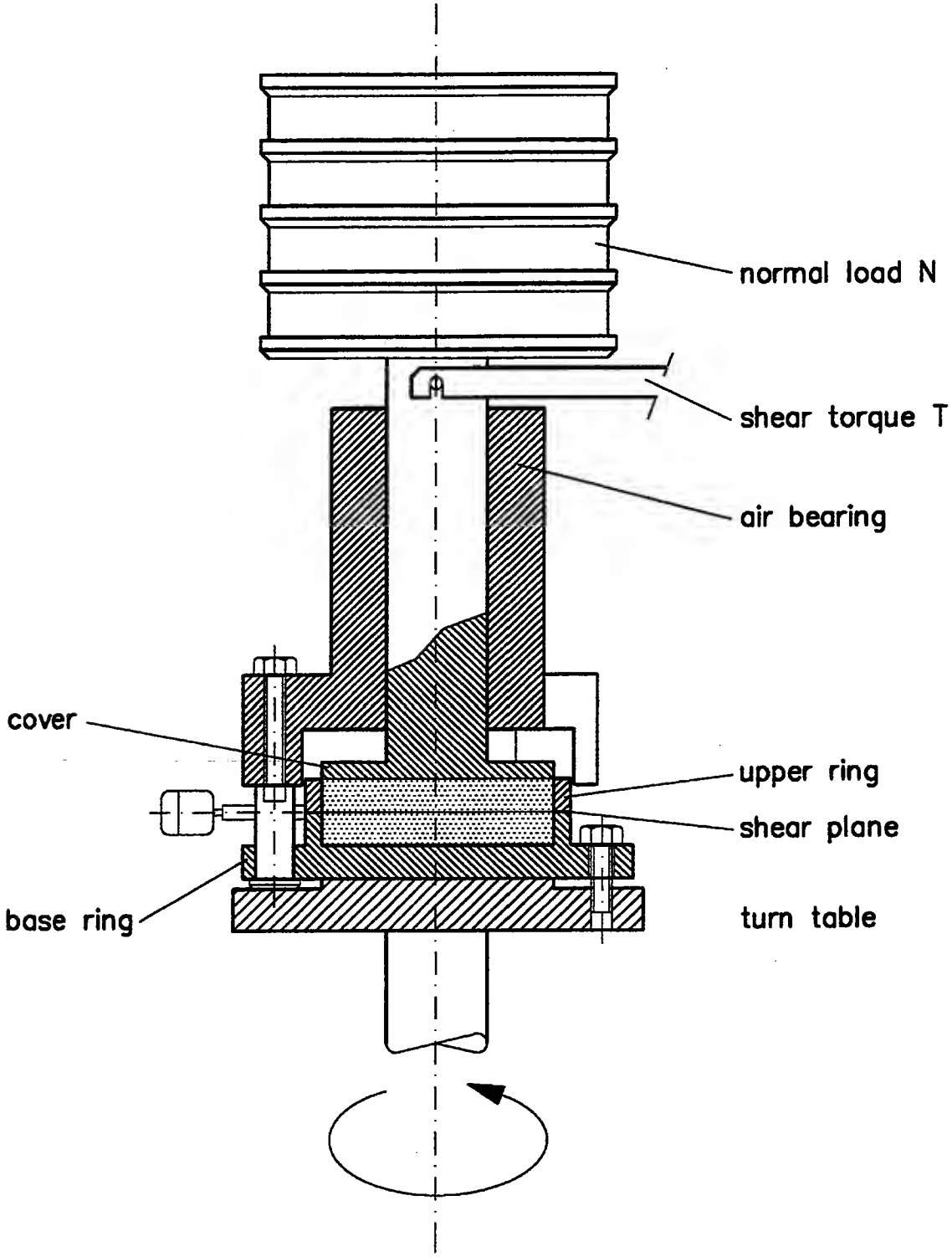


Figure 17 The torsional shear tester



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3.2.8 The Flow Function

A bulk solid's flow behavior is commonly characterized by its *Flow Function* which is defined as the ratio of consolidating principal stress versus product strength

$$FF = \frac{\sigma_1}{\sigma_c}$$

Hence it follows that a bulk solid is flowing the easier the higher its flow function is. The flow function allows for a rough evaluation of a product's flow behavior

FF < 1	non-flowing product
1 < FF < 2	very cohesive product
2 < FF < 4	cohesive product
4 < FF < 10	easy flowing product
10 < FF	free-flowing product

Above limits are represented in Fig. 15 . Commonly the Flow Function and with it a bulk solid's flow behavior are function of the state of consolidation. In most cases the value for the Flow Function FF increases with increasing consolidating stress σ_1. Hence it follows that the flow behavior of various bulk solids may only be evaluated based on the Flow Function figures in case all measurements have been carried out on specimens prepared the same way (i.e. having the same state of consolidation).

3.2.9 Wall Friction

Bulk solid flow along a hopper wall or in a chute results in wall friction. Wall friction is function of the wall material characteristics (adhesion, smoothness of surface), the wall inclination, etc.

A linear model similar to that the proposed for the internal friction of bulk solids can be used to represent the relationship between the normal force F_N pressing a particulate product against a constraining surface and the shear force S_W required to cause the product to slide along that surface

$$S_w = \mu_w F_n + C_w$$

where μ_w is the coefficient of wall friction

C_w is a coefficient defining the adhesion between bulk solid and containing surface

The line defined by above equation is termed *wall yield locus* WYL. For most bulk solids it is found below the yield locus YL and the angle of wall friction ϕ_w being less than the angle of internal friction ϕ.

To investigate a bulk product's frictional behavior on bin walls the shear testing concept is slightly modified. The lower ring of the shear cell is replaced by probes of possible bin wall materials. The testing procedure is the same as described above: The forces or the torques respectively which are required to make the sample shear under varying normal loads are measured. Again Wall Yield Loci (WYL_i) can be produced when plotting the shear stresses versus the corresponding normal loads.

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3.2.10 Moisture Content

Product moisture can have a dramatic influence on a bulk solid's flow behavior in addition to cause effects such as chemical change, deterioration of quality, etc. Moisture analysis is therefore an important task in characterization of bulk solids.

A common expression for a bulk solid's moisture content is the percentage of water to wet solid:

$$\text{moisture content} = \frac{m_{H_2O}}{m_{wet}} \times 100[\%H_2O]$$

It has to be noted that water presence in bulk solids may be in two different forms

- as *surface water* present on the surface of the particles only,
- as *inherent moisture* resulting from water being crystallized within the structure of the particles.

The most common method for determining a bulk solid's moisture content is still by determining the loss in weight for a product sample when drying it in an oven until its weight remains constant. The moisture content can then be calculated as

$$\text{moisture content} = \frac{m_{wet} - m_{dry}}{m_{wet}} \times 100[\%H_2O]$$

When testing the surface moisture of bulk solids containing crystallized water in the structure of particles (e.g. gypsum, coal, etc.) the drying temperature must be selected that low that product dehydration do not occur.

Recently faster analysis methods have been introduced for moisture determination such as infrared absorption, microwave absorption, nuclear magnetic resonance, ultrasonic, etc.

With regard to a bulk solid's flow behavior commonly two moisture limits can be observed

- a lower limit where flowability will deteriorate resulting in an increasing product strength, a reducing discharge rate, an increased tendency to arch, rat-hole and hang-up formation,
- an upper limit where the bulk solid is saturated (all voids filled) and flowability improves again.

The dramatic effect of moisture on a bulk solids flow behavior has already been mentioned. Testing the product strength at different moisture levels is considered an important task in bulk solid's characterization, i.e. that additional yield loci (Fig. 18) should be produced at the moisture levels observed in industrial live.

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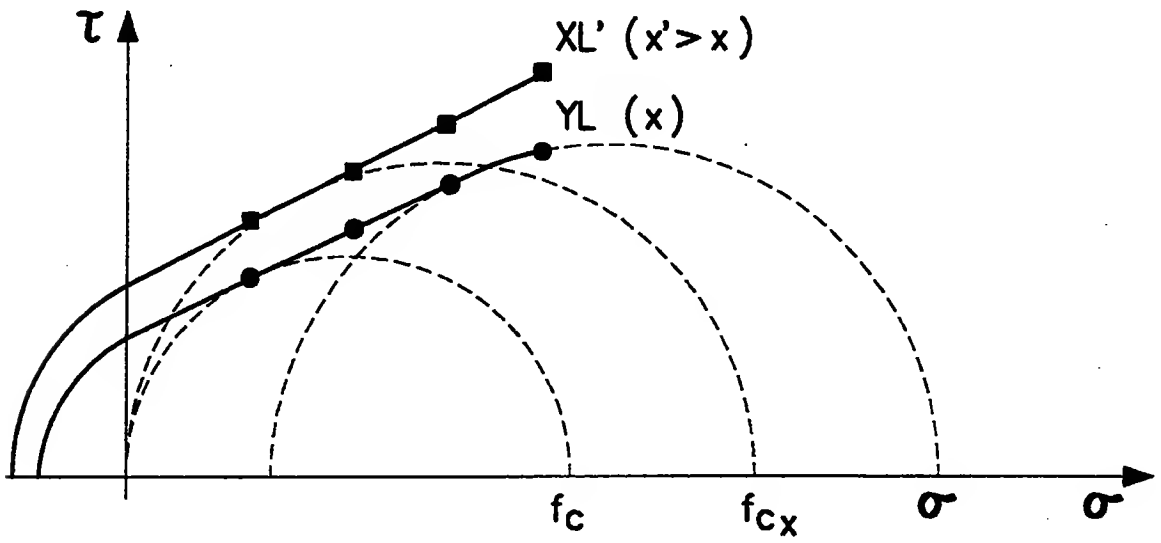
Figure 18 The influence of moisture on the flow behaviour of bulk solids



x = moisture level

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4. DESIGN OF BIN/FEEDER SYSTEMS

4.1 Analysis for Flow

4.1.1 Theories

The theoretical work concerning the flow behavior of bulk solids in bins has been initiated by the already mentioned A.W. Jenike. He adapted theory and experimental technique of soil mechanics to the problem of bulk solid handling. He postulated that bulk solids behave similar as Coulomb solids. Their flow behavior is thus characterized by limiting stress envelopes, so called Yield Loci (YL_i) which separate the range of stresses that will cause no permanent deformation from the range of stresses that will cause failure and plastic flow. The flow behavior of bulk solids depend on their pre-history, i.e. their consolidation, moisture take-up, storage at rest, etc. Hence it follows that the flowability of bulk solids can be characterized sufficiently by a family of yield loci only. This in contrast to Coulomb type solids for which a single yield locus is a sufficient characteristic.

Following Jenike's theory bin/feeder systems can be designed for consistent operation. Precondition is that the design prevents any formation of obstructions to flow.

4.1.2 The flow - no flow Criterion

Consider a mass-flow bin. The bulk solid is unconsolidated when it is fed into a bin but becomes consolidated with continuing filling, i.e. with increasing filling height. In the cylindrical bin section a *static* or *peaked stress field* develops with the consolidating stress σ_1 varying from zero at the top surface towards a maximum near the transition to the converging hopper section. At this transition the stress field changes its orientation from peaked (static) to arched (dynamic). This results in an abrupt increase in stress level to a peak value. Below the transition in the bin discharge hopper the consolidating stress σ_1 decreases within the prevailing *dynamic (arched) stress field* towards zero at the virtual apex of the hopper. Development of the consolidating stress within a bulk solid handled in a bin is shown in Fig. 19 .

The compressive stress σ_1 causes consolidation (packing) of the bulk solid handled in the bin. The product get strength. Strength development of a bulk solid in a bin may be represented by the unconfined yield stress σ_c (Fig. 19). It shows a profile similar to that of the consolidating stress σ_1. Note the product strength at the top surface as well as at the discharge hopper's virtual apex will be zero for free-flowing bulk solids only but show a positive value for cohesive bulk solids

Imagine now that a cohesive arch has formed at some point of the converging hopper. In effect the bulk solid consolidation in the bin is sufficient that a stable structure can form which spans the bin outlet, supports the weight of above product column and transfers it to the wall of the hopper. Experience teaches that the *bearing stress* σ_a of a cohesive arch is proportional to its span D_a. I.e. that the bearing stress σ_a will linearly increase for the bin's discharge hopper section but remain constant for its cylindrical section. A graphical representation of the relation bearing stress σ_a versus arch span D_a is shown in Fig. 19 .

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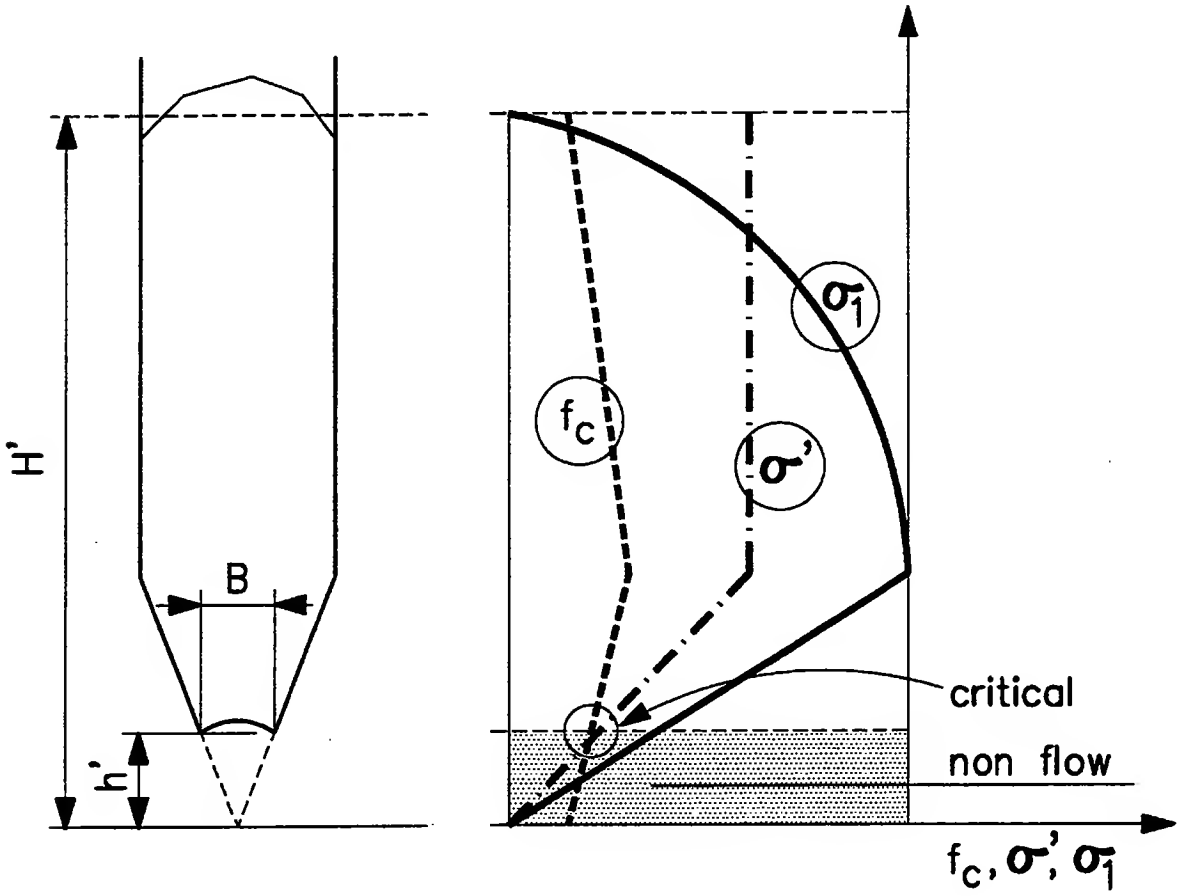
Figure 19 Analysis for flow



$\sigma' > f_c$

The lines representing the arch bearing stress σ_a and the unconfined yield stress σ_c are intersecting. The intersection defines a critical dimension for formation a cohesive arch:

- for positions above the intersection bulk solid strength is not sufficient to support a stable cohesive arch,
- for positions below the point of intersection stable cohesive arches will form as the bulk solid has sufficient strength to support a stable arch.

Above observation allow the formulation of a flow - no flow criterion:

A bulk solid will flow as long as its cohesive strength is not sufficient to support the load of above product column or $\sigma_c < \sigma_a$

For a bin discharge hopper both the bearing stress σ_a of a cohesive arch and the consolidating stress σ_1 are linear functions of the span width D_a. Furthermore each of these stresses approaches zero at the hopper's virtual apex. Hence it follows that the ratio σ_a / σ_1 is a constant for a given hopper geometry. This ratio is commonly called a hopper's *flow factor* ff characterizes a hopper's manner of discharging the contained bulk solid. Note that flow factors have been calculated by Jenike for a wide range of hopper types.

Flow factor ff and flow function FF (= σ_c / σ_1, i.e. the ratio of unconfined yield stress versus consolidating stress) will intersect when plotted in a common stress diagram (see Fig. 20+15). The intersection represents again critical flow condition what allows for a further formulation of the flow - no flow criterion:

A bulk solid will flow at all conditions for which the FF line lies below the ff line or FF < ff

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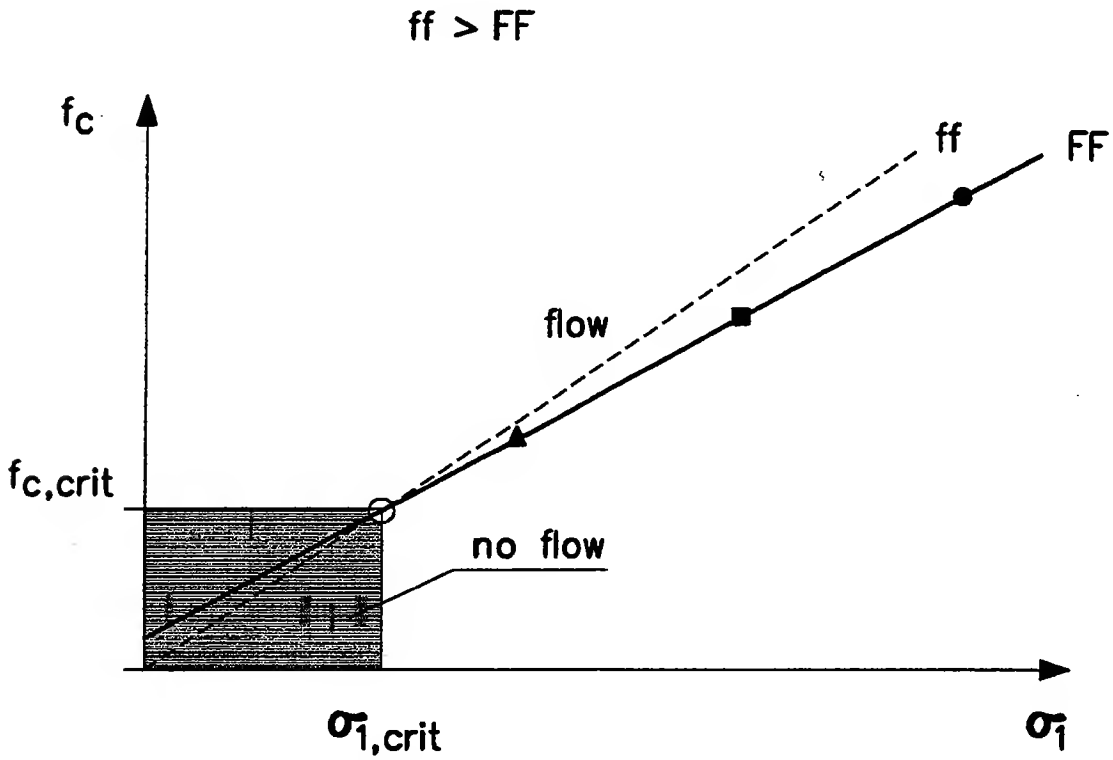
Figure 20 The flow - non flow criterion



4.2 Design Criteria for new Bins

4.2.1 Design Concept

The design of bins and silos can be based on two concepts, either

- on the mass-flow concept, or
- on the funnel flow concept.

An evaluation of both concepts (see Fig. 21) shows clear operation advantages for the mass-flow concept but also its limited qualification for the storage of huge product volumes.

The funnel-flow concept is the preferred concept for large volume stores. It is considered an acceptable selection in cases

- where non-cohesive (free-flowing) products are involved,
- where product segregation is unimportant,
- where product degradation is not likely to be a problem,
- where feed rate consistency is not of prime importance.

The mass-flow concept is the preferred concept for feed bins. But this concept is also recommended for storage purpose

- with cohesive products,
- where qualitative consistency of the reclaimed product is important,

There are some general principles to be observed when designing bulk solid handling systems:

Rule 1: The demands on a large volume store are different from that on a feed bins where reliable product reclaim is prerequisite.

> As a consequence storage and process feed functions should be separated whenever possible.

Rule 2: Feed bins should be designed for reliability of reclaim. This is much easier for small than for large feed bins.

> As a consequence feed bins should be designed for the lowest possible volume but never be abused for additional storage capacity.

Rule 3: Optimum process operation and consistency of product quality is function of consistent and reliable rate control of the components.

> To meet this aim feed bins should be designed for mass-flow on principle.

Bin dimensions

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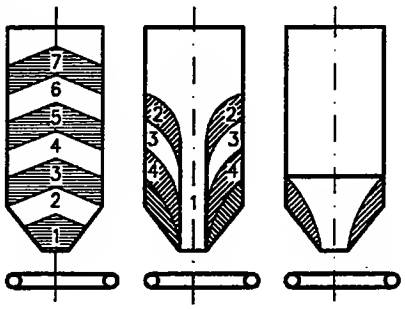
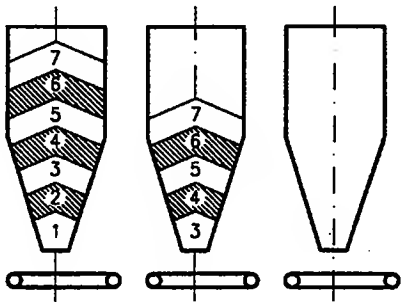
Figure 21 Funnel flow versus Mass - flow

	Funnel–flow	Mass–flow
Bin shape	Compact flat hopper	Slender steep hopper
		
Flow sequence	First–in Last–out	First–in First out
Discharge order	Loading order inversed	As loaded
Silo utilisation	partial only	complete
Handling problems		
– discontinuous flow	arching, rat–holing	arching possible
– no flow	erratic flow, flushing	–
– lump formation hang–ups	possible	–
– product degradation	possible	–
– segregation		counteracted

Precondition for mass-flow to develop in a bin are

- a discharge hopper which is sufficiently steep as to initiate product flow along its walls,
- a hopper outlet which is sufficiently wide to prevent formations of any obstructions to flow,
- a hopper outlet that is fully active.

Comment:

- Dimensioning of new bins and silos should always be based on the tested flowability characteristics of the product in question independent of the selected bin design concept (mass-flow or funnel-flow). Worst product conditions should be considered when designing the shear test program (e.g. moisture levels valid for the wet season).
- For a given product the maximum permissible hopper half-angle θ_c is smaller for a conical hopper than the half-angle θ_p for a plane flow hopper. In addition the required minimum opening width D_c for a conical hopper is typically twice as wide as the minimum opening width D_p for a wedge type hopper. Hence it is clear that the plane-flow hopper concept makes better use of available headroom than conical hopper concept.
- For cohesive (sticky) products reclaim at low rates just by gravity may not be possible as full activation of the required wide outlets is not ensured. In such cases the application of appropriate discharge aids must be considered in the Bin/Feeder concept.

4.2.2 Product Reclaim Systems

Reliable operation of any rate control device depends on consistency of bulk solid supply out of the feed bin. Feeders are typically designed for two functions

- as a discharge aid which activates the bin outlet, and
- as a means of controlling the feed rate to the following process.

Experience teaches that arching and rat-holing may occur even with bins that have been designed for mass-flow according to the rule of (Jenike's) art. This kind of problems can typically be related to partial activation of the bin outlet, resulting in subcritical dimensions of the activated area and causing erratic or even interrupted flow. There are two reasons for partial bin outlet activation

- a closed design of the bin/feeder interface,
- a low discharge rates with cohesive bulk solids.

Bin/feeder interfaces are designed as closed as possible as to prevent spillage and facilitate housekeeping. With such designs product will be activated only in a narrow zone at the rear end of the feeder as that is the only area where free volume is created for the product to flow in (Fig. 22). Activating a bin outlet means that free volume must be created over the outlet's full length. This can be achieved by expansion of the bin/feeder interface as shown in Fig. 23, 1-3, or by means of live bottom feeders.

For cohesive (sticky) products reclaim at low rates just by gravity may not be possible as full activation of the required wide outlets is not ensured. In such cases the application of appropriate discharge aids must be considered as aeration systems for dry powders, vibrating cages for cohesive products, rotary extractors for sticky products

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Figure 22 The problem of the partial outlet activation

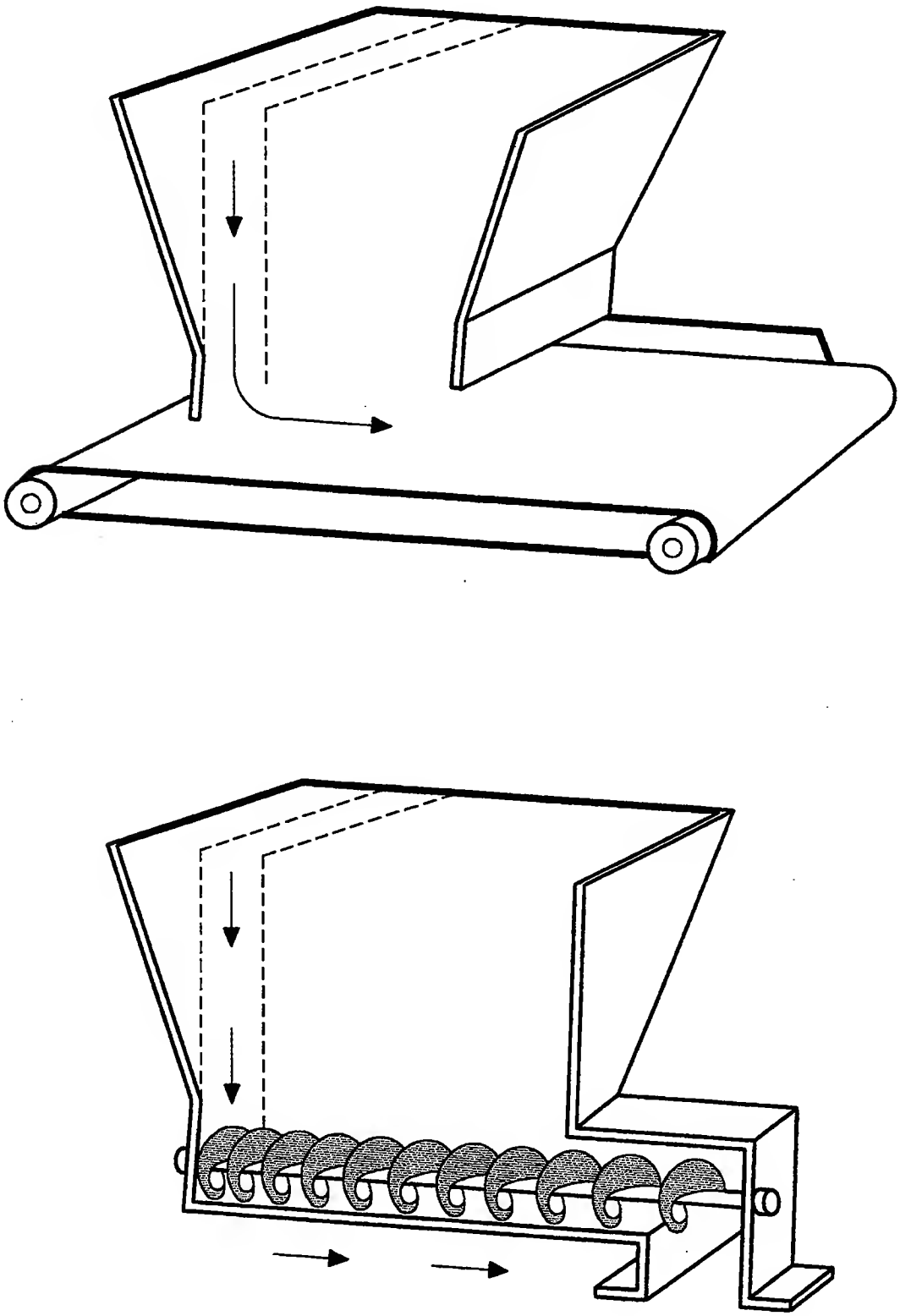

Figure 23.1 The concept for full outlet activation: the expanded outlet

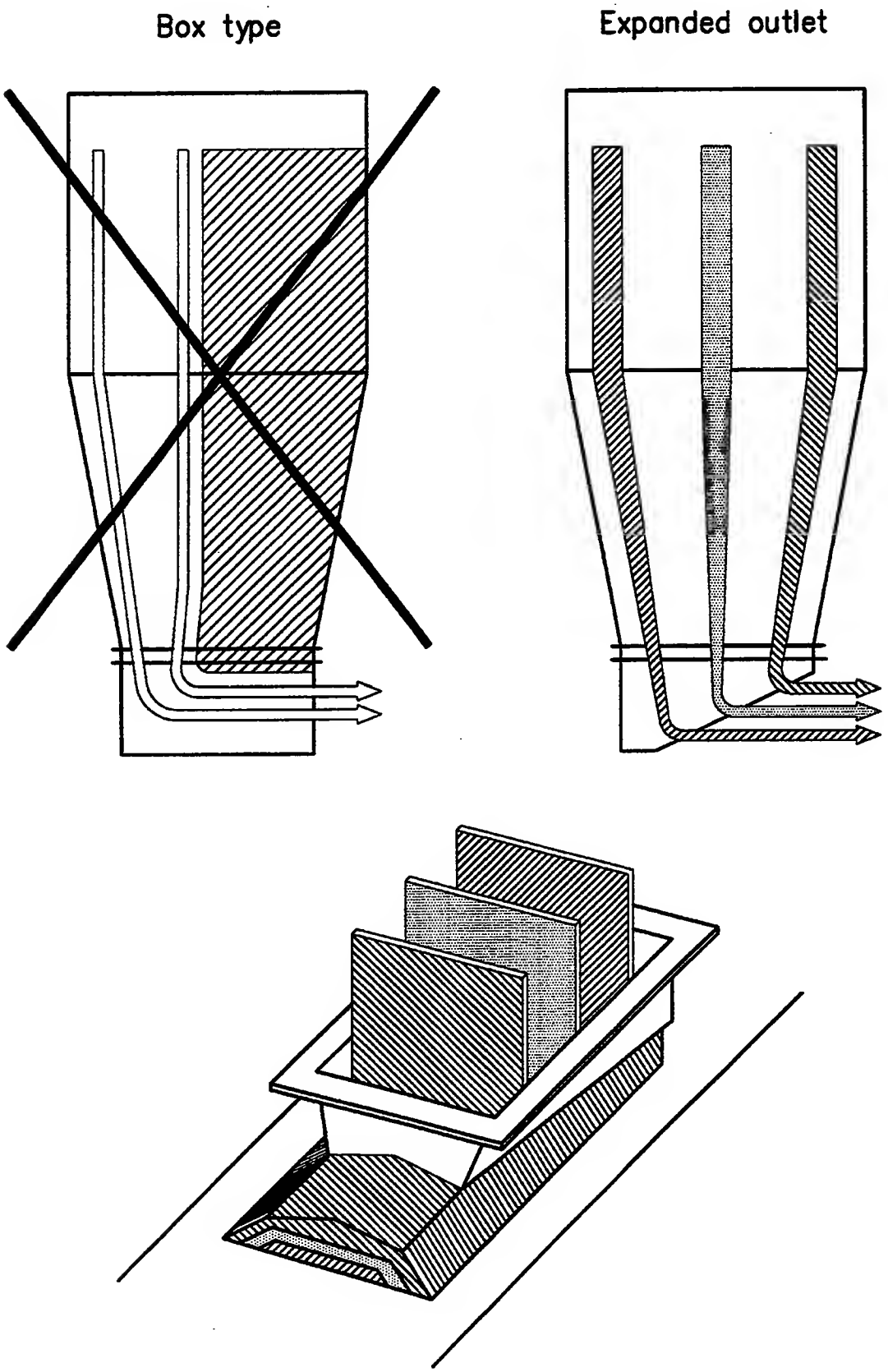

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Figure 23.2 The concept for full outlet activation

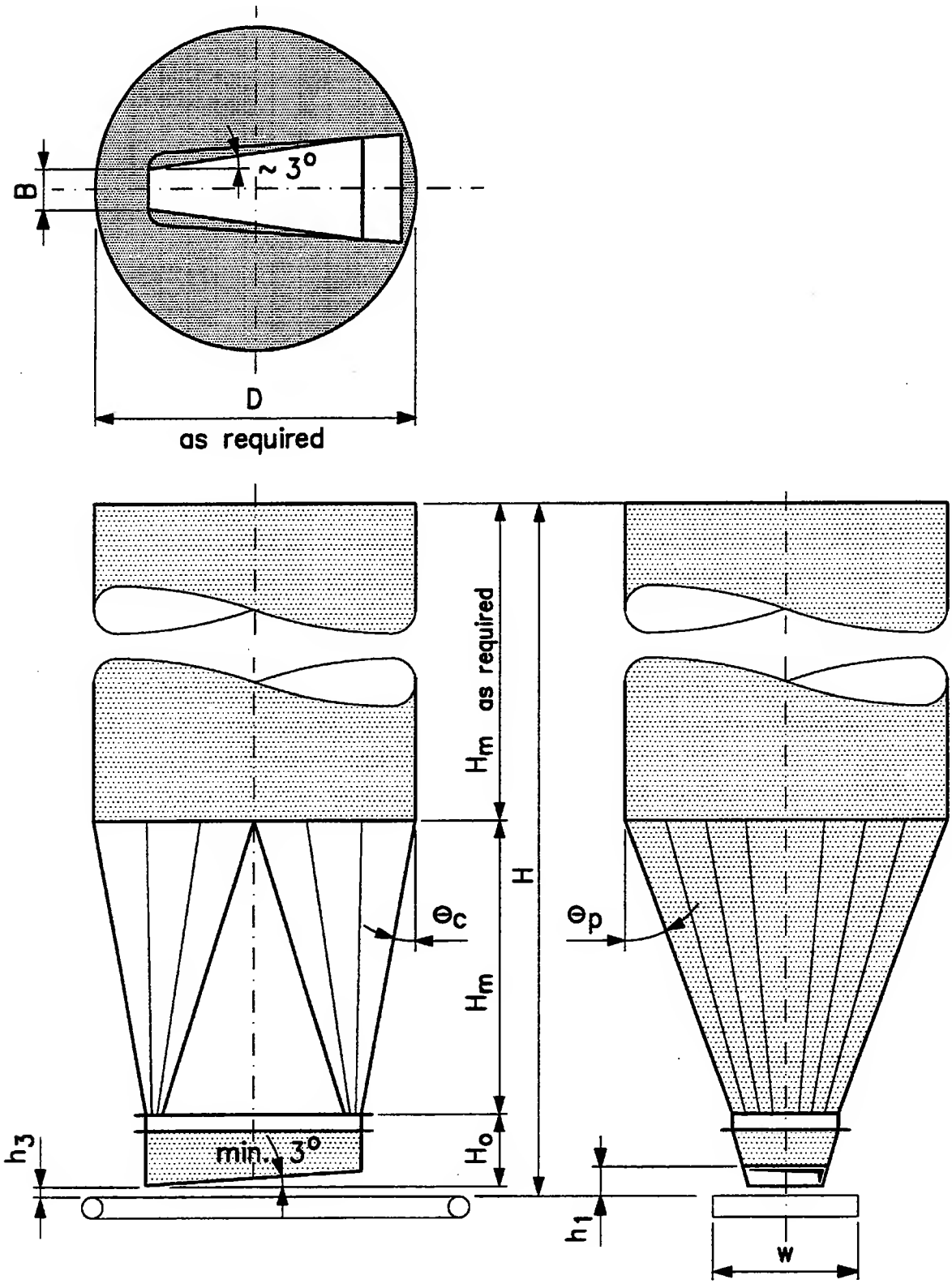


Figure 23.3 The concept for full outlet activation the tapered box

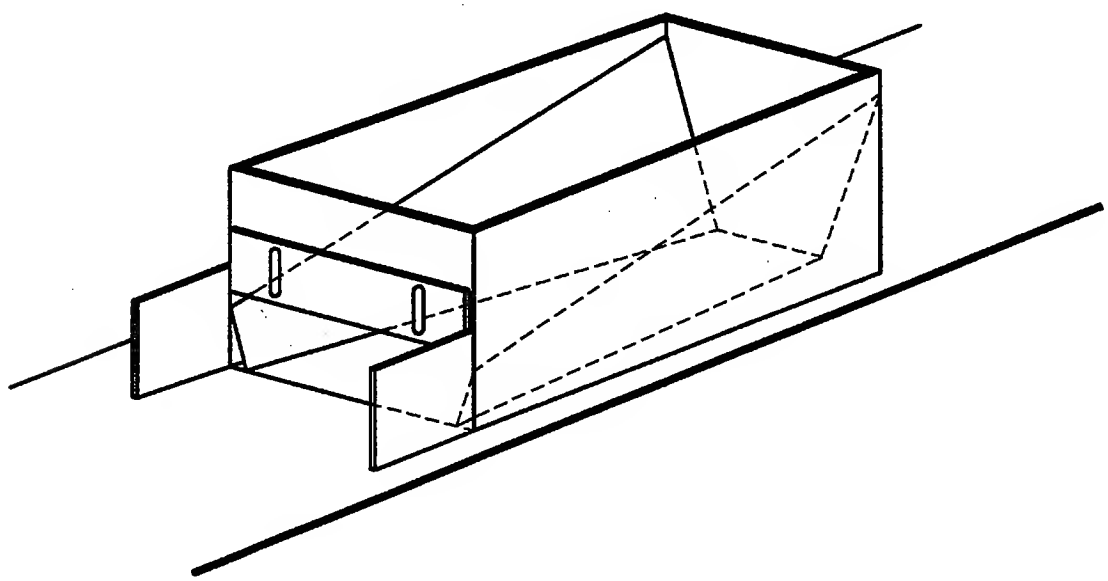
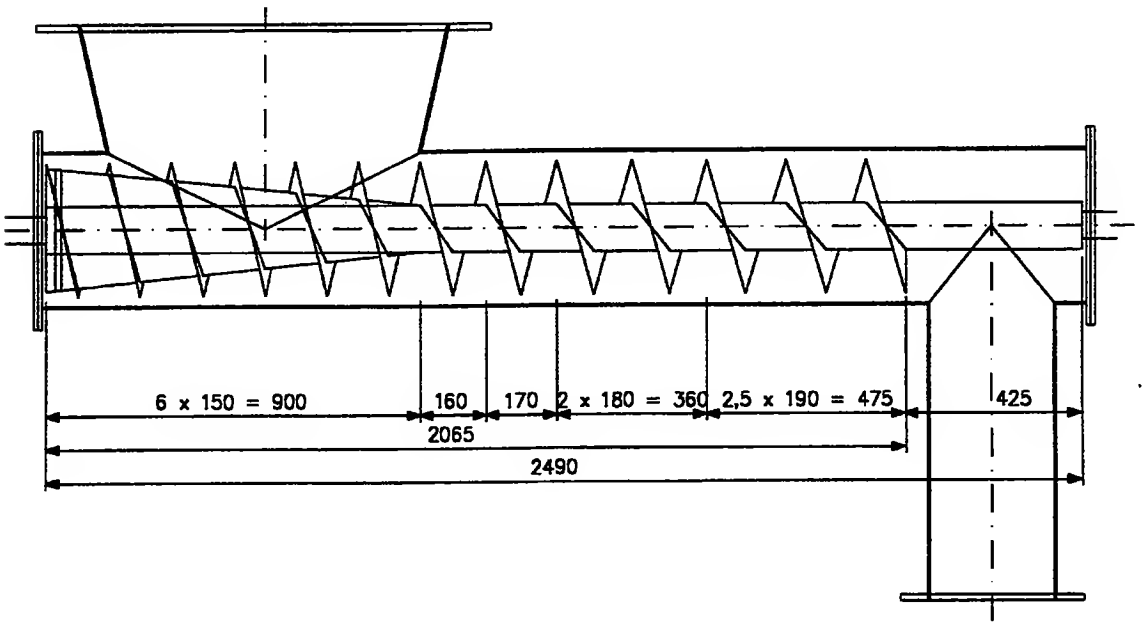


Figure 23.4 The concept for full outlet activation the screw feeder



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Coal Storage and Proportioning

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Summary

Intermediate storage of raw coal and pulverized coal is common practice with nowadays indirect coal preparation systems. As constant feed from such bins either to the coal mill or the burners of a kiln system is a prerequisite for a safe operation the problems related to coal handling and feed rate control will be discussed. For both coal qualities, wet raw coal and dry pulverized coal, criteria for bin design and feeder selection will be given.

1. INTRODUCTION

Intermediate storage of raw coal and pulverized coal is common practice with nowadays indirect coal preparation systems. Constant feed from these bins to the mill and to the burners of the kiln system is a prerequisite for a safe operation. Any interruption in coal supply heightens the risk of explosions inside the mill or inside the precipitators. During start-up of many of the modern coal preparation plants marked discharge problems were encountered - either the bin outlet was crossed by a stable material bridge, or a rathole formed in the bin, or the feeder was flushed by aerated pulverized coal. In most cases such handling problems are directly attributable to an incorrect design of the bin outlets.

In this presentation raw coal and pulverized coal handling shall be discussed, above all the design of storage and feed bins, feeder selection and the problems related to feed rate control.

2. IMPACTS OF BULK MATERIAL FLOWABILITY ON BIN DESIGN AND FEEDER SELECTION

As to create a common base for further discussions, first of all some terms should be cleared up.

- Both, coarse, wet raw coal and dry pulverized coal - as different their properties are - show a typical cohesive behavior. Cohesive means, that the interparticle forces result in strength, being the reason for the ability of coal to form and support stable obstructions to flow as arches or ratholes.
- As utmost availability and reliability of operation are the fundamental requirements to meet by modern coal preparation plants, it is evident that all storage and feed bins for raw coal as well as for pulverized coal must be designed for mass-flow. By the way, mass-flow means that all the material contained in a bin is activated and moves to the bin outlet, whenever any of it is discharged. This in contrast to funnel-flow where material in a narrow core only just above the discharge opening is activated.

3. RAW COAL HANDLING DESIGN OF MILL FEED BINS AND FEEDER SELECTION

As already mentioned, wet raw coal is a bulk material of considerable cohesiveness. The major risks in coal mill operation are interruptions in coal supply. As to guarantee highest reliability of operation, coal mill feed bins must be designed for mass-flow.

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3.1 General Comment on Mill Feed Bin Design

Some comments seem to be opportune:

Do not abuse coal feed bins as additional storage bins. With regard to possible discharge problems they should be designed for capacities as low as possible or even be avoided following the rule that "the best bin is no bin" (figure 1). In case the feed bin is fed from a fully automatic storage pile, a bin capacity of one to three hours of continuous mill operation is sufficient. In case several feed bins - one bin for each coal quality which should be mixed - are fed discontinuously by front end loaders, higher bin capacities, sufficient for one to three shifts of continuous mill operation, are required.

- Never use one bin to feed several mills. As simultaneous operation of the mills cannot be maintained all the time and discharge problems are therefore programmed during a shut-down of one mill.
- Wedge shaped bin discharge hoppers with slotted openings are preferred to conical bin discharge hoppers with circular openings as they can be equipped with smaller feeders. Note: The minimum width of a slotted opening required to prevent arching is half the diameter required for a circular outlet. A slot, by the way, is defined to be a rectangular opening with a length which is at least two and a half times its width.
- From table 3 it becomes evident that all the wall slopes of mass-flow bin discharge hoppers strongly depend on the material of which the hopper is made or by which it is lined. Obviously the general statement that hopper walls should not be inclined less than 70 degree is not sufficient. This statement is only valid for Ultra High Molecular Weight (UHMW) Polyethylene (PE).
  Hence it follows that new coal feed bins should be designed with PE lined wedge shaped discharge hoppers, and that, by a PE-lining, feed bins with a funnel-flow sequence can be in some cases converted to mass-flow.
- When selecting a feeder system it should be kept in mind that bin, interface to the feeder and feeder work as one system. Any change in one will affect the performance of the whole system.
  What selection of feeder size concerns, there is a primateship of the requirements for discharge reliability over all other requirements, i.e. that sometimes, based on material's flow properties, a wider feeder must be selected than required when considering the feed rate as exclusive design criteria.
  The design of the interface is of particular importance with regard to the flow pattern that will develop in a bin. Mass-flow will develop provided that the interface is designed such that the entire bin discharge opening is activated whenever material is withdrawn. In case of raw coal this is problematic. On one hand, raw coal requires wide discharge openings due to its cohesion. On the other hand, the required feed rates are low. Together with most feeder's tendency to discharge preferably from the rear, this result in an only partly active outlet (figure 2). To remedy this problem the interface must be designed expanding, i.e. such that the volume of the material bed on the feeder increases in direction of material flow. This in turn initiate material flow not only at the rear but across the full discharge opening (figure 2).

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3.2 Comment on Feeder Selection

Volumetric feeders are sufficient for feed rate control to the coal mill. The possibility of gravimetric feed rate control which offer some feeders is not a sufficient criterion to select a more expensive or more complicated feed system.

The belt conveyor is quite common for the discharge of tipping hoppers. There is no reason which would exclude its application as feeder underneath a raw coal bin. As belt conveyors are not gas tight, an additional device must be installed to provide for the air lock to the mill. Gravimetric feed rate control is possible if the belt conveyor is designed as weigh feeder. With regard to an expanding outlet, the feeder width should be selected to be twice the width of the bin discharge opening (figure 4).

The apron feeder proved to be a very reliable but expensive feeder. Its application is justified at least with very large raw coal storage bins. As to an expanding outlet design, similar criteria apply as for weigh feeders.

The casing of a trough chain conveyor is completely air and dust tight. With such feeders there is no problem with spillage. A sufficient air lock to the mill is achieved if a minimum material height of 3 m is maintained in the feed bin. Trough chain conveyors show a strong tendency to discharge material preferably from the rear. The most advanced feeder Redler - one of several suppliers of such feeders - supplies for raw coal applications is equipped with an expanding interface (figure 5) and a nucleonic scale as to provide gravimetric feed rate control.

Trough chain conveyors are judged to be very reliable but expensive feeders.

The table feeder is quite common for raw coal applications. With the feed bin arranged just above the mill inlet a very compact plant design can be reached. Table feeders allow a volumetric feed rate control. Such a feeder is usually a quite large equipment as it is applied with conical bin discharge hoppers - remember, the minimum diameter of a circular discharge opening required to prevent arching is twice the width of a slotted discharge opening. With regard to a fully active outlet the side skirts at the end of the feed-in line should have a 'corkscrew' shape
(figure 6).

The un-coaler feeder (figure 7) is quite a new development. Its mode of operation is similar to that of a bin activator, but its design is much more rigid. A center dome functions as activator or live bottom when vibrated. By this, material flow becomes unproblematic. Furthermore such a feeder can be fitted directly to the bin discharge opening without any need for an expanding interface. Automated control systems arranged to respond to belt scale signals allow volumetric feed rate control.

The un-coaler feeder is considered to be a very reliable feeder for raw coal.

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3.3 How would a Mill Feed Bin for Raw Coal look like, when Designed for Reliable Operation?

(figure 3)

- The bin would definitely be designed for mass-flow.
- The front wall of its wedge shaped discharge hopper would be by about 5 degree steeper than its rear wall.
- The slotted bin discharge opening would rather be trapezoidal than rectangular, with a minimum width of 800 mm and a taper of about 2 - 3 degrees.
- Guide plates would be fitted to the pin gate, corresponding to the shape of the bin discharge opening.
- The interface would be expanding in direction of material flow.
- A feeder of at least 1600 mm in width would be selected.

4. PULVERIZED COAL HANDLING DESIGN OF STORAGE BINS AND FEED RATE CONTROL

With pulverized coal, handling problems are similar to that with raw coal, although the mechanical properties of these two coal qualities are quite different. As raw coal to the mill, pulverized coal must be fed at a controlled rate to the burner as to guarantee complete combustion. Prerequisite for an accurate feed rate control is again reliable material discharge from the storage bin. And again, mass-flow would be the most suitable flow sequence in a pulverized coal storage bin.

4.1 Standard Design Criteria for Pulverized Coal Handling Systems

Nowadays, storage bins of a standard design are offered for pulverized coal by specialized suppliers. Such bins are pressure shock resistant and fully equipped with feeding and flow rate control systems, with the necessary monitoring equipment like load cells for bin content control, temperature and CO measuring devices, with safety equipment as explosion doors and inertization systems. Suppliers design bin discharge hoppers with a standard slope of not less than 70 degree and made of stainless steel. To prevent caking due to condensation effects the discharge hoppers are provided with a thermal insulation.

Modern pulverized coal feeders are of a gravimetric type and have proved successful in supplying the fuel at a controlled rate to the burner.

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4.2 Discharge Problems with Pulverized Coal

Most discharge problems from pulverized coal bins are due to a deficient design of the bin outlet section.

To overcome bridging and ratholing discharge aids have been installed either of a mechanical type such as paddle wheels or of a pneumatic type such as pulsed air nozzles.

But still, for some solutions, pulverized coal's flow properties have been ignored. A typical example: For a bin discharge design (figure 8) where the bin's discharge hopper penetrates a rectangular cube, discharge problems are immanent for a cube width smaller than 600 mm, even if a paddle wheel is installed. A sledge hammer, an old fashioned device, often proved to be the most suitable discharge aid.

Note: The sections which must be activated to prevent arching of pulverized coal are at least 1200 mm in diameter for circular discharge openings or at least 600 mm in width for slotted discharge openings.

In operational practice it also became manifest that the standard hopper slope of 70 degree even with stainless steel is not sufficient in all cases. As a result ratholes have formed in the bins. Again, aeration systems were installed as to promote material flow (figure 9).

Note: Although pulsed aeration systems are developed so far, that reliable bin operation is achieved, we doubt, whether aeration is the adequate measure to promote flow for new installations. This, above all, when considering the complicated venting and dedusting systems, which must be installed to ensure a trouble-free operation of the subsequent feeders.

The flow regime can also be influenced when changing the wall frictional behavior. A UHMW PE lining in a 70 degree sloped bin discharge hopper would in most cases induce mass-flow. Nevertheless, it is not recommended to base the design of new storage bins for pulverized coal on a UHMW PE lining due to PE's low melting point (~ 135 ° C). But a PE lining is accepted to be an effective measure to convert a funnel-flow sequence to mass-flow in an existing bin.

Bin inserts too, influence the flow regime in a bin. By such inserts, the amount of stagnant material in a funnel-flow bin may be reduced considerably (figure 10). Usual shapes are tables or cones which are placed above the outlet. In contrast to these corrective bin inserts Jenike and Johanson propose a cone-in-cone design (figure 10). The internal cone is designed for mass-flow what width of discharge opening and wall slope concern. It was found that the half angle of the outer cone may be twice the half angle of the inner one. Material will flow through both, the inner hopper and the annulus between inner and outer hopper.

Note: The cone-in-cone system offers not only a simple measure to convert pulverized coal bins with a funnel-flow sequence into mass-flow bins, but also is a possibility to design new bins with discharge hoppers of a reduced height.

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4.3 Feeder Systems

Nowadays various gravimetric systems are used for feeding pulverized coal to the several burners of a kiln system. The most important feeding systems are the loss-in-weight and the impact flow meter systems.

In the following, not the advantages of before mentioned feeders should be discussed, it should rather be reflected how a feeder system, as simple as possible could look like.

The discharge equipment of a SIMPLEX feeder with its activating paddle wheels and its proportioning screw is a perfect proportioning screw feeder system when fitted directly to the storage bin (figure 11). Particular care should be taken with the design of the proportioning screw which should be expanding, e.g. with decreasing core and increasing pitch in direction of material flow.

For multiple outlet bins several proportioning screws can be arranged in parallel to each other. If the diameter of the screws is selected sufficiently wide, no further activators are needed (figure 12). Again each proportioning screw should be of an expanding design.

The flow rate can be measured by a subsequent impact flow meter or a proportioning rotor scale, which is quite a new development and will be discussed later. The flow rate is controlled by altering the speed of the screw(s). For calibration purposes the bin should be installed on load cells and can be used as a whole as loss-in-weight system.

Live bottom feeders consist of a vibrated frame with a set of blades. Each blade acts as a vibratory feeder. With this type of feeder, discharge problems are prevented as it promotes flow across the entire section of the bin's discharge opening. These feeders are fitted to the storage bin's discharge opening. These feeders are fitted to the storage bins by means of pressure shock resistant flexible sleeves (figure 13). Again the flow rate can be measured by a subsequent impact flow meter or a proportioning rotor scale. The flow rate can be controlled by altering the vibrator amplitude. For calibration purposes the bin should be installed on load cells.

All above mentioned feeder systems not only consist of the discharge and measuring devices, but additionally of a rotary gate which provides for the air lock to the subsequent pneumatic conveyor. At this point, the proportioning rotor scale should be mentioned, where feeding and weighing are integrated in the same device.

The proportioning rotor scale (figure 14), a suspended scale, consists of a rotor (with vertical axis) installed in an explosion-resistant housing. The two suspension heads are arranged in line with the feed-in duct and the delivery line as to prevent that external forces which result from incorrect connections affect the weighing. The measuring pivot is arranged at the opposite.

For the future the proportioning rotor scale seems to be a very competitive feeder.

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4.4 How a Modern Bin Feeder System for Pulverized Coal would look like, when Designed for Reliable Operation

(figure 15)

- The bin would definitely be designed for mass-flow.
- A cone-in-cone design would be selected for the bin discharge hopper with wall slopes of 65 degree for the external and 77 degree for the internal hopper.
- Material of construction for both hoppers would be stainless steel plates.
- The minimum diameter of the circular hopper discharge openings would be at least 1400 mm.
- Material discharge would be activated by a live bottom feeder.
- The flow rate would be controlled by a proportioning rotor scale (or a flow meter).

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  VA 84/5070/E

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Table 1 Funnel-Flow versus Mass-Flow

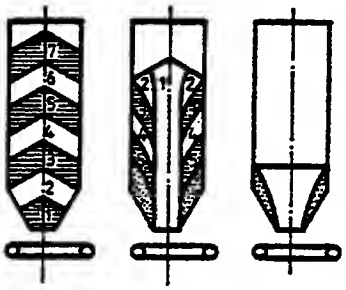
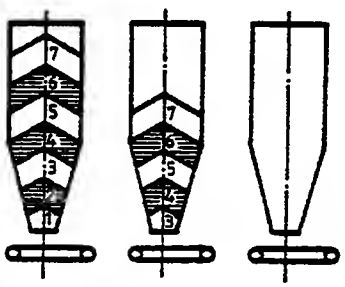
	Funnel - flow	Mass - flow
Bin shape	Compact flat hopper	Slender steep hopper
		
Flow sequence	First in - Last out	First in - First out
Discharge order	Loading order inversed	As loaded
Silo utilisation	partial	complete
Handling problems		
- no flow	arching piping	arching
- discontinuous flow	erratic flow flushing	–
- formation of materials build-ups on silo walls	possible	–
- segregation	inevitable	counteracted

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Table 2 Handleability Characteristics of Raw Coal

Moisture content	7 - 15 % H_2O
Granulometry:	
- Reference value	50 mm
- Minus 50 mm	95 %
- fraction	50 %
Minus 5 mm fraction	
Material strength	5000 - 7000 Pa
Effective angle of friction	47 - 52 deg
Angle of internal friction	37 - 45 deg
Angles of wall friction:	
- Mild steel	27 - 30 deg
- Stainless steel	28 - 28 deg
- Uhmw polyethylene	22 - 25 deg

Table 3 Design Parameters of Mass-Flow Hoppers for Raw Coal

Type of hopper	**Conical**	**Wedge type**
Minimum hopper halfangles:	θc	θp
- Mild steel	10 - 13 deg	18 - 20 deg
- Stainless steel	12 - 15 deg	20 - 22 deg
- Uhmw polyethylene	15 - 20 deg	22 - 25 deg
Bin discharge opening:		
- Shape	Circular	Slotted
- Min. diameter / width	1200 mm	700 mm
Feeder		
- Min. width	2000 mm	1500 mm

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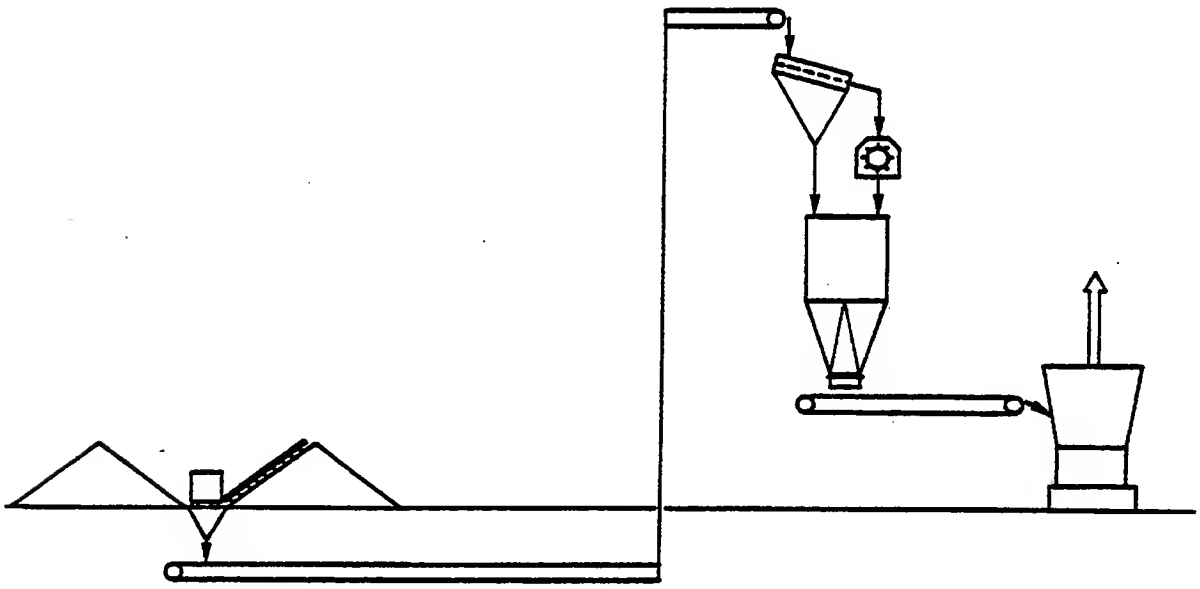
Figure 1 Storage Capacity of Raw Coal Mill Feed Bins

Continuous feed of raw coal bin - Bin capacity: 1 - 3 hours continuous mill operation



Discontinuous feed of raw coal bins with integrated blending - Bin capacity: 1 - 3 shifts continuous mill operation

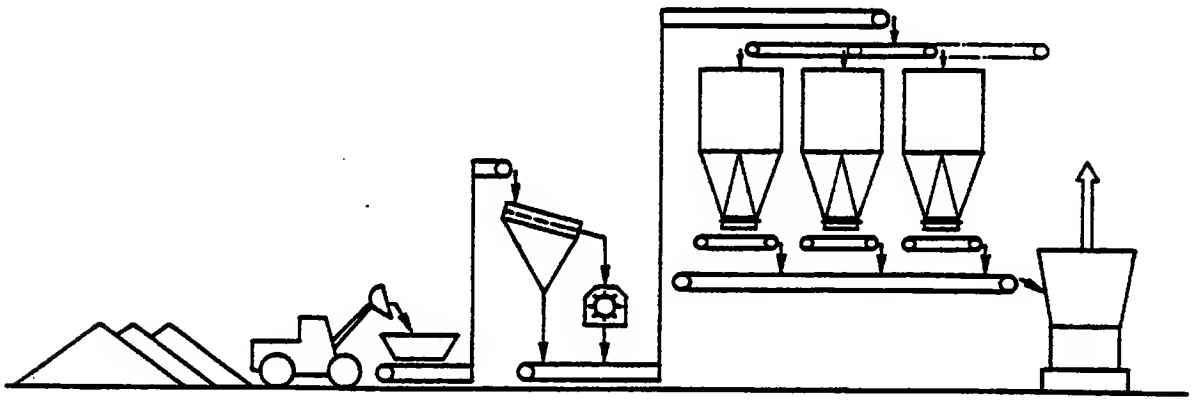


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Figure 2 Outlet Design

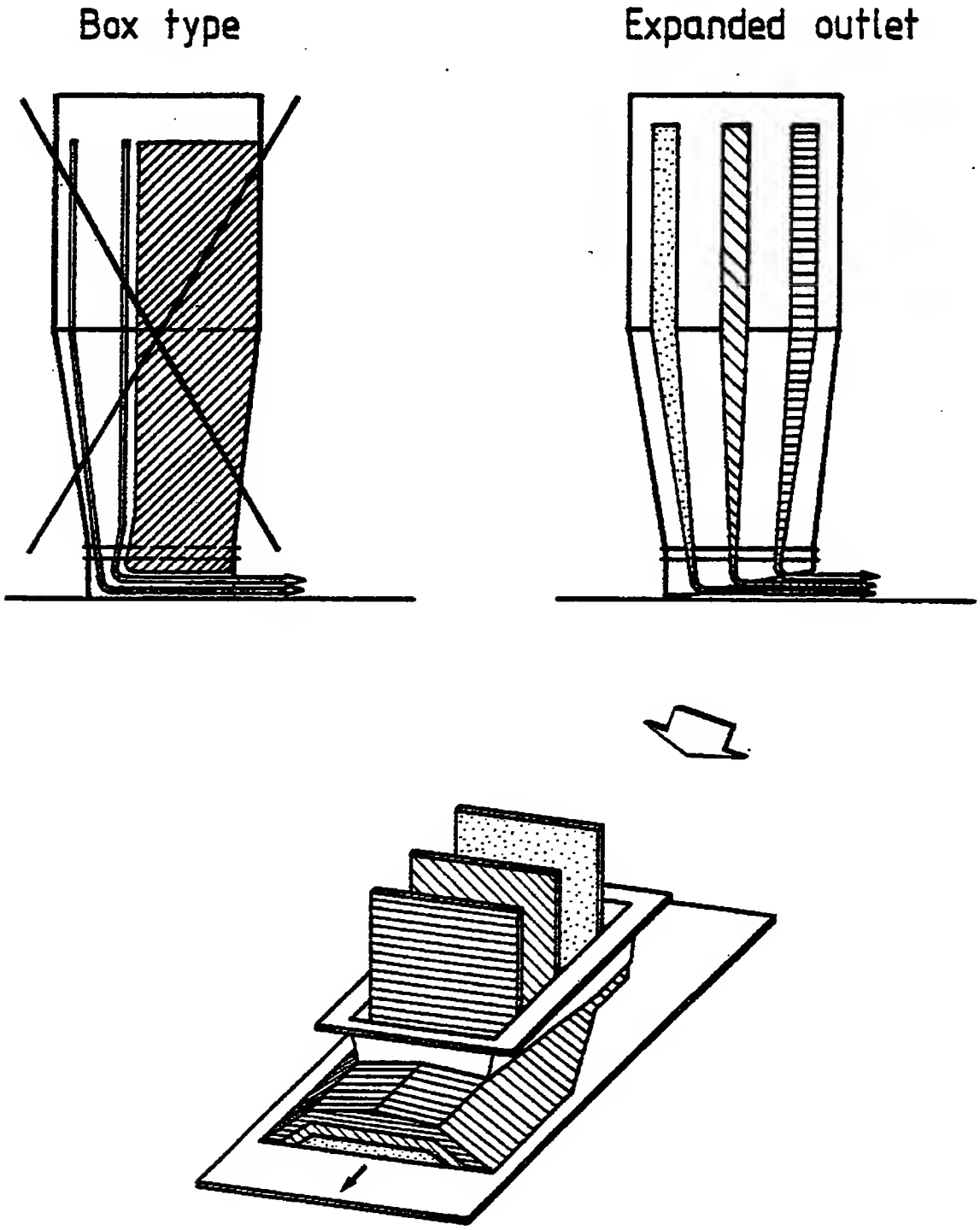

Figure 3 Expanding Bin Outlet

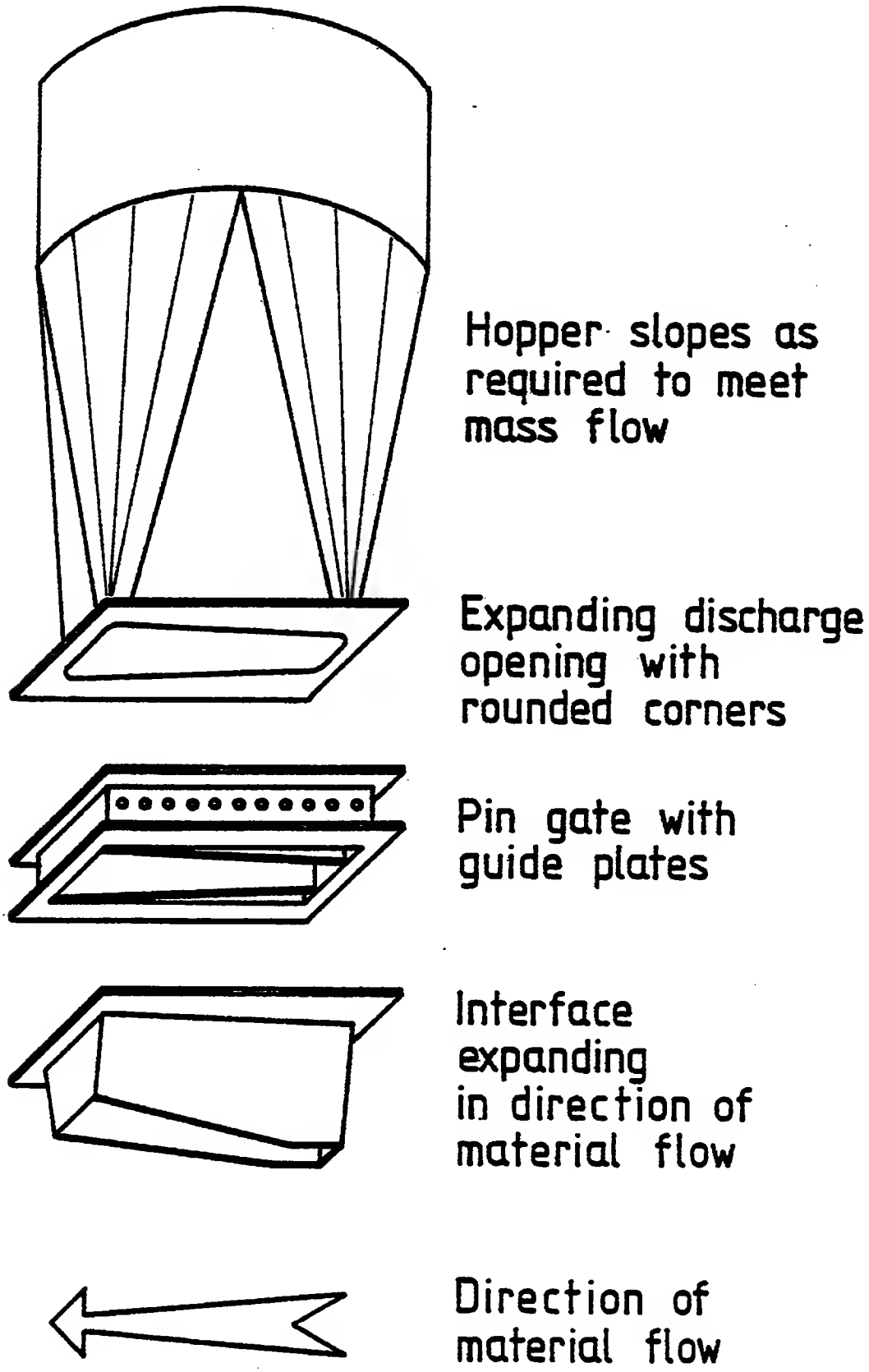

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Figure 4 **Apron and weigh feeders with expanded outlet**

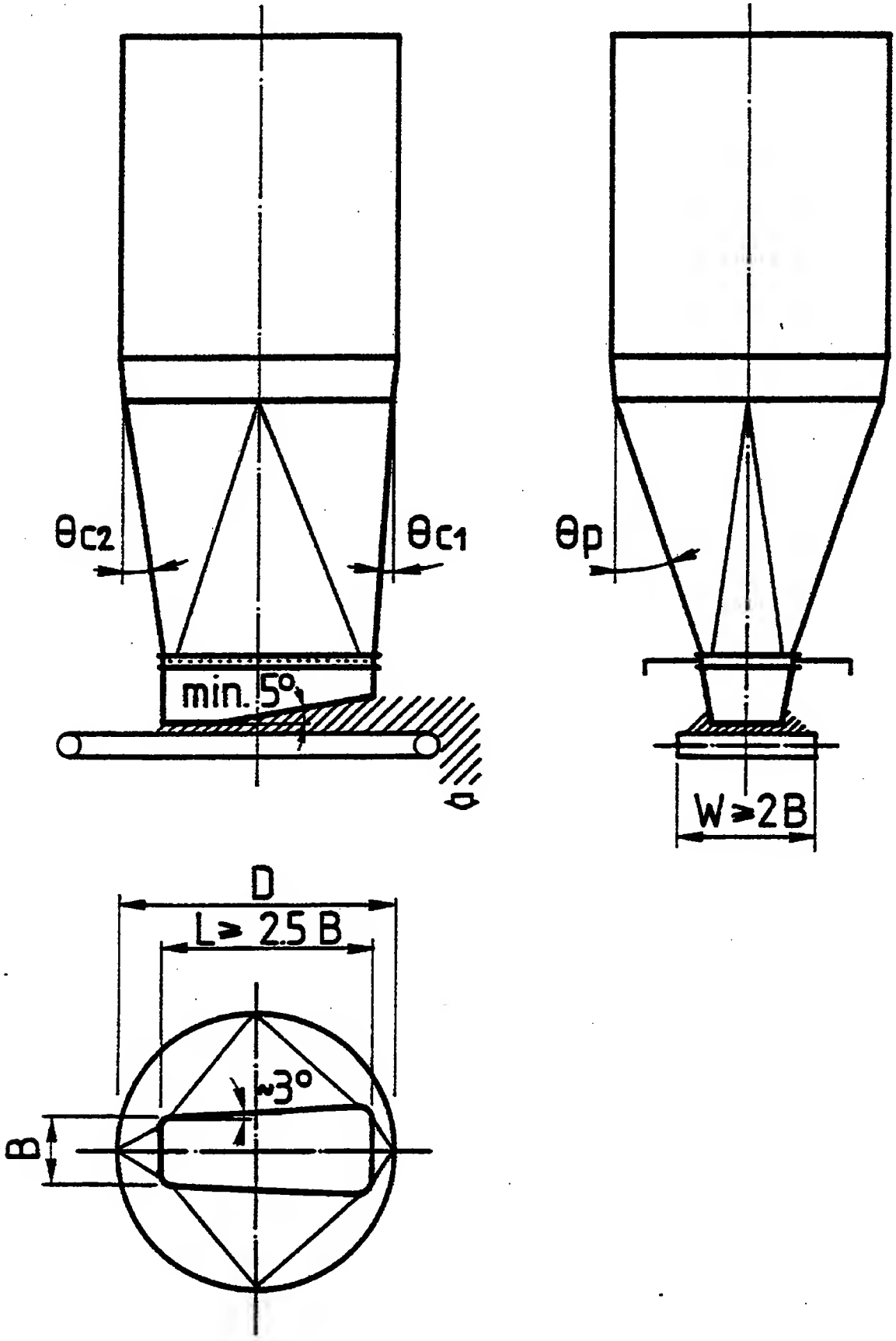

Figure 5 Trough-Chain Feeder with Expanded Outlet

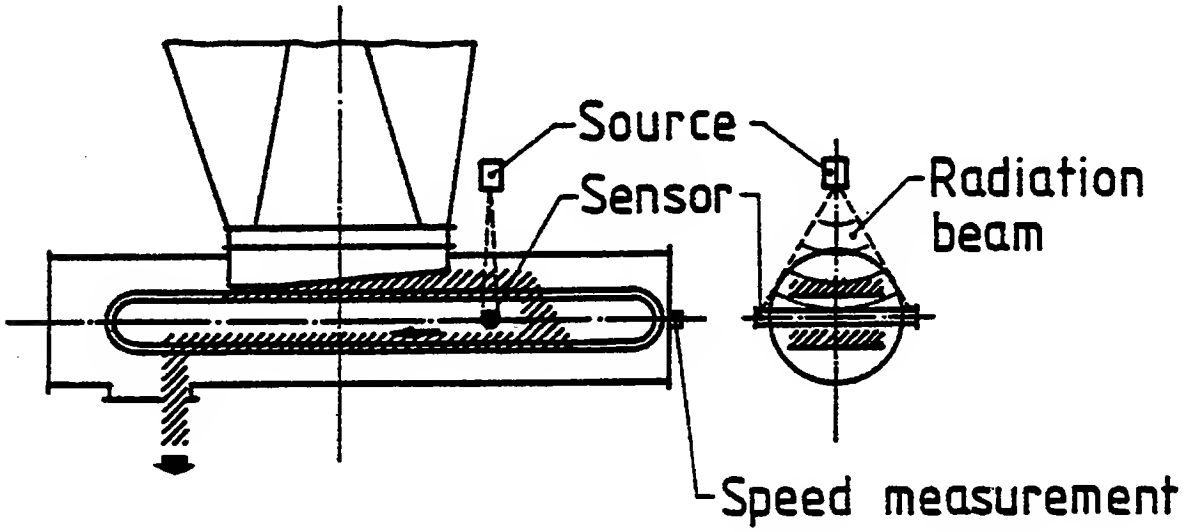

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Figure 6 Table Feeder with Expanding Outlet

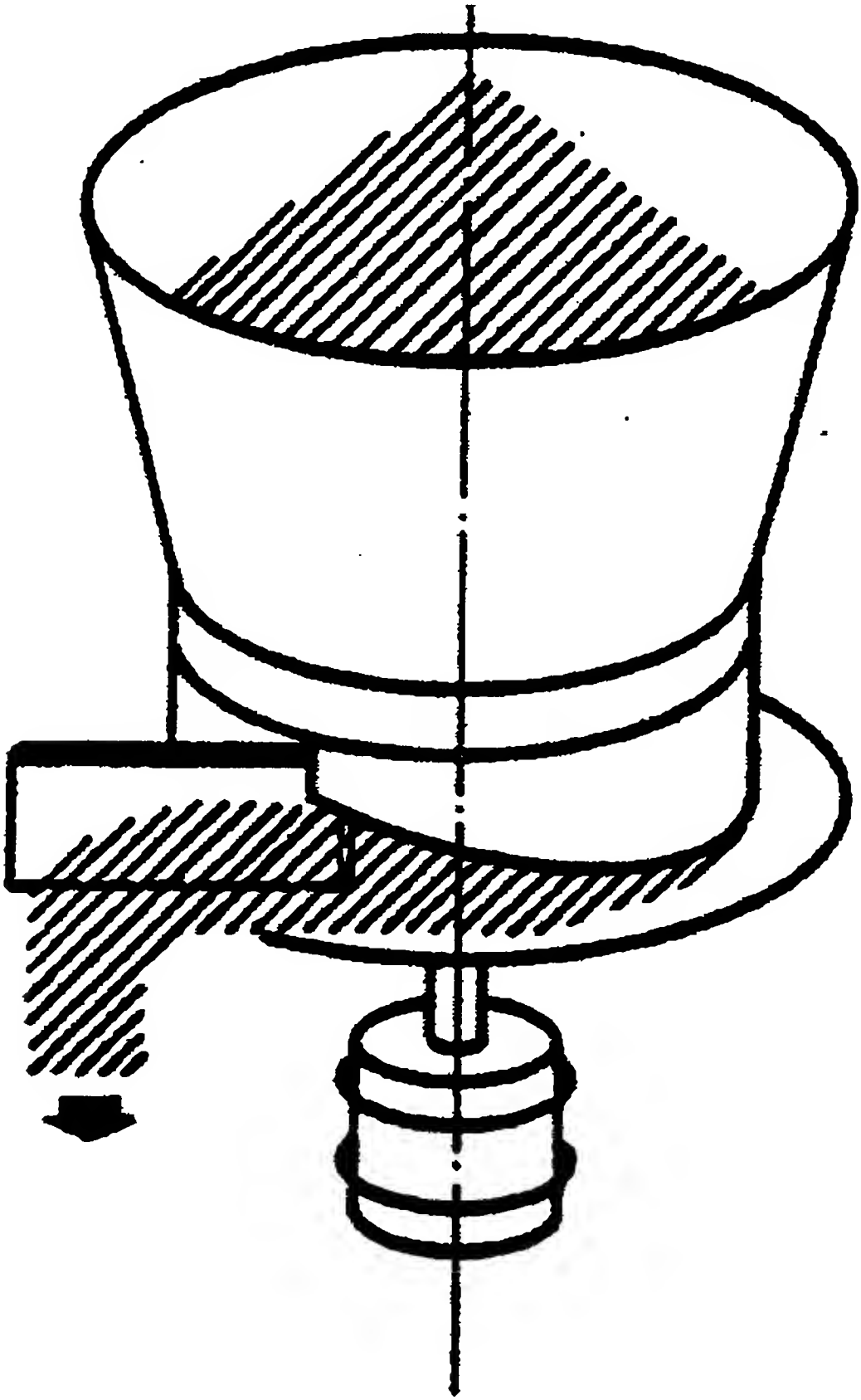

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Figure 7 Un-Coaler Activator / Feeder

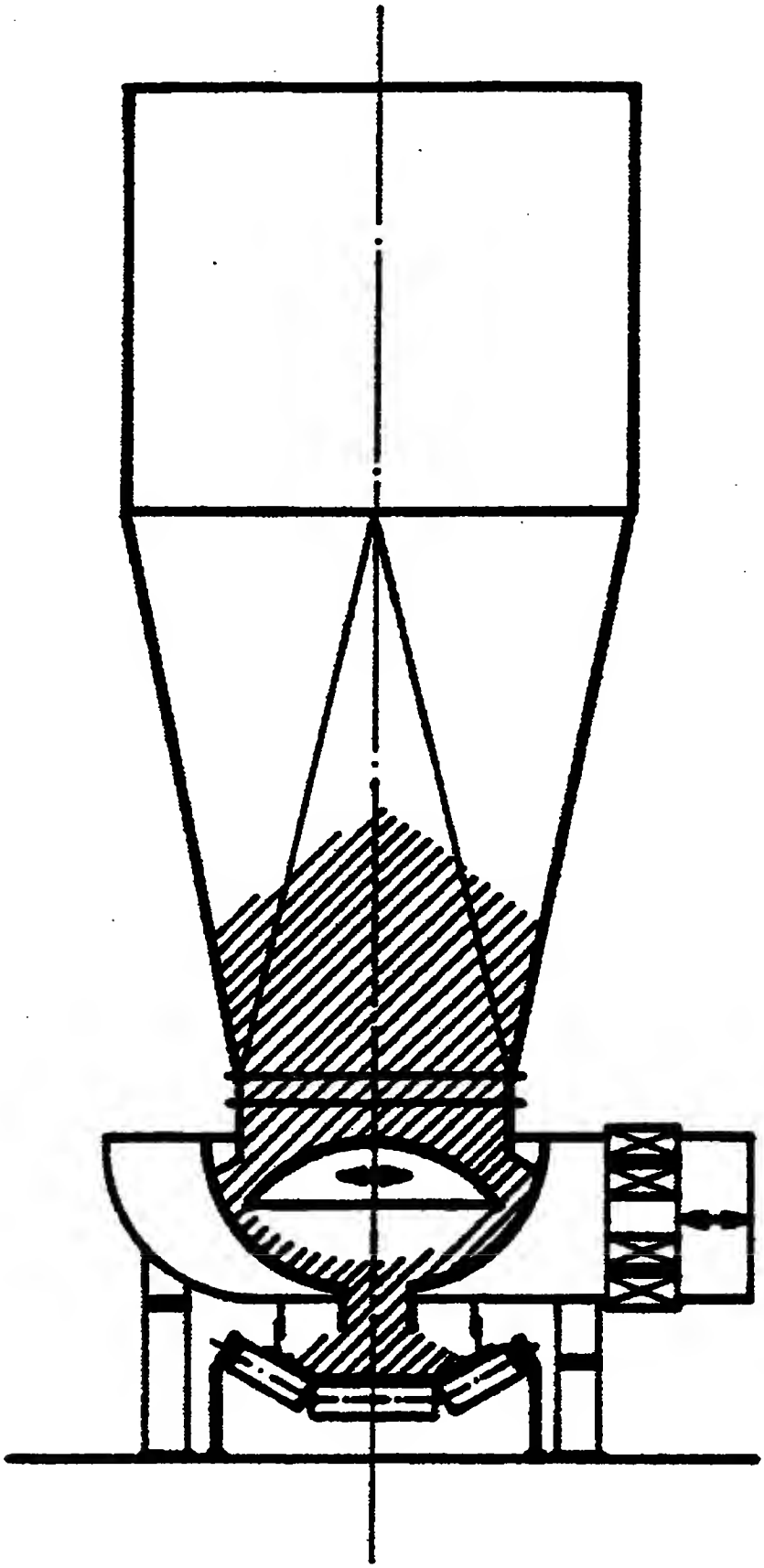

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Figure 8 Multiple Bin Discharge for Pulverized Coal

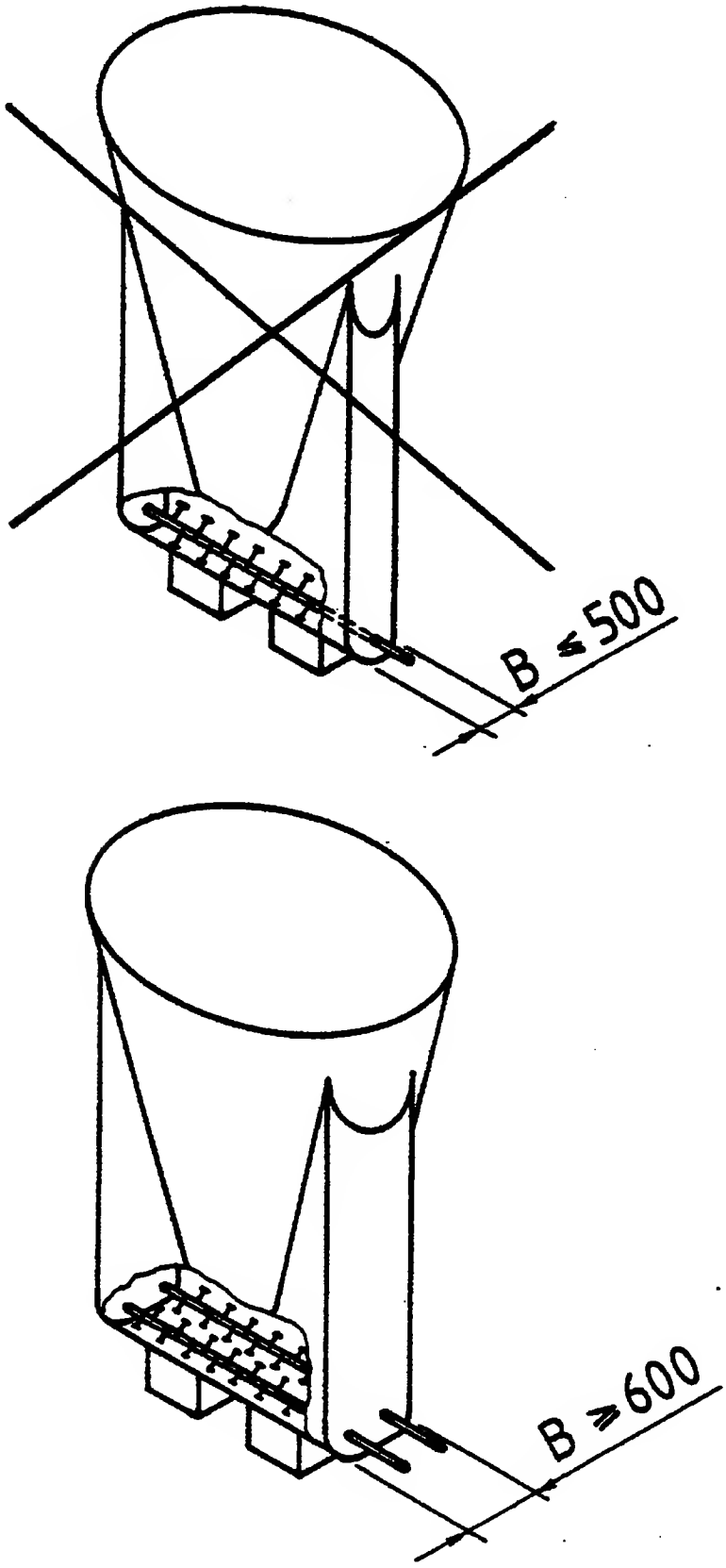

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Figure 9 Flow Promotion by Aeration

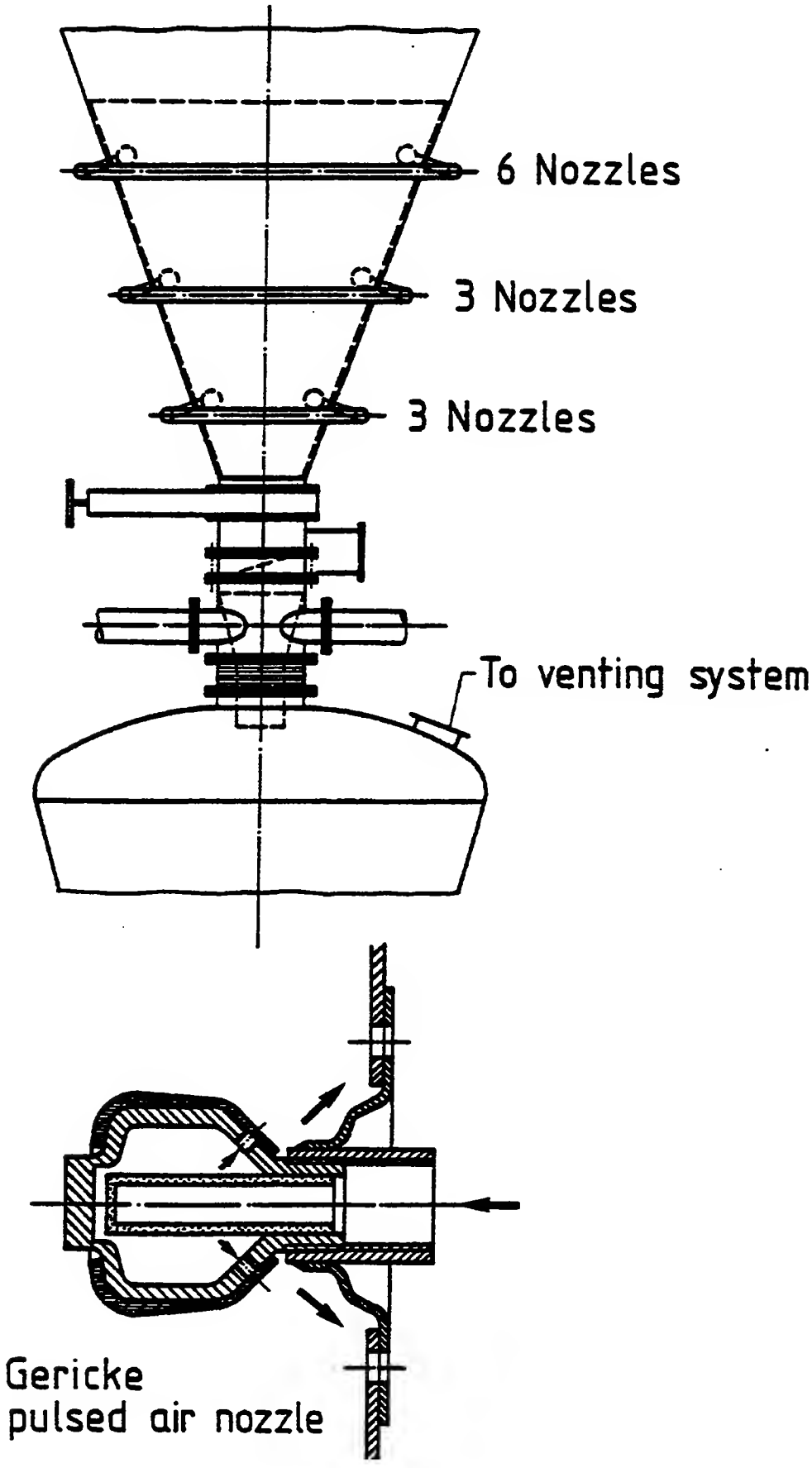

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Figure 10 Modification of Flow Pattern by Bin-Inserts

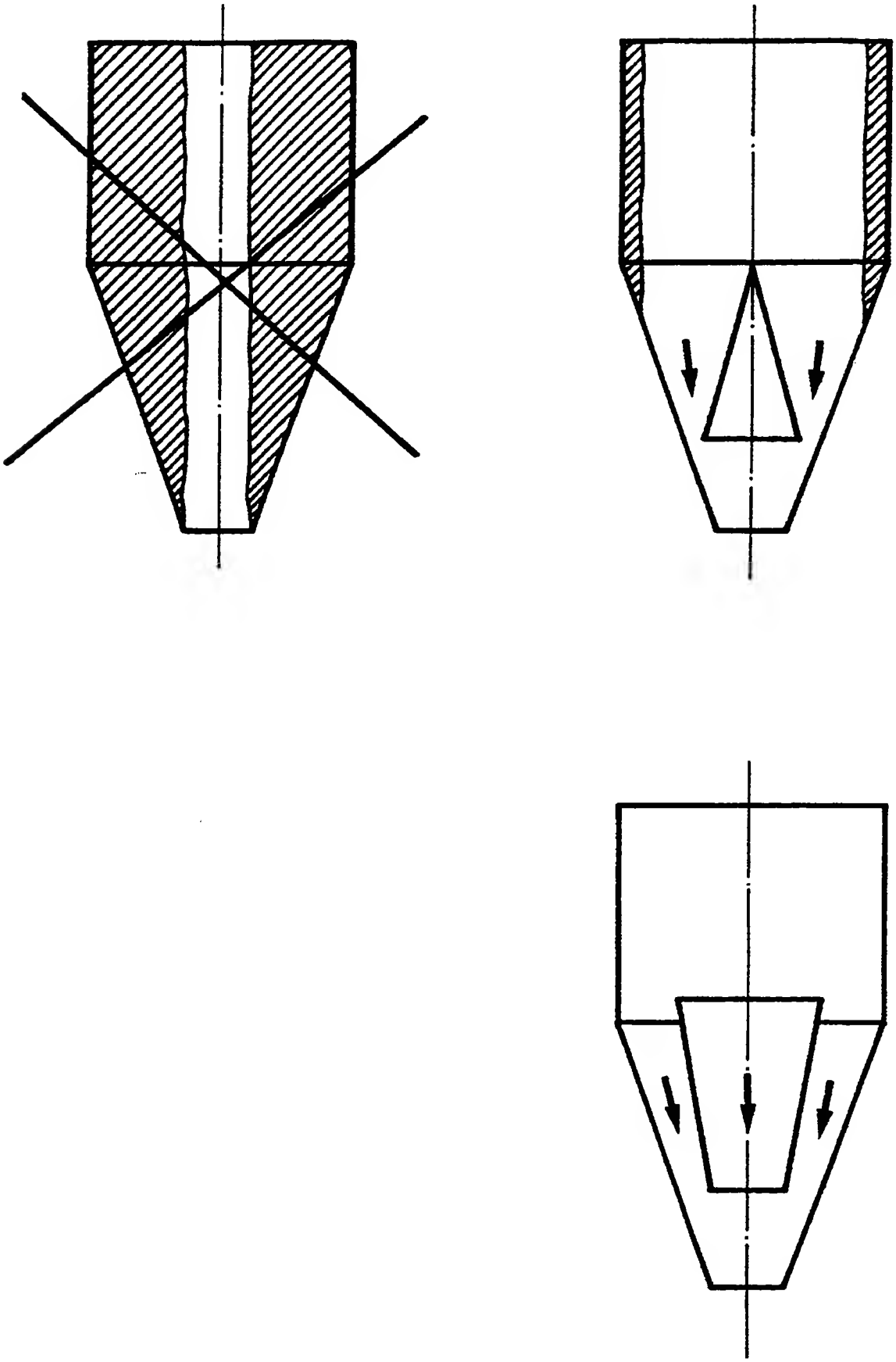

Figure 11 Silo Discharge with Double Paddle and Screw Feeder

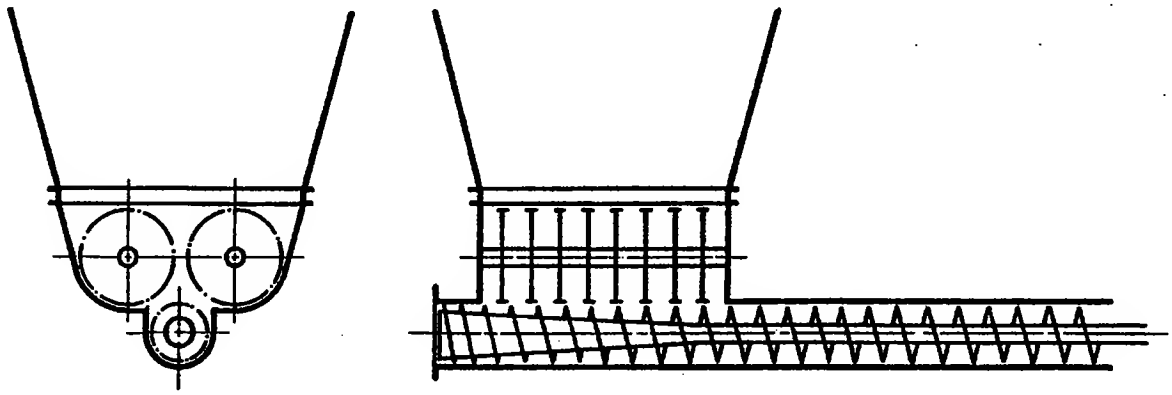

Figure 12 Silo Discharge with Multiple Screw Conveyor (FLS)

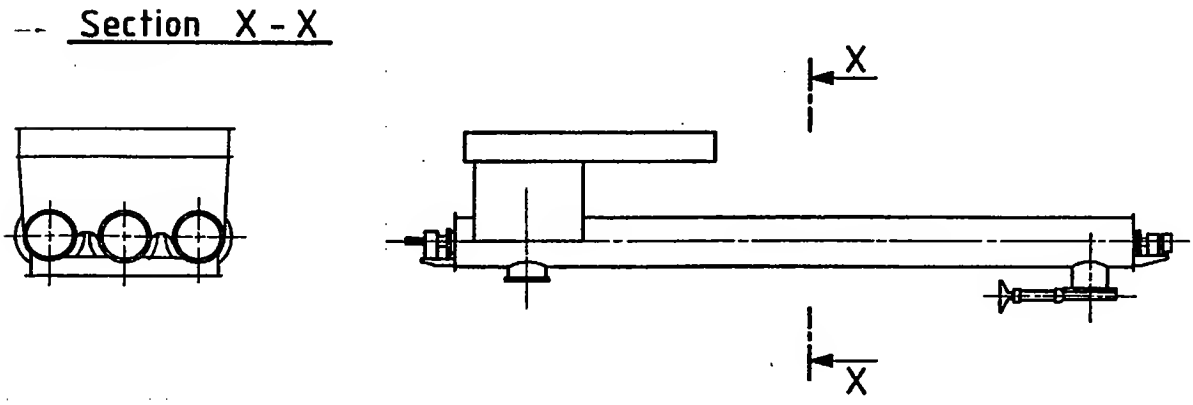

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Figure 13 Live Bottom Feeders

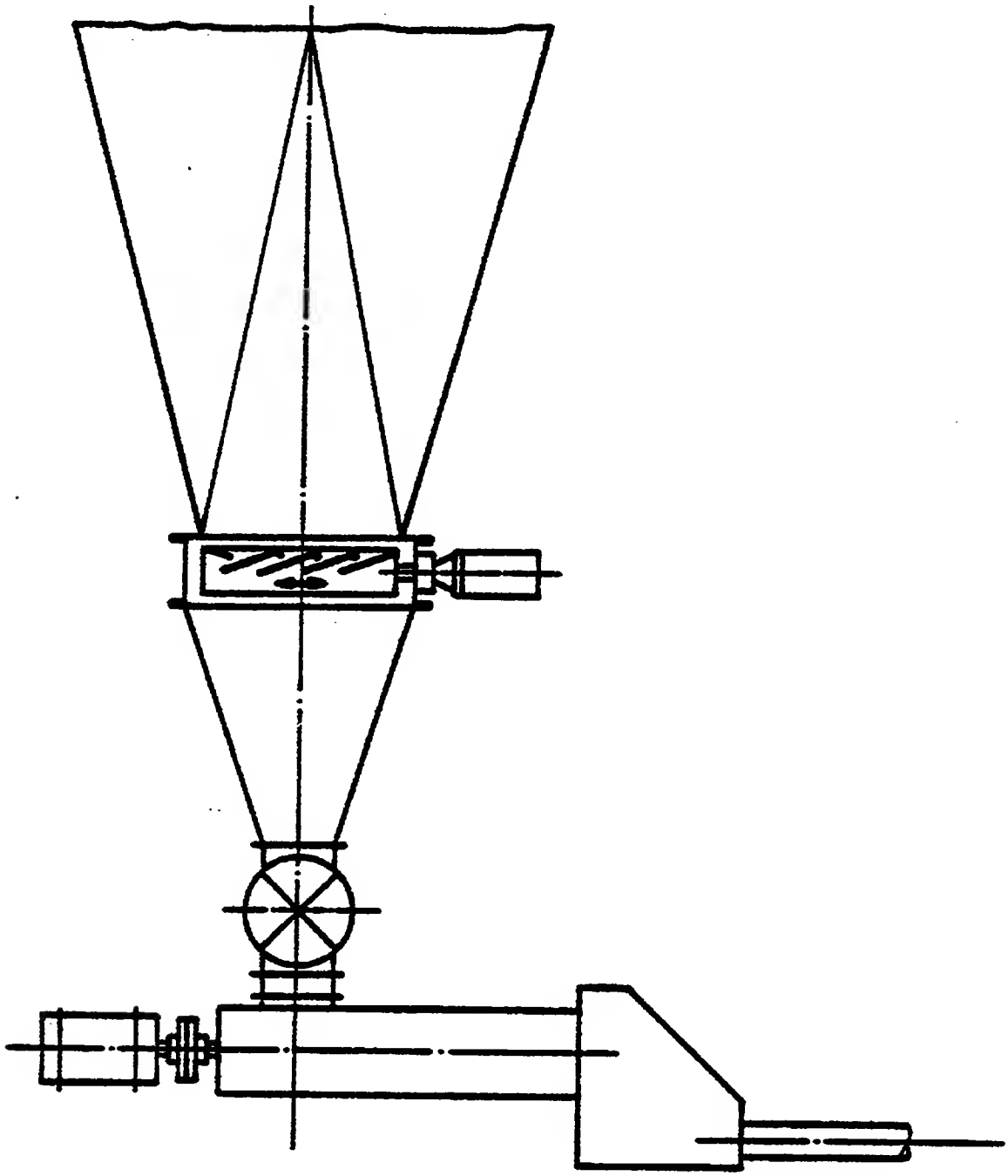


Figure 14 Proportioning Rotor Scale

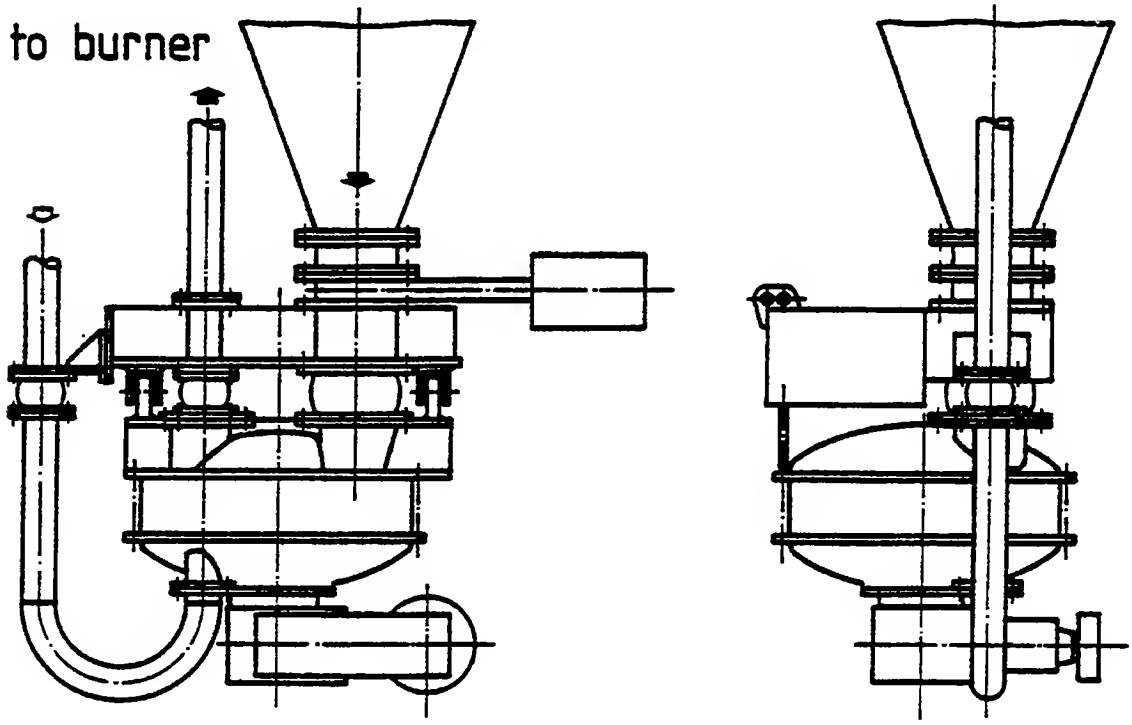
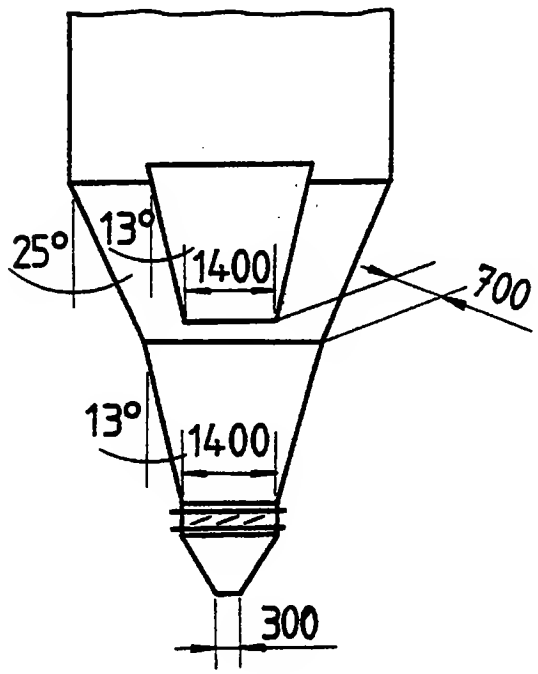


Figure 15 Pulverized Coal Mass-Flow Bin



Cone - in - Cone Design

Live Bottom Feeder

to
Proportioning Rotor Scale /
Impact Flow Meter

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Chapter 4

Raw Material Homogenization and Storage

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2. Preblending of Raw Materials 209

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Aspects of Raw Material Homogenization

F. Bucher
VA 83/5004/E

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Summary

Raw material prehomogenization became a most essential part in modern plants. As daily practice demonstrates, chemical fluctuations often influence kiln operation considerably by coating and ring formation as well as encrustation and clogging problems. As a consequence not only the operation is hampered but also the life time of the lining is shortened.

The homogenizing efficiency is normally expressed by

$e = s_\alpha / s_\beta$

However a better value to judge a homogenizing element with regard to effectiveness is the standard deviation (s_β) of the homogenized material. The standard deviation is only applicable if the samples are statistically independent which can be verified by the correlation function.

Homogenization is achieved by two procedures, by 'blending' or 'mixing'. Blending means that two or more material streams or layers are combined whereas mixing signifies the homogenization of several materials by agitation. By both methods good homogenizing efficiencies are achievable. However a limit is given by the length of chemical fluctuations that can be homogenized.

During the whole raw material preparation process the following homogenizing elements can be met:

- selective quarrying
- Preblending bed
- raw material proportioning
- grinding
- raw meal or slurry homogenization

To select and dimension these homogenizing elements correctly, the determination of the chemical fluctuations is of utmost importance. On existing plants the chemical fluctuations are relatively easily detected whereas on plants in projection they have to be estimated with the aid of the drill hole prospection of the raw material deposit.

If the efficiency of the homogenizing elements has to be judged the chemical fluctuations after each element must be determined.

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1. INTRODUCTION AND DEFINITIONS

As the daily practice demonstrates, chemical fluctuations in the raw mix have a considerable influence on kiln operation e.g. coating and ring formation as well as encrustation and clogging problems.

Therefore the degree of homogeneity directly influences the life time of the refractory lining. Homogenization of the raw materials and the raw mix is, thus of utmost importance.

1.1 Homogenizing Efficiency

normally expressed by

$e = s_\alpha / s_\beta$

where
s_α = standard deviation of the unhomogenized raw material
s_β = standard deviation of the homogenized raw material

$$\text{Standard deviation}: s = \sqrt{\frac{1}{n-1}\sum_{i=1}^{n}(xx_i - \bar{x})^2}$$

$$\bar{x} = \frac{1}{n}\sum_{i=1}^{n} x_i = \text{mean value}$$

1.2 Other Measures of Variability

- The variance is the mean square deviation of single observation from the mean, expressed as s^2.
- The coefficient of variation is a relative measure of the variability.

  $v = s/x$

- The range is the difference between the largest and the smallest value.

  $R = x_{max} - x_{min}$

1.3 Problems with the Homogenizing Efficiency e

Since the homogenizing efficiency is a function of s_α and s_β, the factor e can be low although the preblending bed or the homogenizing silo works satisfactorily, i.e. the fluctuations s_β of the homogenized material are within the tolerable limit. This is the case if the original fluctuations s_α are small. If, contrarily, s_α is very large, the fluctuation s_β can be unacceptably high although the homogenizing efficiency factor is large and within the specifications.

Due to this ambiguity of the factor e, it is better to specify the desired maximum value for the standard deviation s_β of the homogenized material.

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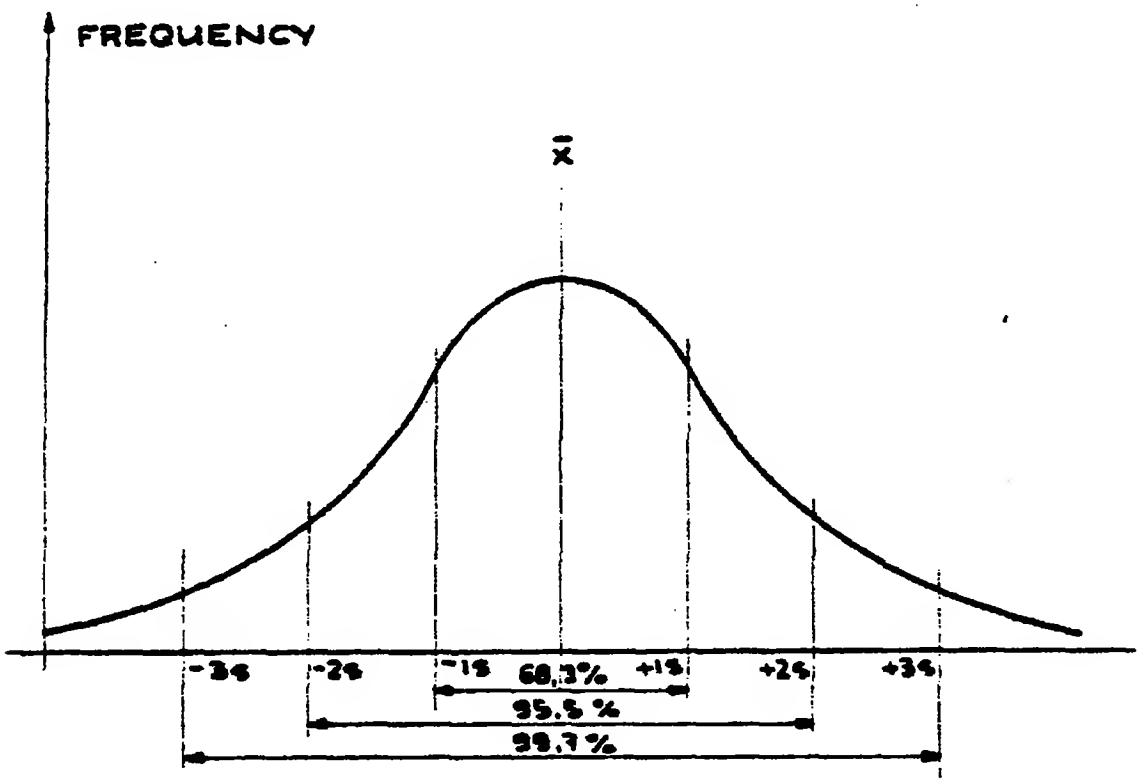
2. INTERPRETATION OF THE STANDARD DEVIATION

Practice has shown that in most cases the single observations follow a 'Normal Distribution' or a 'Gaussian Distribution'.

The 'Normal Distribution' is determined by the mean and the standard deviation shown in fig. 1.

Figure 1 'Normal Distribution'



For the normal distribution about 68.3 % of the observations are expected in the interval x +/- 1s, about 95.5 % in the interval x +/- 2s and 99.7 % in the interval x +/- 3s.

If the distribution is not 'normal', the standard deviation allows no direct interpretation. Therefore other criteria to interpret the variability are demanded (e.g. the range).

The following fig. 2 illustrates another presentation of the standard deviation, a plot related to time.

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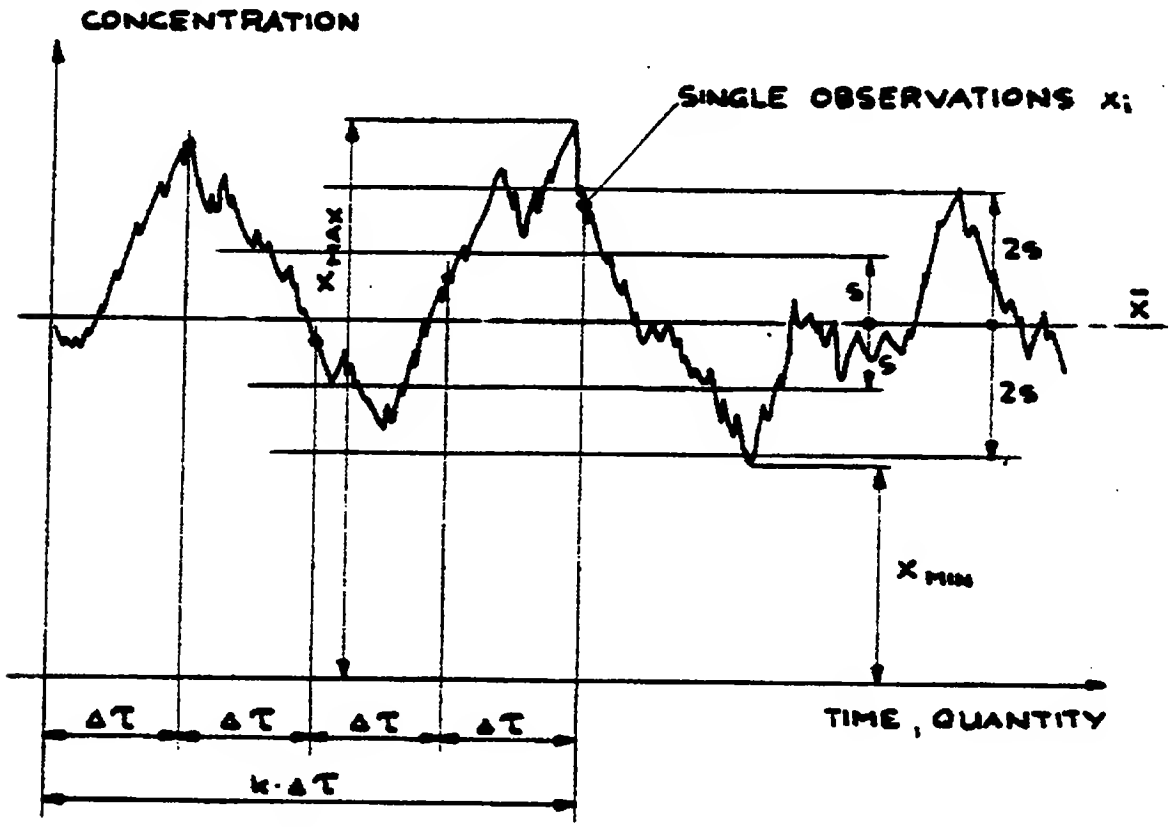
Figure 2 Chemical Values and Standard Deviation versus Time



Note: Normally the range of +/- 2s (about 95 % of all observations) is taken into consideration, that means a confidence interval of 95 % corresponding to +/- 2s.

2.1 Minimum Number of Samples Taken

To arrive at a meaningful sufficiently accurate standard deviation a minimum number of 30 better 50 samples is required.

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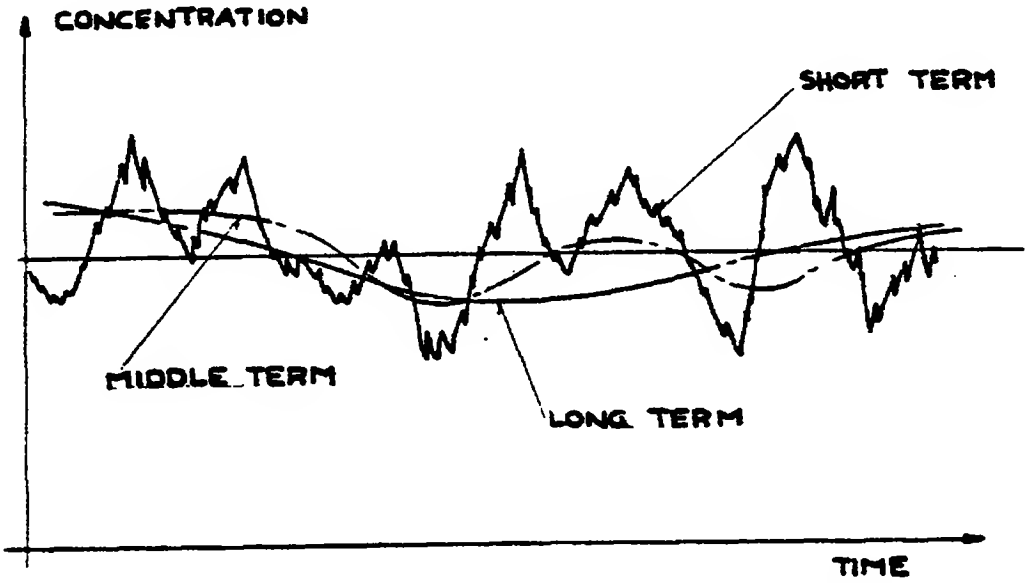
2.2 Estimation of s

A rough approximation of the standard deviation is achieved by taking



whereby single observations which are far out of the range are not considered.

3. CHARACTERIZATION OF CHEMICAL FLUCTUATIONS

A global feature to express the chemical fluctuations in a raw material composition is the 'standard deviation' as indicated in the previous chapter.

However, the standard deviation gives no indication about the time behavior, i.e. the frequency and amplitude of the fluctuations. Furthermore it is only applicable for statistically independent samples.

3.1 Interpretation of Statistical Independency

The samples of a series are statistically independent (uncorrelated) if, in the average, a subsequent sample is at the opposite side of the mean value. Contrarily, statistically dependent (correlated) means that, in the average, a subsequent sample always lays at the same side of the mean value. A mathematical description of this phenomena is given by the correlation function.

The presumption of statistically independent samples is only valid for raw material samples located in the quarry sufficiently apart of each other. The chemical difference of subsequent raw material samples is reduced with decreasing distance between the samples and from a certain point the latter become statistically dependent.

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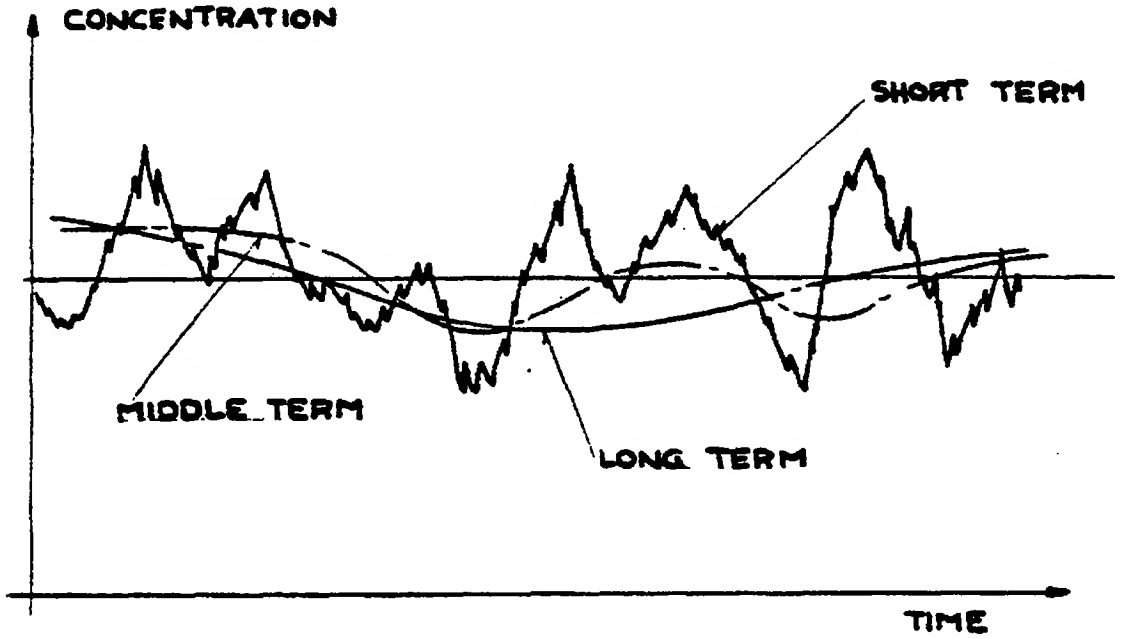
Figure 3 Type of Chemical Fluctuations



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3.2 The Correlation Function

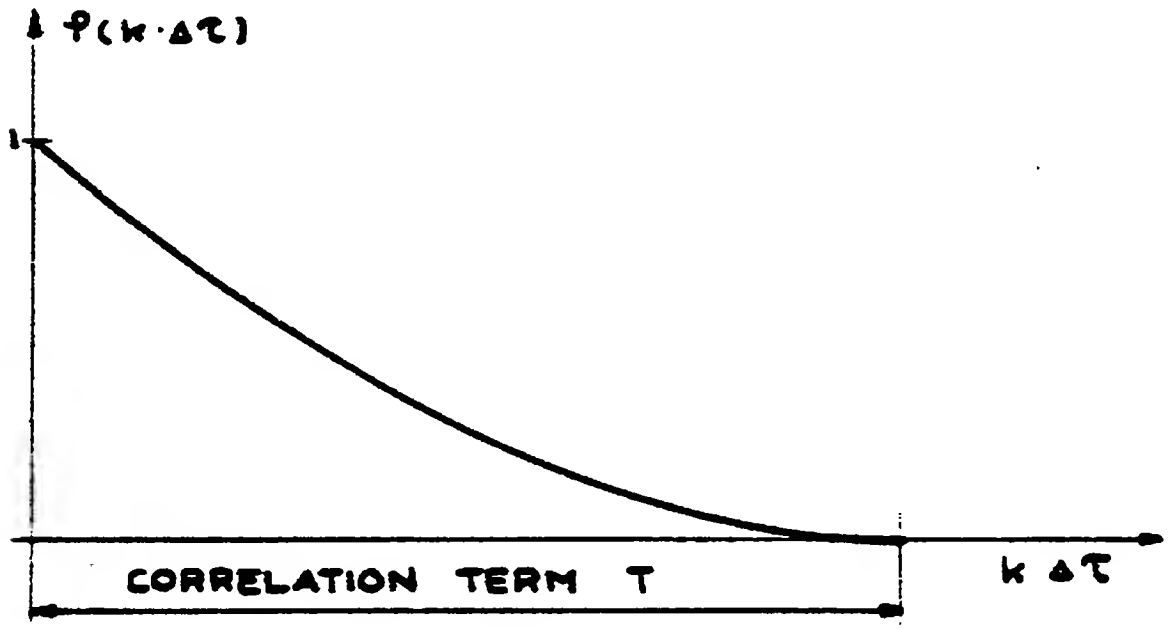
The correlation function indicates the statistical dependency of the single samples in a series. It is calculated in the following way:

$$\rho(k \bullet \Delta\tau)\left[\frac{1}{n-k-1}\sum_{i=1}^{n-k}(x_i-\bar{x})(x_{i+k}-\bar{x}\right]$$

$$\text{var} = s^2 = \frac{1}{n-i}\sum_{i=1}^{n}(x_i-\bar{x})^2$$

where:
k = parameter for the number of $\Delta\tau$, indicating the distance related to time or material quantity between single samples
I = parameter indicating single samples
n = number of samples
$\Delta\tau$ = distance or material quantity between single samples

Figure 4 Correlation Function



Interpretation:

$k\,\Delta\tau = 0$	full correlation (1)
$k\,\Delta\tau > 0$	correlation (dependency) is valid
$k\,\Delta\tau \geq T$	statistically independent

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4. HOMOGENIZATION

Homogenization is achieved by two procedures:

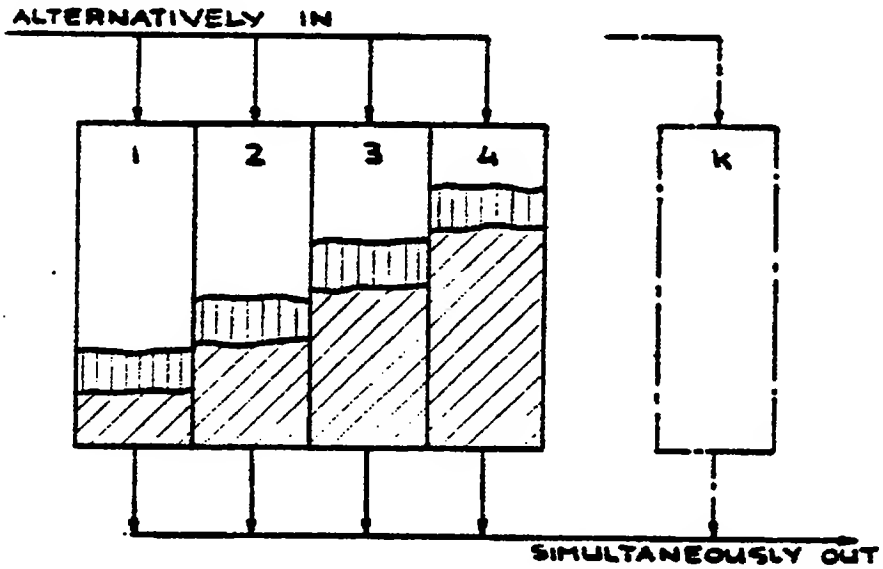
- By 'blending' where two or more material components are combined. The materials are fed alternately or time delayed in a common and reclaimed simultaneously afterwards.
- By 'mixing' where two or more different material components are fed in a common system and agitated such as to achieve one homogeneous component.

4.1 Blending

4.1.1 Model

The following model idea describes the principle of 'blending'.

Figure 5 'Blending' Model



Model description

The portions $\Delta\tau \cdot m_{in}$ (where m_{in} is the average input mass flow) are entering the system subsequently succeeding in time intervals $\Delta\tau$ from compartment 1 to k. Due to the different material levels in the compartments the simultaneous extraction effects a time delayed combinement of single material portions $\Delta\tau \cdot m_{in}$.

4.1.2 Homogenizing Efficiency of Blenders

Due to the special filling procedure of the individual silos and assuming a sufficiently large $\Delta\tau$ it can be expected that the raw materials extracted from each silo are statistically independent (uncorrelated).

Therefore the blending efficiency approaches the factor

$$e = \frac{1}{\sqrt{k}}$$

If the material components to be blended are statistically dependent (correlated), the blending efficiency can be much below above value.

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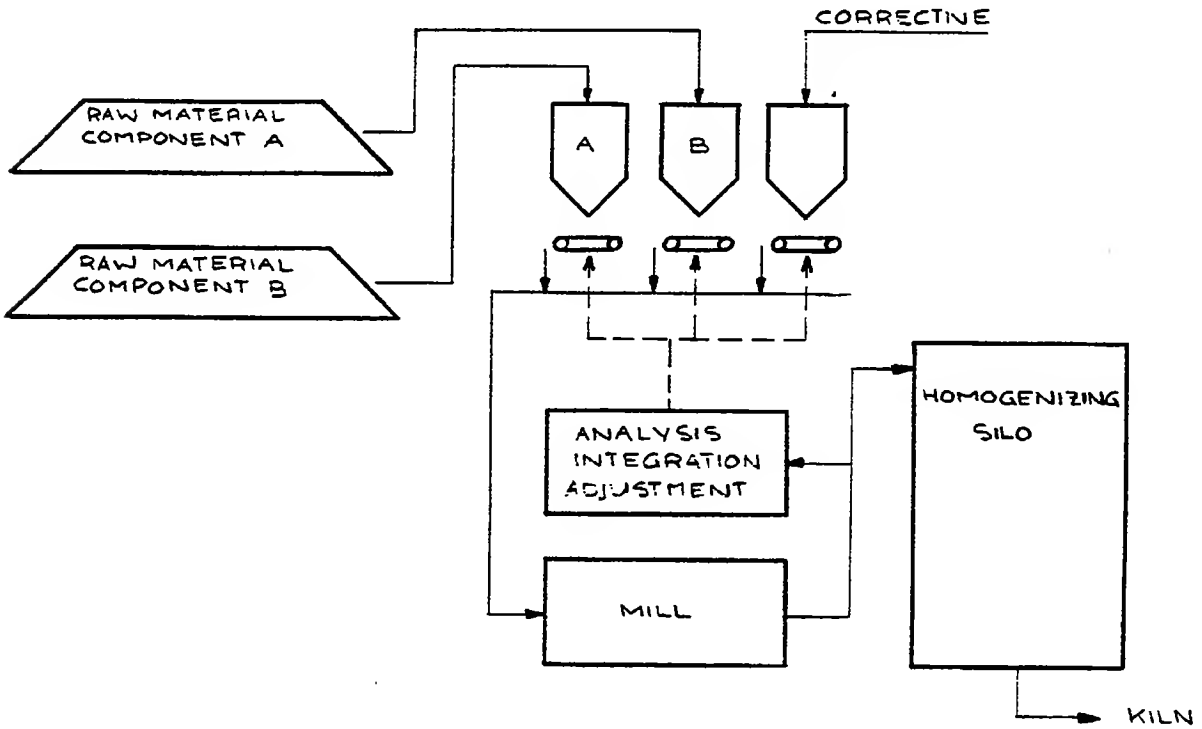
4.1.3 Examples

- 'Blending' at the quarry
  From two or more quarry faces the materials are fed simultaneously to the crusher (selective quarrying).
- 'Blending' in a prehomogenizing bed
  The raw material is stacked in several layers into the bed and reclaimed in a way that all layers are simultaneously cut, i.e. 'preblended'.

Figure 6 Raw Mix Proportioning

- 'Blending' of the raw mix by proportioning



Note: The final raw mix is prepared by proportioning single raw material components. This procedure effects the combinement of the single material components, and thus 'blending' resp. dampening of their original chemical fluctuations is achieved. Of course by wrong adjustments of the components, also very large errors (additional fluctuations) can be introduced into the raw mix.

- 'Blending' of raw meal
  Raw meal simultaneously extracted from parallel bins which have been filled subsequently (time delayed) as shown on the model (usual solution before the large raw meal homogenizing silos were known).
- 'Blending' of slurry
  Extracting from several parallel slurry tanks and feeding into a common storage basin.
- 'Blending' of coal
  (If no special preblending bed is provided).
  Stacking the coal in longitudinal rows and reclaiming it from the front with wheel loaders.

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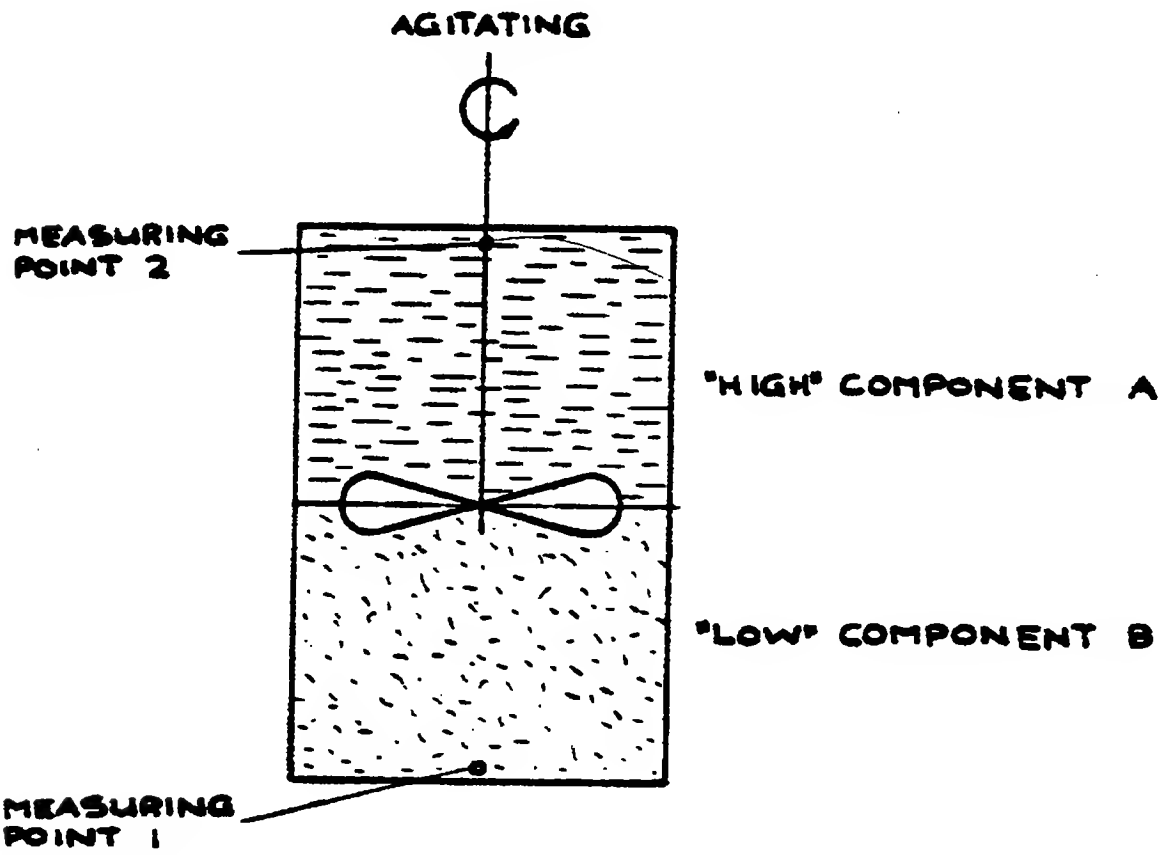
4.2 Mixing

4.2.1 Model

The following model idea describes the principle of mixing which is done by agitation.

Figure 7 Mixing Unit



Model description

It is assumed that two components (A) and (B) are fed in a mixer so that they first are completely separated. By agitating, the components are homogenized (fig. 7).

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4.2.2 Homogenizing Efficiency of Mixers

The homogenizing efficiency is shown by the plots below (fig. 8 and 9) where the measuring points 1 and 2 are periodically sampled and analyzed.

Figure 8 Analyses of Measuring Point 1

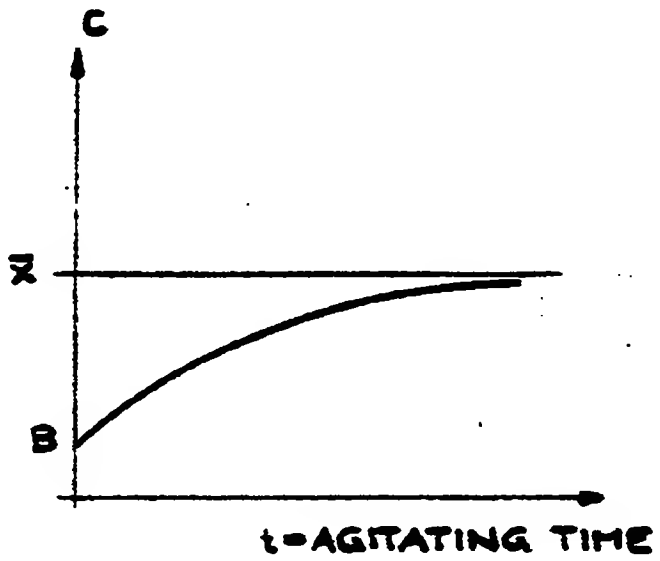
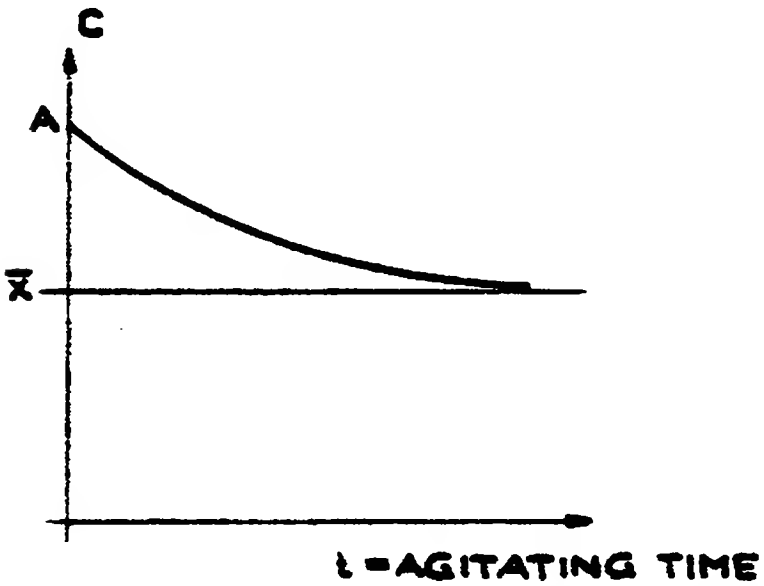


Figure 9 Analyses of Measuring Point 2



Similar measurements on several systems have confirmed that above plots follow the so-called 'ideal blender' curve which is mathematically described as follows:

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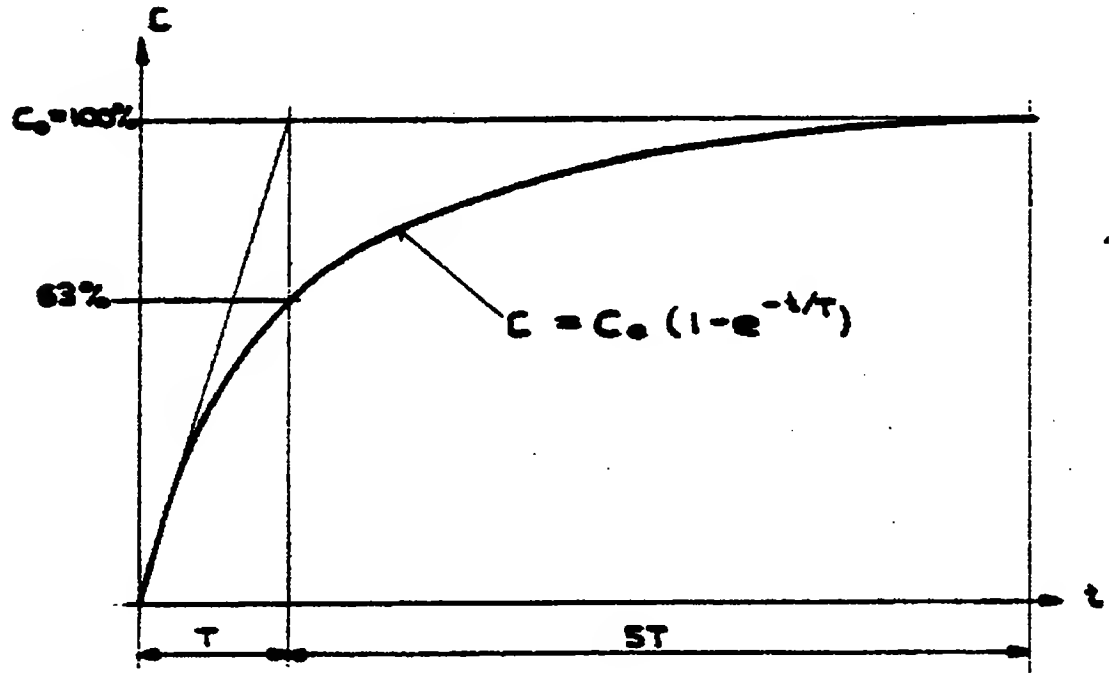
Figure 10 'Ideal Blender' Curve



where: T = Characteristic agitation time

The mixer shall be agitated approx. 5 times (T) so as to achieve the optimum homogenizing efficiency.

Comment

Mixing units are operated either batchwise or continuously. In both cases mixers provide very good homogenizing efficiencies (see figures above).

However chemical fluctuations larger than the batch volume (in case of batchwise operation) or the mean retention time, i.e. v / v_f, v = volume of the mixer, v_f = volume flow, (in case of continuous operation) are not homogenized and pass the mixer undampened.

In case of continuous mixing it has to be assured that the mean retention time is larger than the required characteristics agitation time T.

The mixing should also not be made too long since after a certain mixing period the mix starts segregating again.

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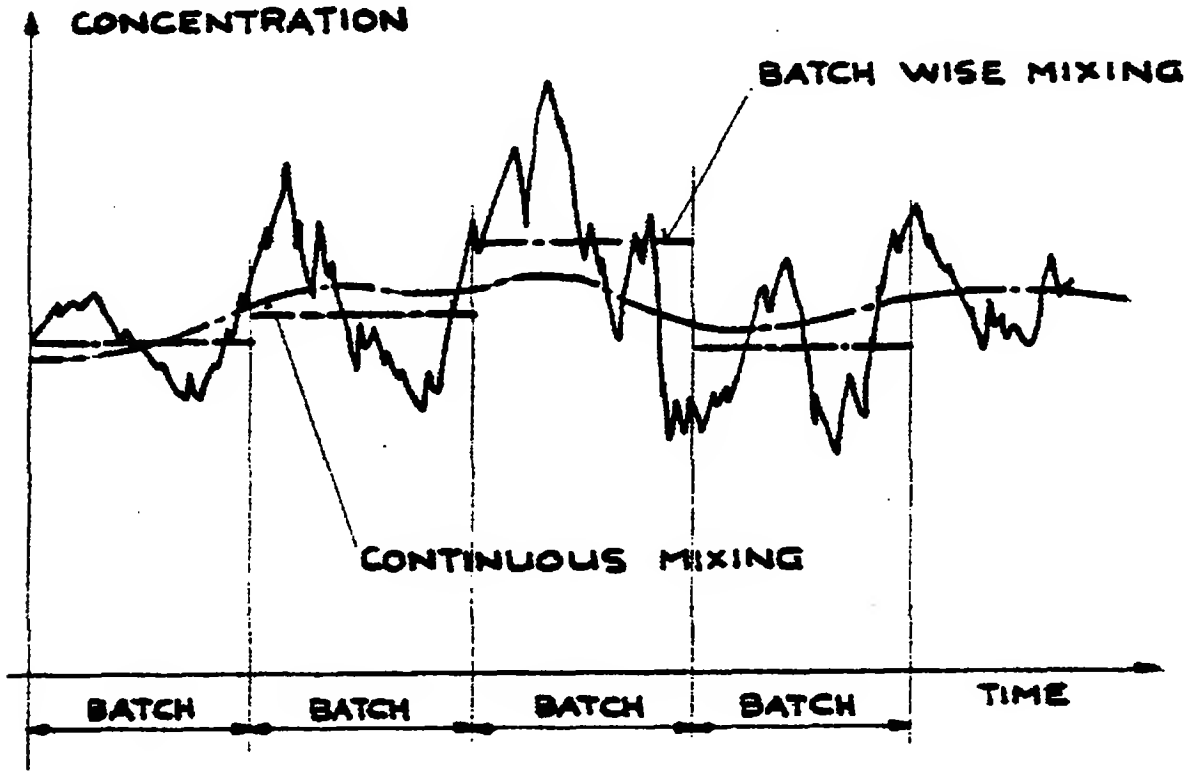
Fig. 11 illustrates the mixing behavior between batch and continuous operation.

Figure 11 Batchwise - Continuous Mixing



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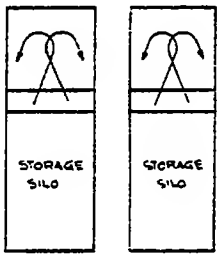
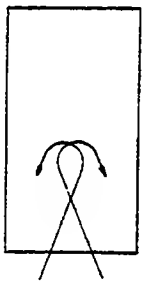
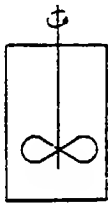
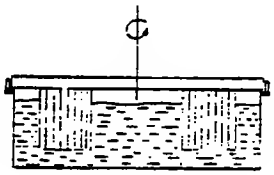
4.2.3 Examples

Homogenizing Element	Homogenization	
	Batch Type	Continuous
Homogenizing Silo		
Slurry Tank resp. Basin		
Mill		

4.3 Homogenizing Elements

Homogenization takes place during the whole raw material preparation process. It starts already in the quarry where blending occurs by combining different components at the crusher feed and ends at the bin feed where raw meal from several silos is extracted.

During the total process the following homogenizing elements can be met:

Homogenizing element	Homogenizing procedure	Type of homogenized fluctuations
Selective quarrying	Blending	Long-term to middle-term fluctuations
Preblending	Blending	Middle-term to long-term fluctuations
Proportioning	Blending	Long-term to middle-term fluctuations
Grinding	Mixing	Short-term fluctuations
Homogenizing	Mixing / blending	Short-term to middle-term fluctuations

4.3.1 Definition of Fluctuations

The type of fluctuation as generally outlined in chapter 3, fig. 3, can be refined more specifically in time relation as follows:

Short-term fluctuations: min., h
Middle-term fluctuations: 1 - 5 days
Long-term fluctuations: 5 - 7 days

4.3.2 Comment on the Homogenizing Elements

4.3.2.1 Selective quarrying

If the quarry operation is well organized in an integrated preblending bed concept and the raw material components are located not too far away, premixing of certain material qualities by alternate crusher feed can reduce long and middle term fluctuations to hourly or daily fluctuations.

4.3.2.2 Preblending

Depending on the size of the preblending bed, fluctuations within the rage of 5 - 7 days (preblending bed capacity) are normally blended.

4.3.2.3 Proportioning

More component preblending bed concepts incorporate the raw material qualities separated in two or more preblending beds. By proportioning the single components to the final raw mix already blended, fluctuations within pile capacity (approximately one week) or even exceeding pile capacity can be combined (it means blended). As previously mentioned also very large errors (additional fluctuations) by wrong adjustments can be introduced into the raw mix.

4.3.2.4 Grinding

If grinding is considered as a mixing element it holds true only for short time fluctuations which are related to the retention time on material particles passing through the mill.

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4.3.2.5 Homogenizing silo

In the continuous homogenizing silo fluctuations of few hours depending on the inactive volume of the silo, are mixed which is equal to the retention time of material passing through the silo. Whereby in the batch type homogenizing silo the certain batches are first mixed and combined afterwards in the succeeding storage silo. There, fluctuations amounting up to days are blended.

5. DETERMINATION OF CHEMICAL FLUCTUATIONS

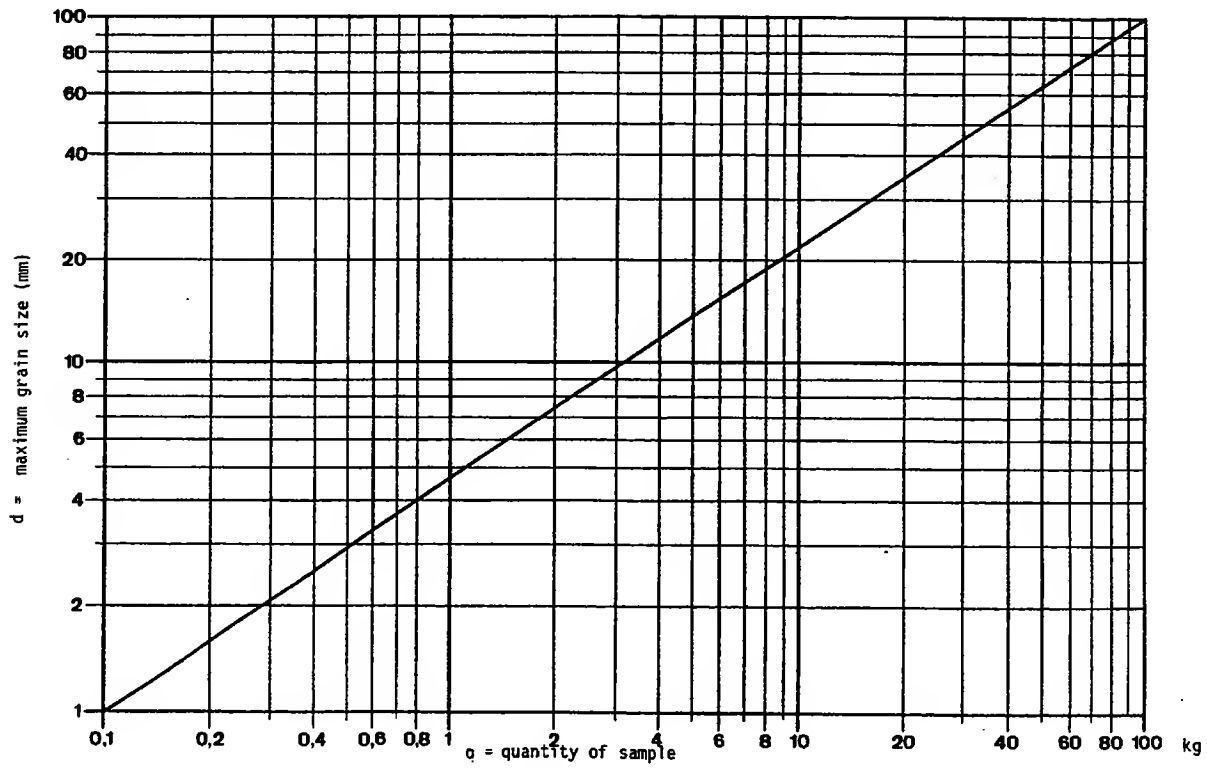
The determination of the chemical raw material fluctuations is mandatory in order to select the appropriate homogenizing elements and to dimension them correctly. Furthermore it allows to judge their homogenizing efficiency.

Chemical fluctuations are detected by taking samples. The required sample number depends on the quantity and the homogeneity of the material to be sampled. The required number of samples rises with increasing inhomogeneity of the raw material. With an increasing number of samples also the sampling reliability improves. To be representative, the sample must be collected in an appropriate quantity depending on the maximum grain size. The relation between maximum grain size and necessary sample quantity is given in fig. 17 as guide line.

Please note that the error of analysis has to be deducted in the following way:

$$s_{true} = \sqrt{s^2_{measured} - s^2_{analysis}}$$

Figure 17 Relations between the Maximum Grain Size and the Corresponding Necessary Quantity of Sample



5.1 Investigation of Chemical Fluctuations in the Quarry

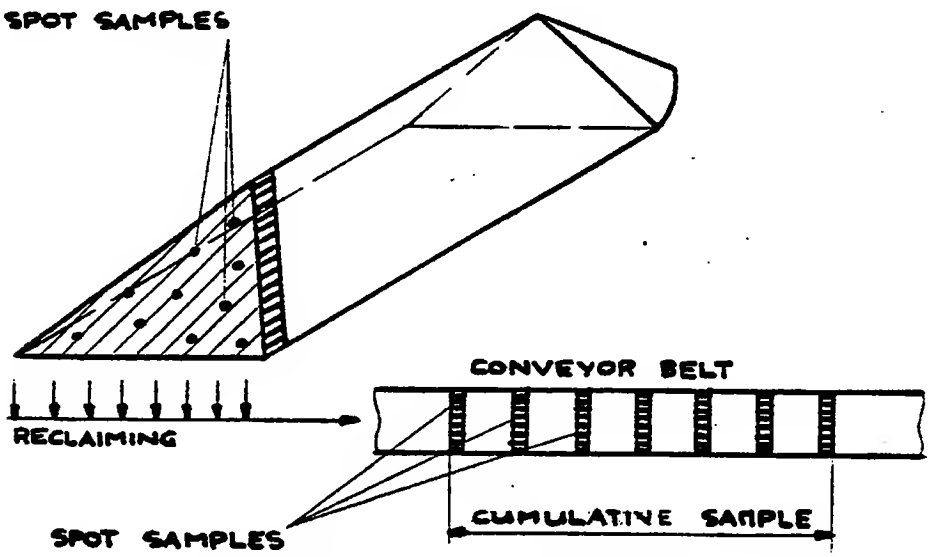
On existing plants the chemical fluctuations of the quarried material are detected by hourly samples taken after the crusher e.g. on a belt transfer point. In order to get a representative sample, the whole cross section of the transport belt ought to be sampled. Another, less accurate method is to sample the dust of the blast drill holes.

On plants in projection the chemical fluctuations of the future quarry must be determined with the aid of the drill hole prospection which is executed to assess the future raw material deposit. The chemical analyses of all drill hole cores are taken to calculate the standard deviation. If the drill holes cover the total deposit area, sufficiently accurate results can be expected for the overall chemical fluctuations of the quarry. The absolute accuracy is only dependent on the number of realized drill holes and the homogeneity of the deposit. However, in the raw material preparation process not the whole quarry fluctuations are relevant but only the fluctuations within one blast approximately or within the raw material quantity of one preblending bed. Usually the raw material deposit shows some stratification so that the chemical fluctuations are not equal in all three directions, i.e. horizontally (length and width) and vertically. Thus, with the application of a certain quarrying method, the chemical fluctuations within one blast are reduced as compared to the overall quarry fluctuations. In order to assess this reduction, the chemical fluctuations in each direction shall be calculated. From these calculations the standard deviation of an individual blast must be estimated.

5.2 Chemical Fluctuations after the Preblending Bed

The sample procedure after the preblending bed takes place in the following way (fig. 18).

Figure 18 Sampling of the Preblending Bed



In order to judge the fluctuations at the exit of the preblending bed the different cross sectional slices of the pile have to be compared. It is that within one slice approximately 6 to 9 spot samples have to be taken from which one cumulative sample is made. The samples can be taken from the exit belt leaving the preblending bed on an equal distance with a quantity per spot sample of about 10 kilograms. The cumulative sample has to be prepared and representatively divided down to 8 to 10 kilograms from which the analysis in the laboratory is made representing then the concentration value of one cross sectional area.

In this described way 60 to 80 cumulative samples distributed over the whole stock pile must be taken. Each sample must represent approximately the same stockpile mass.

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From all cumulative samples the mean value and standard deviation is calculated.

Note: An achievable homogenizing efficiency of the preblending bed obtained by the above sampling procedures (under the circumstance of normal raw material qualities) amounts to

$e = s_\alpha / s_\beta = 5 - 6$

However, not the homogenizing efficiency e is important. But to judge the effectiveness of the preblending bed, the most interesting value represents sβ which is in normal case expected to be below 1 % $CaCO_3$.

5.3 Chemical Fluctuations after the homogenizing silo

To assure an appropriate representativity of the sample the following considerations are essential:

- The sampling location is preferably on a meal discharge point where the total material flow is accessible, if necessary by taking some subsamples in short time intervals, which afterwards are combined to one 'spot sample'.
- The samples themselves should be spot samples or 'spot samples' as described above, evenly distributed over the total sampling period.
  - Period for continuous homogenizing silo: 24 h (e.g. every 30 min. one sample = 48 samples).
  - Period for batch homogenizing silo: should contain 3 batches (e.g. 3 time 8h = 24 h again every 30 min. one sample = 48 samples).
- The sampling quantity per spot sample amounts to approx. 1 kg which then is homogenized and divided.

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Preblending of Raw Materials

F. Bucher
VA 83/5005/E

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Summary

Raw material components incorporating chemical fluctuations with a standard deviation of s > 2 % $CaCO_3$ should definitely be blended in preblending beds.

Various mathematical models and methods have been developed to determine the homogenizing efficiency of the preblending bed in function of the primary chemical material fluctuations, the number of layers and the geometry of the preblending bed.

The principle of blending is simple as the material from the quarry is stacked in layers to a pile and reclaimed in right angles to the layers in thin slices.

The mostly applied preblending system utilizes Chevron stacking (or possibly Chevcon stacking in case of a circular preblending bed) and front reclaiming.

The stacking machines are either throw-off carriages under the ridge of the roof (for covered longitudinal piles) or laterally traveling boom stackers (for open air stock piles) or slewing stackers (for circular preblending beds). The highest travel speeds for stackers attain 25 - 30 m/min. corresponding to 400 - 500 layers with a pile width of 30 m and a stacking capacity of 1000 t/h.

The reclaimers usually consist of bridge-mounted scraper chains, equipped with moving harrows or sweeping ropes to bring the stacked material to the base of the pile. The largest reclaiming capacities are approx. 500 m^3/h, the span width of the bridge being up to 50 m.

For sticky materials and for very small storage capacities are still the 'homogenizing pit' or, in case of low homogenizing efficiency requirement, the well known side reclaimer is applied.

If preblending piles have to be covered, circular beds are preferable since roofing can be done at relatively low cost a dome-type structure.

The integrated preblending bed is used if different materials (two or more components) are dumped on the same pile in a controlled manner so as to attain a predesired chemical value for the whole pile. This goal is achieved by continuously sampling and analyzing the material feed to the pile, the amount of sampled material being about 1 % of the total.

The homogenizing effect of a preblending bed is approx. 3 : 1 to 6 : 1.

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1. INTRODUCTION

Raw material components incorporating chemical fluctuations with a standard deviation of s ≥ 2 % $CaCO_3$ should definitely be blended in preblending beds.

For fluctuations with s ≤ 2 % $CaCO_3$, no preblending is required since the discontinuous batch type homogenizing silo provides a sufficient homogenizing efficiency.

However, preblending beds are also installed for other reasons than for their homogenizing efficiency:

- The preblending bed represents a fully automatic raw material storage. There is no difficulty in obtaining also very large storage capacities.
- Decoupling of the quarry operation from the plant, thus avoiding a multiple shift operation in the quarry and allowing for the application of bigger quarrying machines.
- There is no problem in handling also very sticky materials in preblending beds.
- Selective quarrying and premixing of different raw material components is easily done in the preblending bed.

2. HOMOGENIZING EFFICIENCY OF PREBLENDING BEDS

2.1 Principle of Preblending

The operation principle of the preblending bed is very simple:

The raw material from the quarry is stacked in layers to a pile and reclaimed in right angles to the layers in thin slices. This principle can be put into practice by various combinations of pile structure and extraction processes. The homogenizing process in a preblending pile takes place in the following way:

a) As a result of layer-by-layer stacking, the raw material flow is divided into sub-quantities of $\Delta\tau$ tons (fig. 1).

b) Because of this the characteristic variations which occur in the raw material flow are also divided into equal intervals $\Delta\tau$ (fig. 1).

c) As a result of the layered build-up of the blending bed the variations within the sub-quantities $\Delta\tau$ are superimposed.

d) On considering cross-sections at right angles to the layers, it is seen that, because of the above-mentioned superposition, there occur altered characteristic variations within the cross-sections and variations from one cross-section to another, i.e., between the cross-sections.

Note: Material of subsequent layers are only insignificantly different, in the region of the stacker reversing points (fig. 2) thus for the purpose of calculation only every second layer is considered (i.e. every 'double layer').

Fig. 1 to 4 illustrate the working principle of the preblending bed.

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Figure 1 **Raw Material Fluctuations at the Inlet of the Preblending Bed**

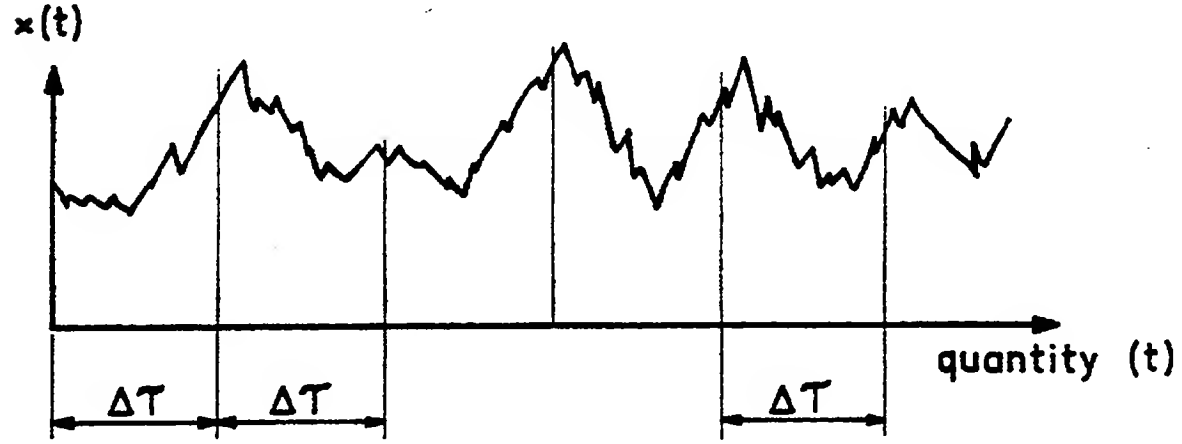

Figure 2 **Build-Up Model of the Preblending bed**

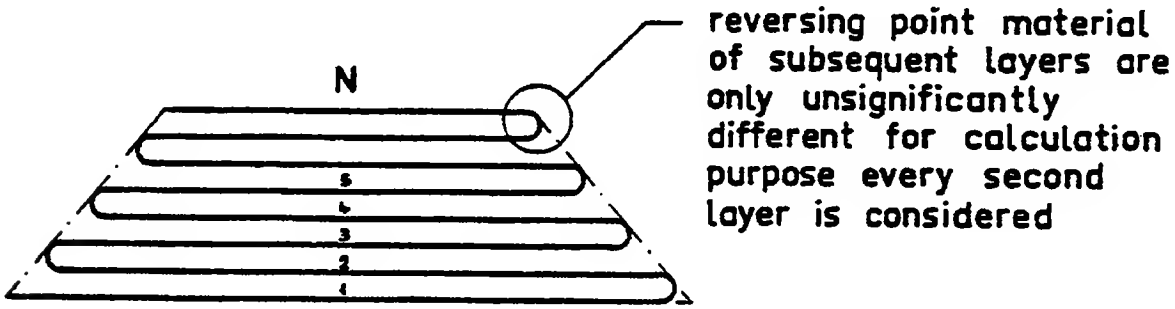

Figure 3 **Model Illustrating Build-Up and Reclaiming**

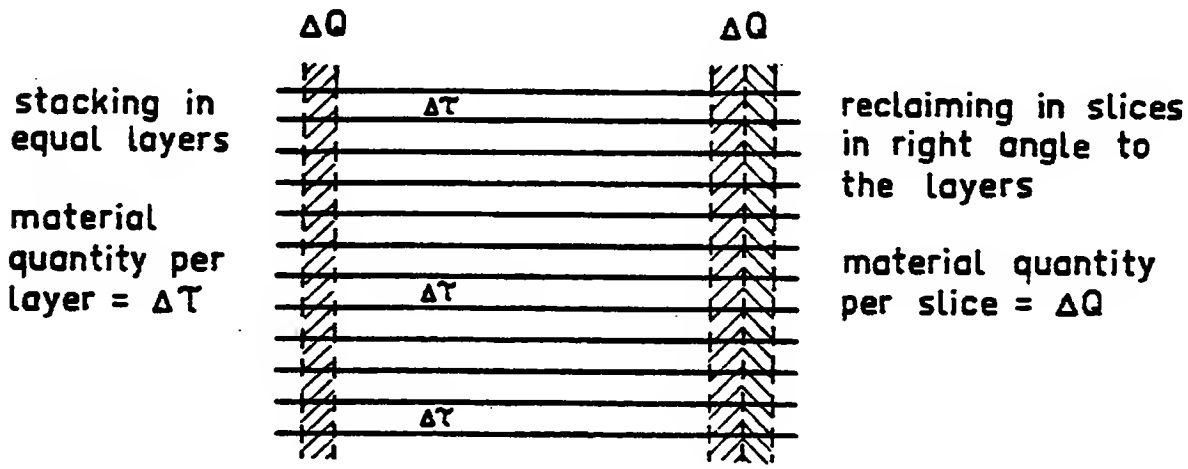

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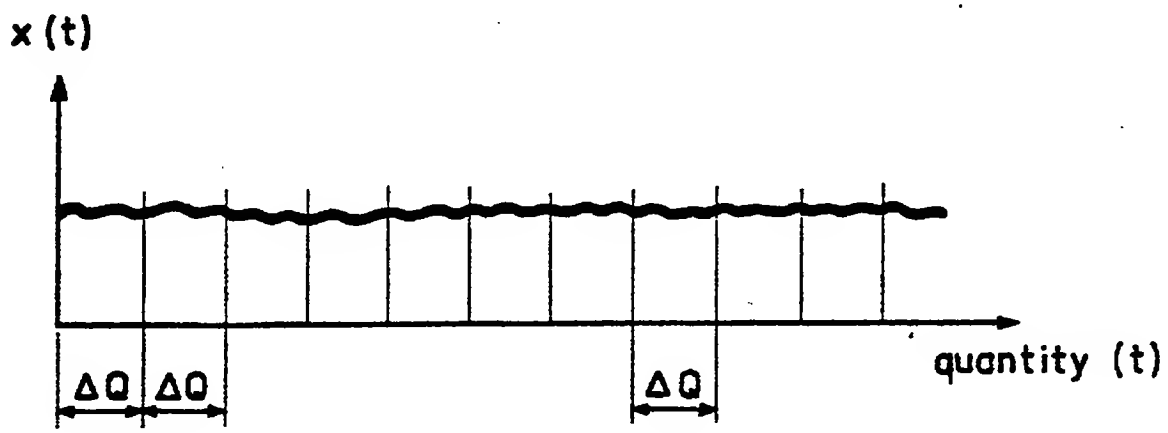
Figure 4 Raw Material Fluctuations Smoothed out on Leaving the Blending Bed



2.2 Mathematical Models for the Homogenizing Efficiency

Various mathematical models and methods have been developed to determine the homogenizing efficiency of the preblending bed in function of the primary chemical material fluctuations, the number of layers and the geometry of the preblending bed. In the following short summaries of the most important methods are provided.

2.2.1 Method 1

Assumption: The raw material properties of the (N) 'double layers' are:

* normally distributed
* statistically independent

Based on the above assumption the basic relations of statistics will hold

$S^2\beta = S^2\alpha \ 1/N$

Thus the homogenizing efficiency is only a function of the realized number of 'double layers' (N):

$$e = \frac{s_\alpha}{s_\beta} = \sqrt{\frac{s_\alpha^2}{s_\alpha^2 \frac{1}{N}}} = \sqrt{N}$$

Comment

The assumption of statistically independent raw material 'double layers' holds true only if a small number of thick 'double layers' is realized. As soon as the number of 'double layers' is large adjacent layers become statistically dependent (correlated) and the achievable homogenizing efficiency is below $\sqrt{N}$.

The practical application of this method is thus rather limited.

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2.2.2 Method 2

2.2.2.1 Correlation Function

The model below considers the correlation between the material 'double layers' in the preblending bed and is thus an important improvement compared to model 1. The model derived in the following form was invented by Van der Mooren (see (2) (3)).

$$e = \frac{s_\alpha}{s_\beta} = \left[\frac{1}{N} + \frac{2}{N^2}\sum_{k=1}^{N-1}(N-k)\rho_\alpha(k\Delta\tau)\right]^{\frac{1}{2}}$$

where:

s_α = input fluctuations to the preblending bed
s_β = output fluctuations out of the preblending bed
N = number of 'double layers'
ρ_α = correlation function
k = parameter for the number of $\Delta\tau$, indicating the material quantity between single 'double layers' of the preblending bed
$\Delta\tau$ = material quantity of one 'double layer' of the preblending bed

Comment
In practice the application of this model is mostly not possible since the correlation function $\rho\alpha$ of the raw material to be blended is not known.

2.2.2.2 Frequency Response

The same model was described also by the frequency response (see (1)) in the following way:

$$|G(f)|^2 = \frac{1}{N}\left[\frac{\sin(N\pi\Delta\tau f)}{\sin(\pi\Delta\tau f)}\right]$$

where:

G(f) = frequency response
f = frequency
N = number of 'double layers'
$\Delta\tau$ = material quantity per 'double layer'

Comment
This form of the model is well suited in case that the whole material preparation chain including preblending, raw mix proportioning and raw meal homogenizing is treated. Each element of the chain is then described by its individual frequency response and connected to one large control and homogenizing model. Of course, for the application of this form of the model the similar problems arise as mentioned above. In this case not the correlation function $\rho\alpha$ is usually unknown but the spectrum of the chemical input fluctuations.

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2.2.3 Method 3

This model represents a simplified version of the model 2 by assuming the shape of the correlation function $\rho\alpha$.

Assumption: $\rho\alpha\ (k\ \Delta\tau) = \exp\ (-\ k\ \Delta\tau / T)$

where:

$\rho\alpha\ (k\Delta\tau)$	=	correlation function
$\Delta\tau$	=	material quantity of one 'double layer' of the preblending bed
k	=	parameter for the number of $\Delta\tau$ indicating the material quantity between single 'double layers' of the preblending bed
T	=	correlation term

Using above assumption in the formula of model 2 for $\rho\alpha$ and setting $\Delta\tau = \frac{M}{N}$, the following simplified model arises:

$$e = \frac{s\alpha}{s\beta} = \left[\frac{1}{N} + \frac{2}{N^2} \frac{\exp - \left(\frac{M}{NT} \right)}{\left[1 - \exp\left(\frac{M}{NT} \right) \right]^2} \left[N \left[1 - \exp\left(-\frac{M}{NT} \right) \right] - \left[1 - \exp\left(-\frac{M}{T} \right) \right] \right] \right]^{\frac{1}{2}}$$

where:

e = homogenizing efficiency
s_α = input fluctuation into the preblending bed
s_β = output fluctuation out of the preblending bed
N = number of layers
M = preblending bed capacity (approx. 1 week)
T = correlation term

Correlation term T

The correlation term T indicates the time period or the material quantity of the unhomogenized material, within which the statistical correlation of single elements will hold.

The correlation term can be estimated by determining the time or the material quantity during which the real curve of chemical concentration cuts the line of its mean value (see fig. 5).

Expressed in formula:

$$T \approx \frac{\text{total time, quantity or lenghts}}{\text{number of cuts}} \quad [\text{h, t or m}]$$

The term T can be expressed in time (h), material quantity (t) or length (m) according to the further application.

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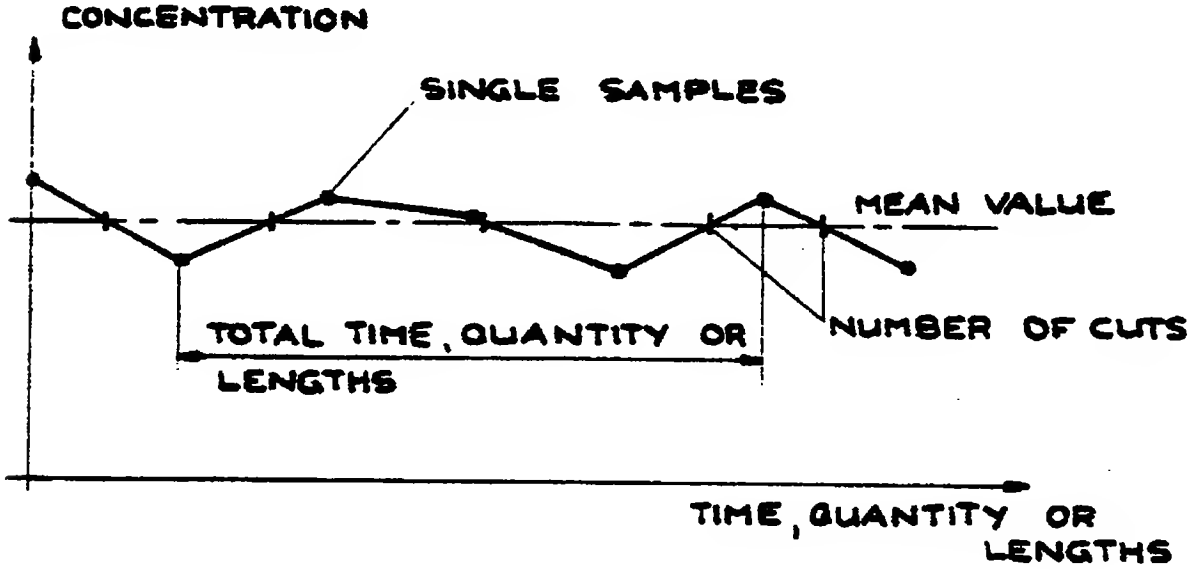
Figure 5 Determination of Correlation Term T



Examples:

- On an existing plant, T is determined by taking samples from the belt conveyor.
- On a plant in projection, T has to be estimated with the aid of the drill hole prospection:

1) The length of T is determined in the main directions where the quarrying is done, e.g. vertical and horizontal if exploitation is done by blasting of benches.
2) Afterwards the 'correlation quantity' is calculated with the aid of the correlation length in each direction, e.g.:

correlation length over the width:	20 m
correlation length over the depth:	5 m
chosen burden during blasting:	3 m
correlation quantity ($\rho = 2.5$ t/m^3):	$T = 20 \cdot 5 \cdot 3 \cdot 2.5 = \underline{750\ t}$

Note: In the same way also the relevant standard deviation of the unhomogenized raw material has to be estimated (see chapter 'aspects of raw material homogenization', section 1).

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2.2.4 Method 4

Often no or only scarce information concerning the raw material properties are available. In such a case, the number of layers N in the preblending pile shall be chosen according to the following criteria:

If required homogenizing efficiency is high: N = 400 - 500
(rather inhomogeneous raw material)

If required homogenizing efficiency is not important: N = 200 - 300
(rather homogeneous raw material)

Note: The maximum number of layers will depend on the stacking performance, the pile cross section and the maximum travel speed of the machine. In most cases however it does not pay to work with such large number of layers because - irrespective of the correlation of the raw material - the effective statistically independent number of layers are smaller than the actual, i.e. for calculation purpose two layers = one 'double layer' (see 2.1 / 2.2.1).

3. LONGITUDINAL PREBLENDING BED

The longitudinal preblending bed represents the most common and simplest type of preblending bed. This type always corresponds to batchwise blending since one pile has to be formed while the other is reclaimed.

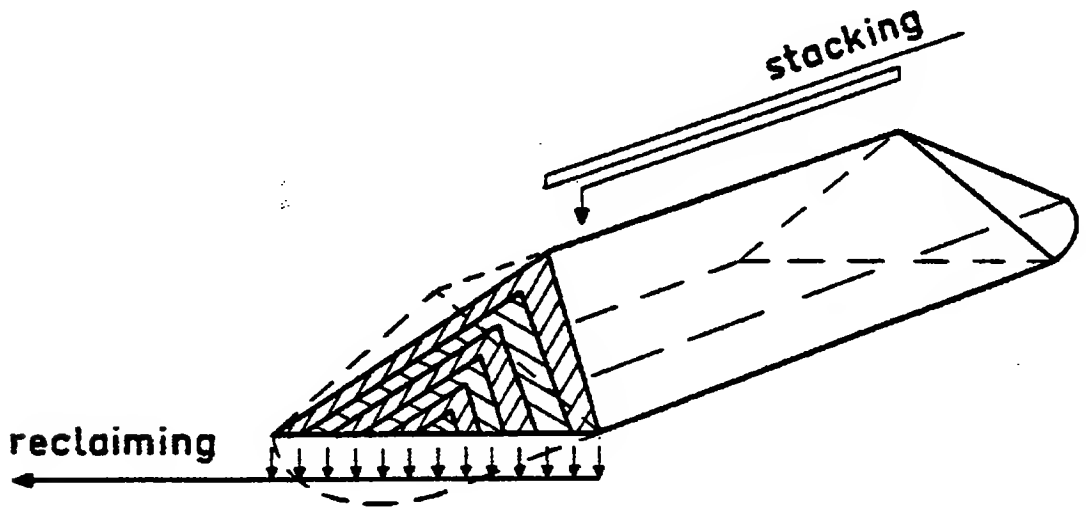
3.1 Stacking / Reclaiming Methods

Different stacking / reclaiming methods are shown in fig. 6 - 9, the Chevron Windrow and Conical Shell stacking as well as the stacking of the homogenizing pit. Reclaiming is normally done by a front acting reclaimer. An exception is the side reclaimer (see chapter 3.4.2).

3.1.1 Chevron Stacking / Reclaiming

Longitudinal layers are placed by a to and fro stacker movement over the ridge of the pile until the full height of the triangular pile is reached. The material is afterwards reclaimed in slices transversal to the pile by means of a front acting reclaimer. This is the common stacking method for a longitudinal pile with front reclaiming (see fig. 6).

Figure 6 Chevron Stacking



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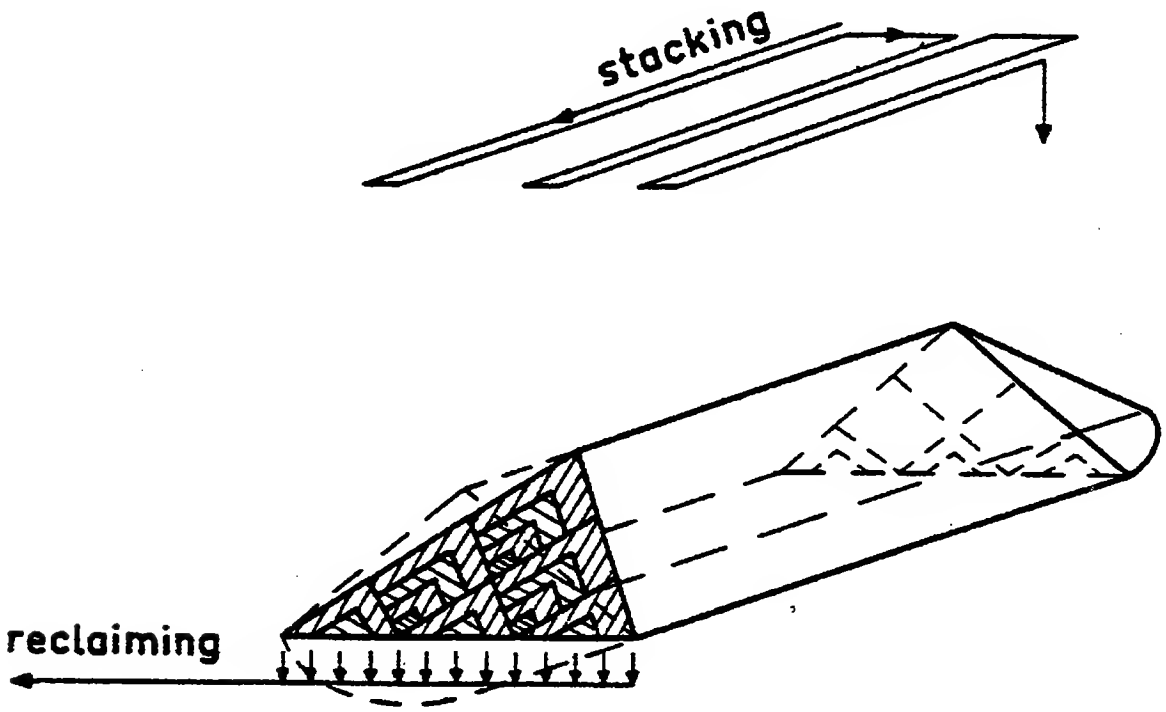
3.1.2 Windrow Stacking / Reclaiming

Several longitudinal layers are stacked parallel over the whole width of the pile in such a way that they form a triangular pile face after completing the full height. In other words the stacker not only makes longitudinal but also transversal movements so that a pattern of parallel, longitudinal rows arises. Reclaiming is always done by a front acting machine (see fig. 7).

Figure 7 Windrow Stacking



Note: Due to the higher investment cost as compared to Chevron, the Windrow stacking is only applied if a large segregation in the end cones is expected (e.g. max. Grain size > 100 mm).

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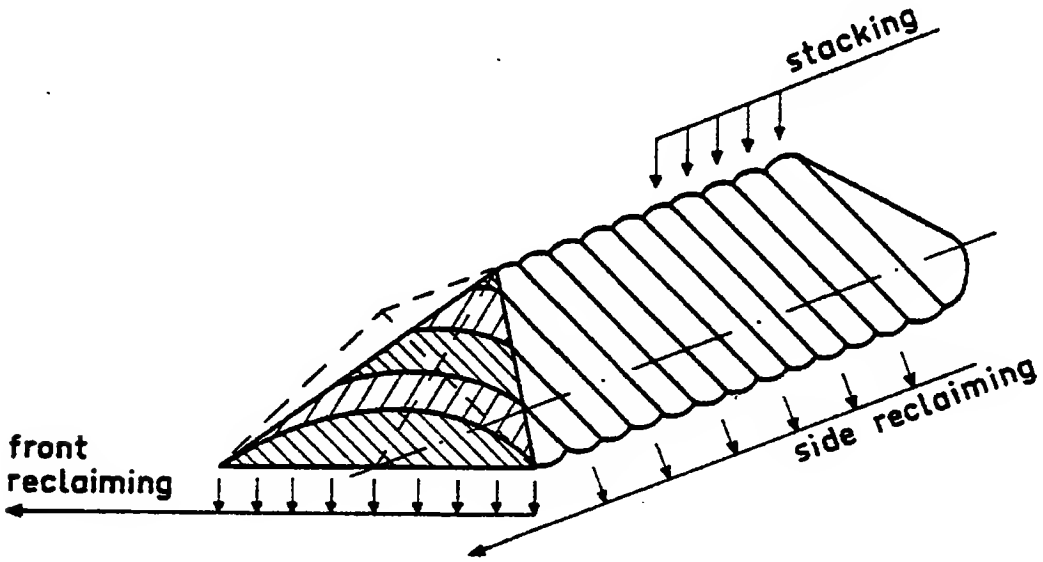
3.1.3 Conical Shell Stacking / Reclaiming

The stacker moves stepwise in longitudinal direction. The next step is only executed after completing the Conical shell up to the full pile height. Reclaiming is normally done laterally by a side-reclaimer (see fig. 8).

Figure 8 Conical Shell Method



Note: The Conical shell stacking method should not be applied in conjunction with front reclaiming since only few material layers are blended so that the achievable homogenizing efficiency is rather poor. The method is used for preblending beds with side reclaimers in case of sticky materials and if the homogenizing efficiency is not important.

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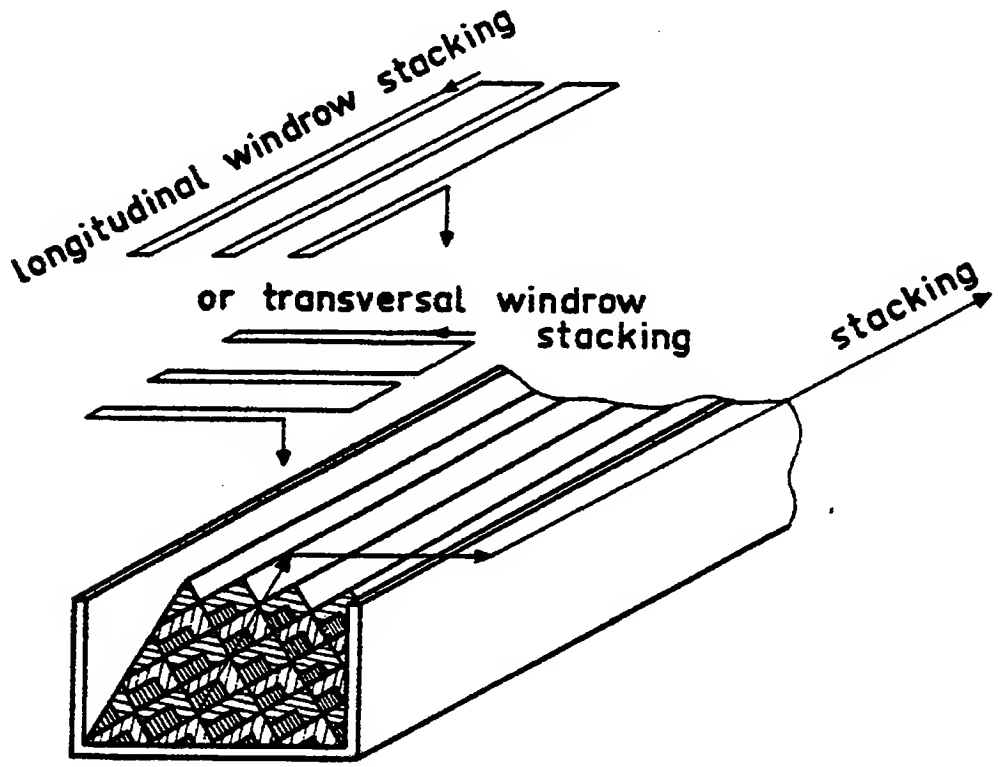
3.1.4 The Stacking and Reclaiming in the Homogenizing Pit

The stacking method in the homogenizing pit could be both, either Windrow stacking in longitudinal or transversal layers. Reclaiming is performed by a bucket chain excavator lifting and discharging the material on to a bridge mounted transversal extracting belt (see fig. 9 and fig. 25).

Figure 9 Homogenizing Pit - Windrow Stacking



Note: The homogenizing pit is applied for small preblending capacities or for sticky materials requiring a good homogenizing efficiency. Due to the high investment cost, the homogenizing pit is not used for 'normal cases' where the longitudinal Chevron bed can be applied.

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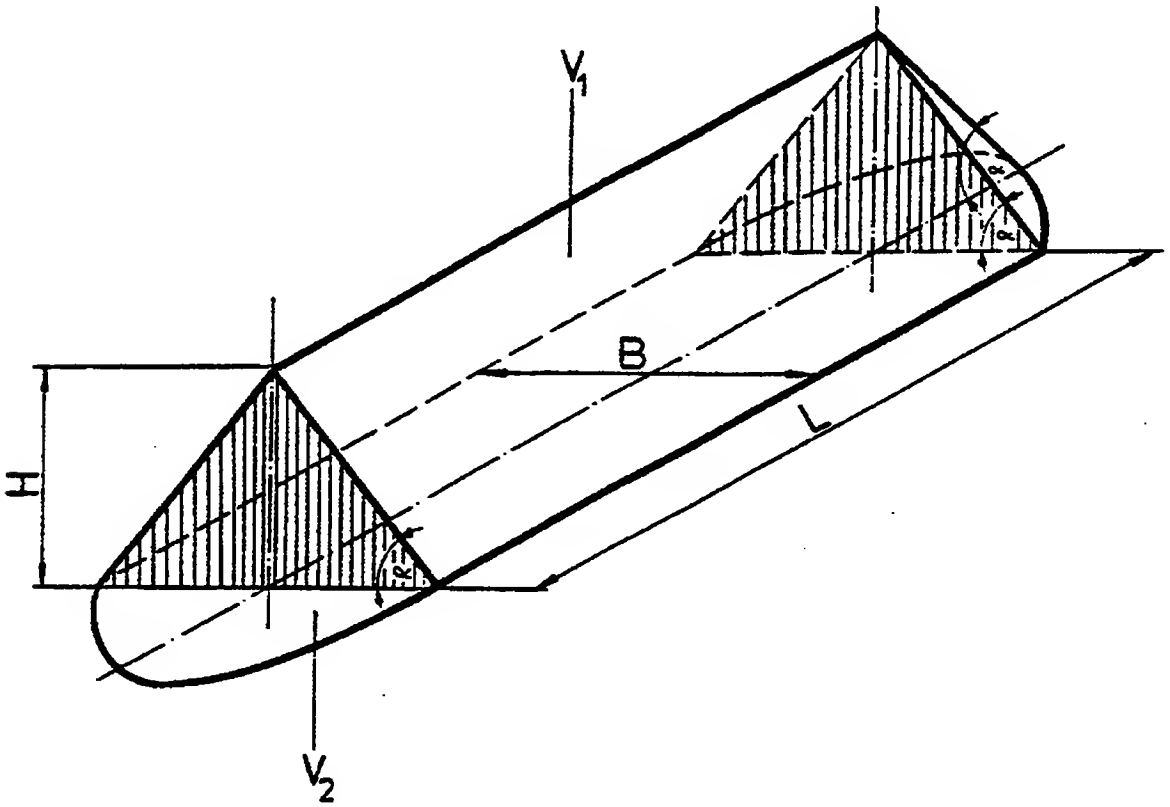
3.2 Pile Geometry

Figure 10 Longitudinal Preblending Bed Geometry



Volume Calculation Terms

Prismic Part:
$$V_1 = \frac{B \cdot H \cdot L}{2} = \frac{B^2}{4} \tan\alpha \cdot L$$

Cone end part:
$$V_2 = \frac{1}{2}\left(\frac{B^2 \cdot H}{12}\pi\right) = \frac{1}{2}\left(\frac{B^3 \tan\alpha}{24}\pi\right)$$

Pile Height:
$$H = \frac{B}{2}\tan\alpha$$

Total volume:
$$V_{tot} = V_1 + 2V_2 = \left(\frac{B^2}{4}\tan\alpha \cdot L\right) + \left(\frac{B^3 \tan\alpha}{24}\pi\right)$$

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3.3 Stacking Machines

The stacking machines dealt with here after allow the stacking according to the Chevron, Windrow and Conical Shell pattern. For stockpiling the material there are, in principle, two different types of machine. Throw-off carriages (tripper) fed by a belt conveyor installed under the ridge of the roof of the storage building can be used only for stockpiles under cover. Laterally traveling boom stackers can be used for covered as well as for open-air stockpiles. Fig. 11 and 12 schematically illustrate a throw-off carriage with a fixed depositing chute for building up bed-blending stockpiles on the Chevron principle. If the throw-off carriage is equipped with a cross-traversible or a slewing throw-off belt, it can also be used for Windrow stacking. For this purpose the stackers are designed with slewing gear or with a telescopic boom.

The solution with a throw-off carriage operating along the ridge of the roof has the disadvantages that the material undergoes segregation as a result of the great height from which it falls, that the material in the pile becomes compacted, and that dust is thrown up. These disadvantages are obviated by stackers equipped with booms that can be lowered. Segregation can be reduced by variable reversing points of the stacker set according to the slope of the pile end (see chapter 5).

Because of the counterweight needed to balance the boom stacker, a machine of this type is rather wide and therefore requires a fairly large amount of clear working space within the storage building (see fig. 13).

The maximum number of layers to be deposited will depend on the stacking performance and the maximum travel speed of the machine. The highest travel speeds are around 25 - 30 m/min., enabling - with a pile width of 30 m and a stacking capacity of 1000 t/h - 400 - 500 layers to be deposited.

In present-day practice the combination of Chevron stacking with a front-acting reclaimer has found widespread acceptance as a favorable system, since this method of stacking can be achieved with the least elaborate machinery.

Figure 11 Belt Conveyor and Belt Tripper

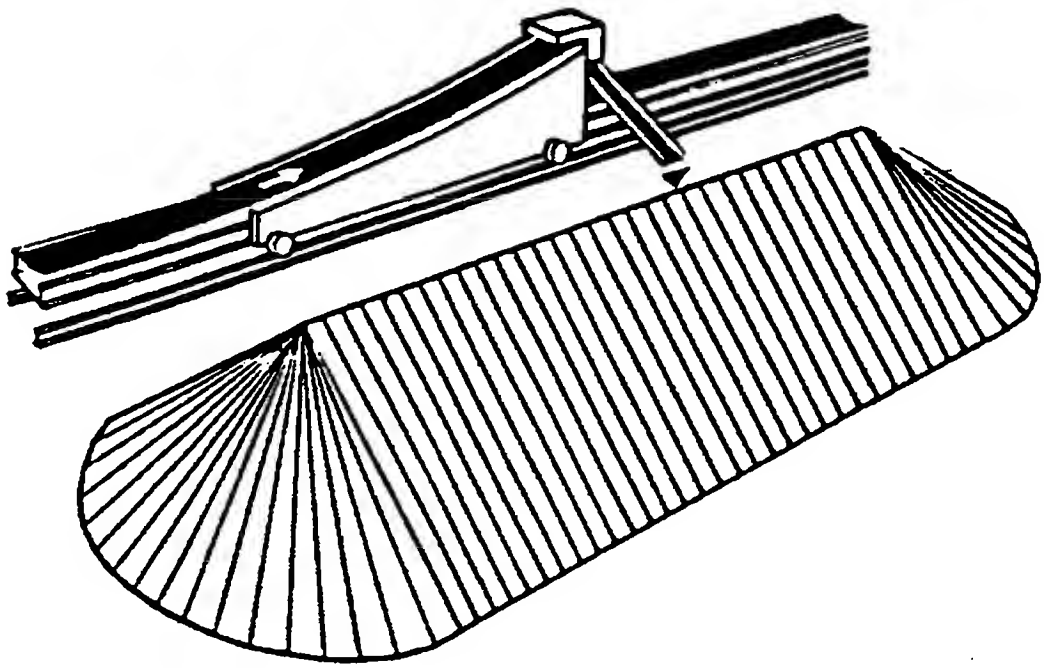

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Figure 12 Reversible Belt Conveyor

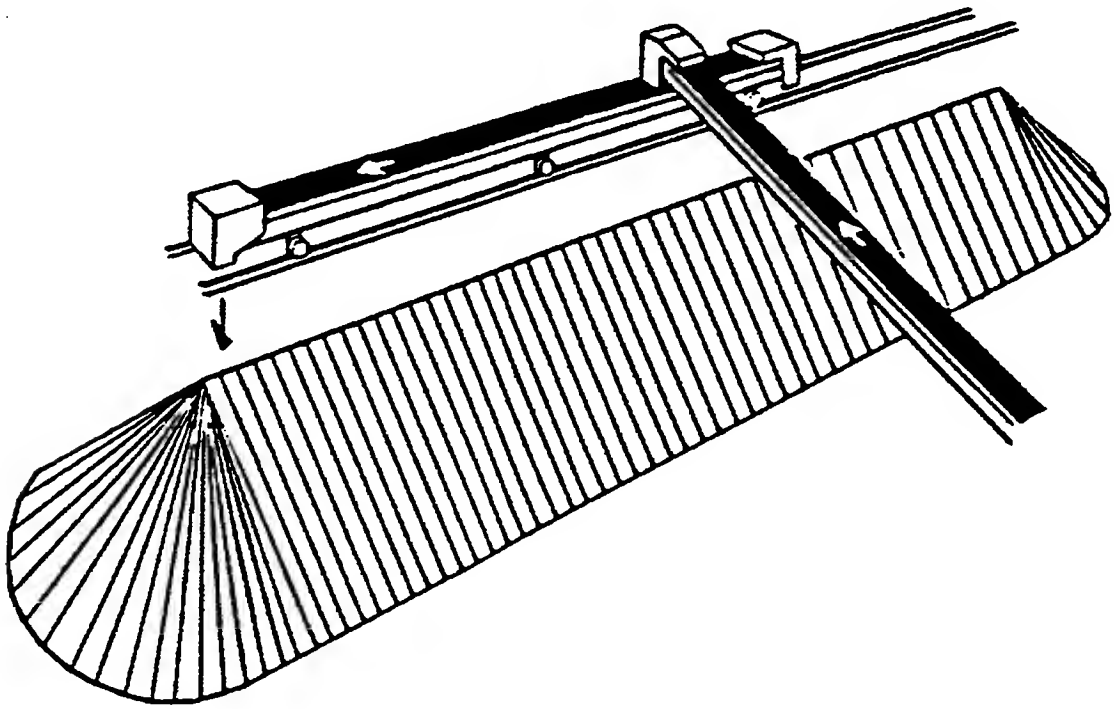
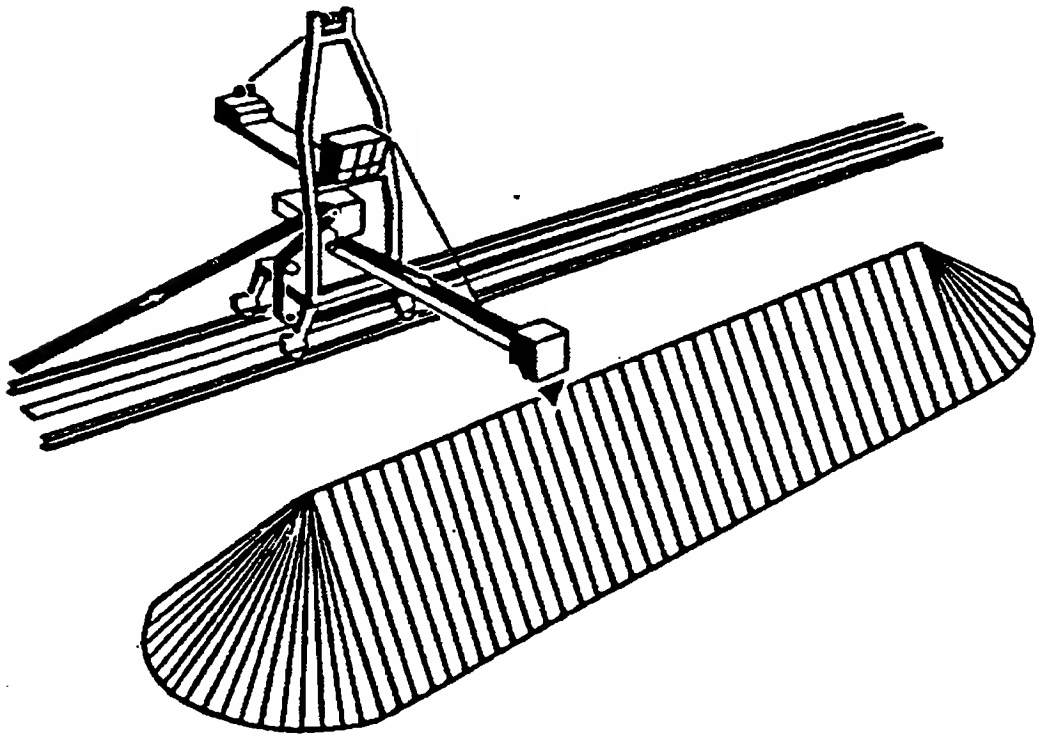


Figure 13 Side Stacker



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3.4 Reclaiming Machines

3.4.1 Front Acting Reclaimers

This type of reclaimer equipped with an extracting scraper chain (see fig. 14) is the mostly applied reclaiming machine.

The highest attainable performance for these bride-type reclaiming machines is around 500 m^3/h. Bridge spans in the region of 50 m are already in use.

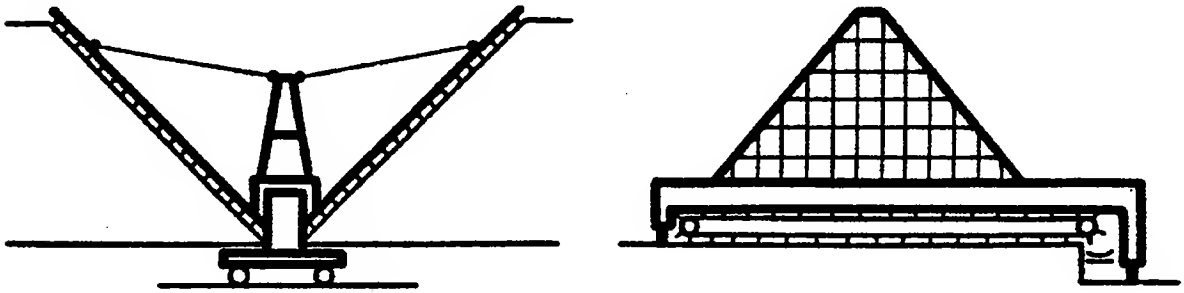
Advantages:

- The homogenizing effect is good because this slices of the whole cross-section of the pile are reclaimed.
- The output rate is constant and easy to control.
- Only a small amount of clear working space is needed for the machine inside the storage shed.
- The working direction of the machine can be changed quite simply.

Disadvantages:

- There is an upper limit to the output attainable.
- For feeding the exit belt conveyor the machine requires, on the discharge side, a feeding table or delivery chute, to enable the discharge to the exit belt.
- As the machine moves continuously into the pile and the scraper chain is filled progressively towards the discharge point the bridge has a tendency to move at the back and more quickly that at the front end. Provision must be taken that this oblique movement can be corrected.

Figure 14 Bridge-Type Reclaimer with Harrow



3.4.1.1 Dislodging Device

All front-acting reclaimers are equipped with a material handling system which can pick up the material only from the toe of the stockpile. A dislodging device, for bringing the material down from the slope, sweeps across the end face of the pile. The material contained in a thin slice undergoes blending while it rolls and slides down. Machines equipped with harrows or rakes, rope-operated scrapers and scraper chains for dislodging the material are available (fig. 15).

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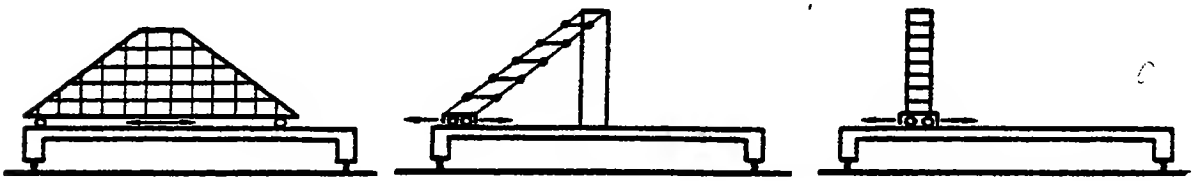
Harrows are normally triangular in shape with renewable teeth and capable of being tilted to the natural angle of repose of the material. Dislodging the material is achieved by the oscillating to-and-fro movement of the device.

Scraper chains are more particularly suited for dislodging difficult and heavy materials from stockpile slopes. In performing their to-and-fro motion they sweep the whole end face and force the material by positive action to the tow of the slope.

The rope-operated scraper comprises two ropes which pass around pulleys at the tip of the pile attached to a slide which performs a shuttle motion along the bridge. In this way the ropes pass to and fro across the whole face of the pile in the manner of windscreen-wipers. For dealing with heavy material the ropes are interconnected by pivoted cross-beams studded with teeth. In this way the loosening and dislodging effect can be substantially increased and the wear on the ropes reduced.

Figure 15 Slope Scraper



3.4.1.2 Scraper Chain

The scraper chain extracts the material to the side of the pile where it is discharged onto the exit belt. Different types of chain arrangements can be distinguished, horizontal, inclined or inclined only at the discharge end on the exit belt (see fig. 16 - 18). A discharge table is for the horizontal as well as for the inclined chain an indispensable prerequisite. The horizontal chain inclined at the discharge end therefore needs only a delivery chute which is less expensive compared with the concrete transfer (see fig. 18).

Figure 16 Horizontal Chain

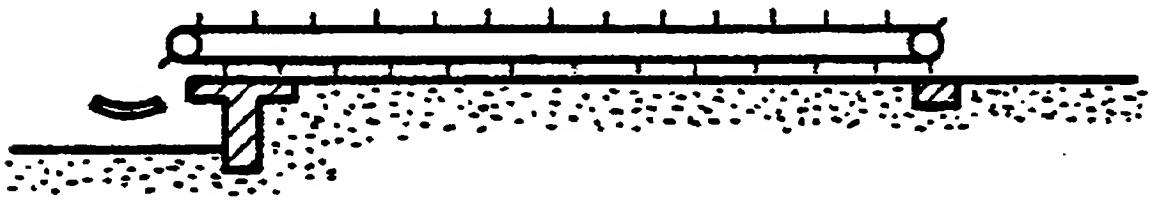


Figure 17 Inclined Chain

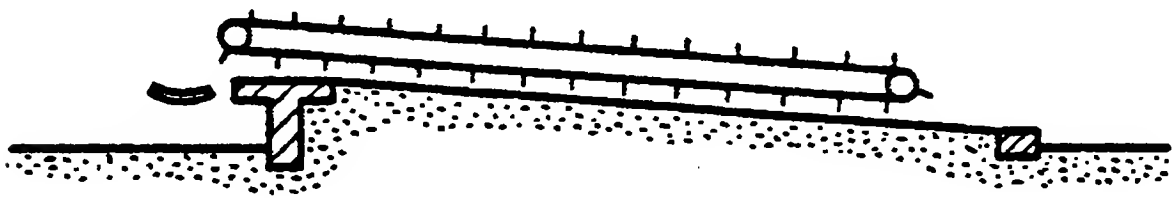


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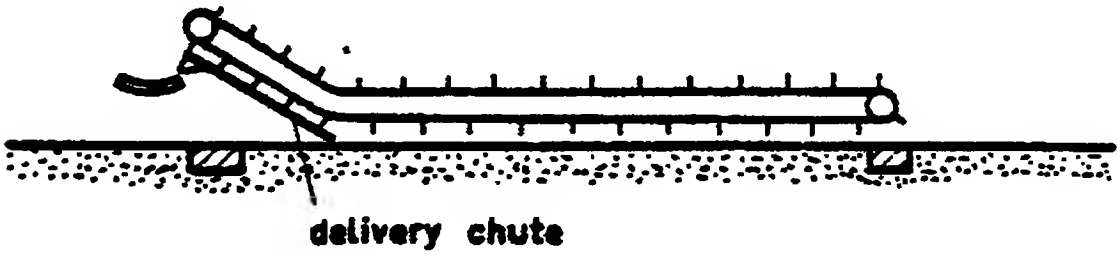
Figure 18 Horizontal Chain Inclined only at the Discharge End



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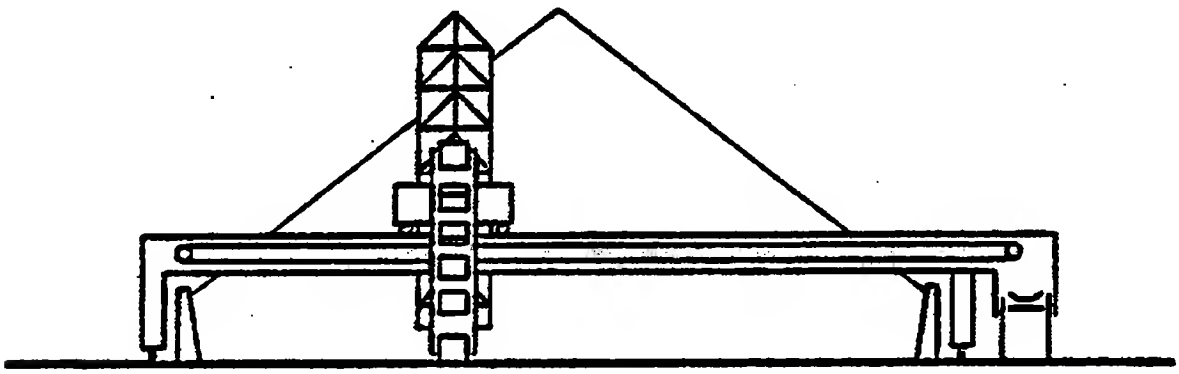
3.4.1.3 Bridge-mounted bucket-wheel reclaimers

A bridge-mounted bucket-wheel reclaimer takes the material from the tow of the slope and discharges it to a transverse belt mounted on the same bridge. The dislodging device is attached to the support of the bucket-wheel and moves to and fro with it (see fig. 19).

Figure 19 Bucket-Wheel Reclaimer as Bridge-Type Machine



Advantages:

- Good homogenizing effect. The variations that occur because the machine does not sweep the whole cross-sectional area of the pile all the time in the to-and-fro motion of the bucket-wheel are of short duration and can without difficulty be evened out in the subsequent raw material processing stages.
- The output performance is virtually unlimited. For high reclaiming rates it is possible to mount two or more bucket-wheels on one and the same bridge.
- Sideways discharge of the material onto the exit belt conveyor is done by a belt installed in the bridge. This is an energy-saving arrangement compared with the bridge-type scraping reclaimer and makes it possible to construct a low retaining wall as the lateral boundary to the toe of the stockpile. Thus there is no risk of overfilling, and the amount of space required is kept down to the minimum.
- The bridge is moved stepwise while the bucket-wheel is in its extreme position. It is therefore no pronounced danger for oblique movement to the bridge.

Disadvantages:

- The rate of discharge of the reclaimed material does not remain constant as the bucket-wheel moves transversely. This drawback is compensated by switching to three different speeds for the traversing of the bucket-wheel. This involves more elaborate control, however.
- The clear working space required is greater than for the bridge-type scraping reclaimer.
- On reversal of the working direction the buckets have to be reversed too.

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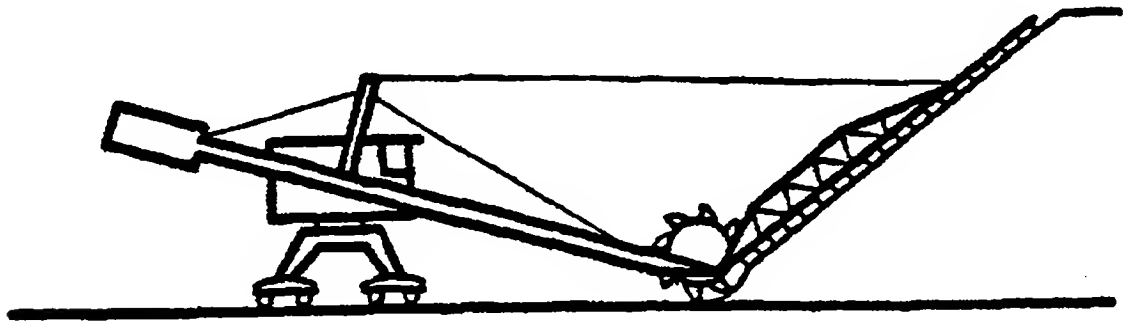
3.4.1.4 Bucket-Wheel Reclaimers with Slewing Boom

Machines of this type operate in the same way as the machines described in section a, except that the reclaiming face is not flat but curved (fig. 20).

Figure 20 Bucket-Wheel Reclaimer as Slewing Machine



Advantages:

- These are the same as those offered by the bridge-mounted bucket-wheel reclaimer.
- Since the rails on which the machine is mounted are within the stockpile itself, there is an additional saving of space.

Disadvantages:

- These are the same as those associated with the bridge-mounted bucket-wheel reclaimer.
- Turning the machine inside a building requires much space.

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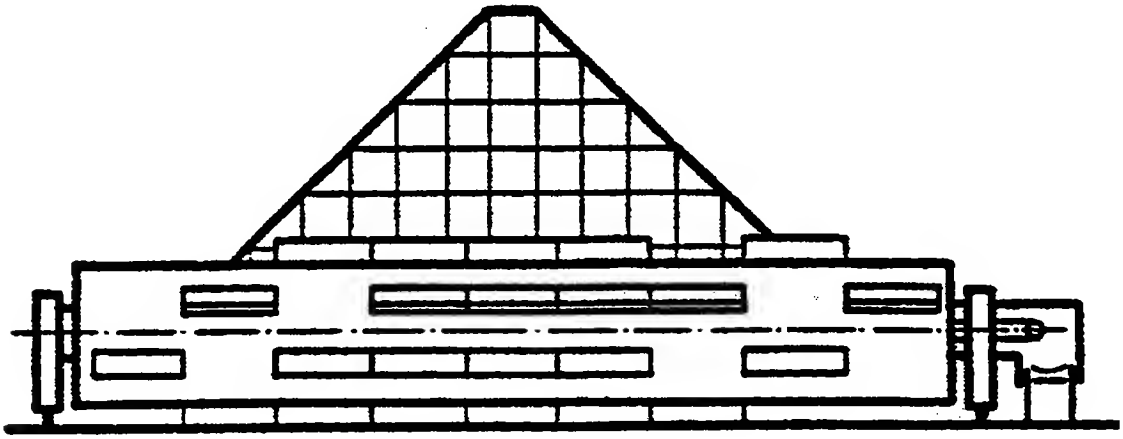
3.4.1.5 Tube Reclaimers

With this type of machine the material dislodged by the to-and-fro motion of a harrow is picked up by scoops mounted on a rotating tube or drum and is deposited onto a belt conveyor running inside the tube (fig. 21). The blending effect that they can attain is excellent, as their reclaiming action comprises the whole cross-section of the stockpile. However, they are sophisticated and expensive machines which can be economically employed only for very high reclaiming rates (upwards of 2000 t/h).

Figure 21 Tube Reclaimer



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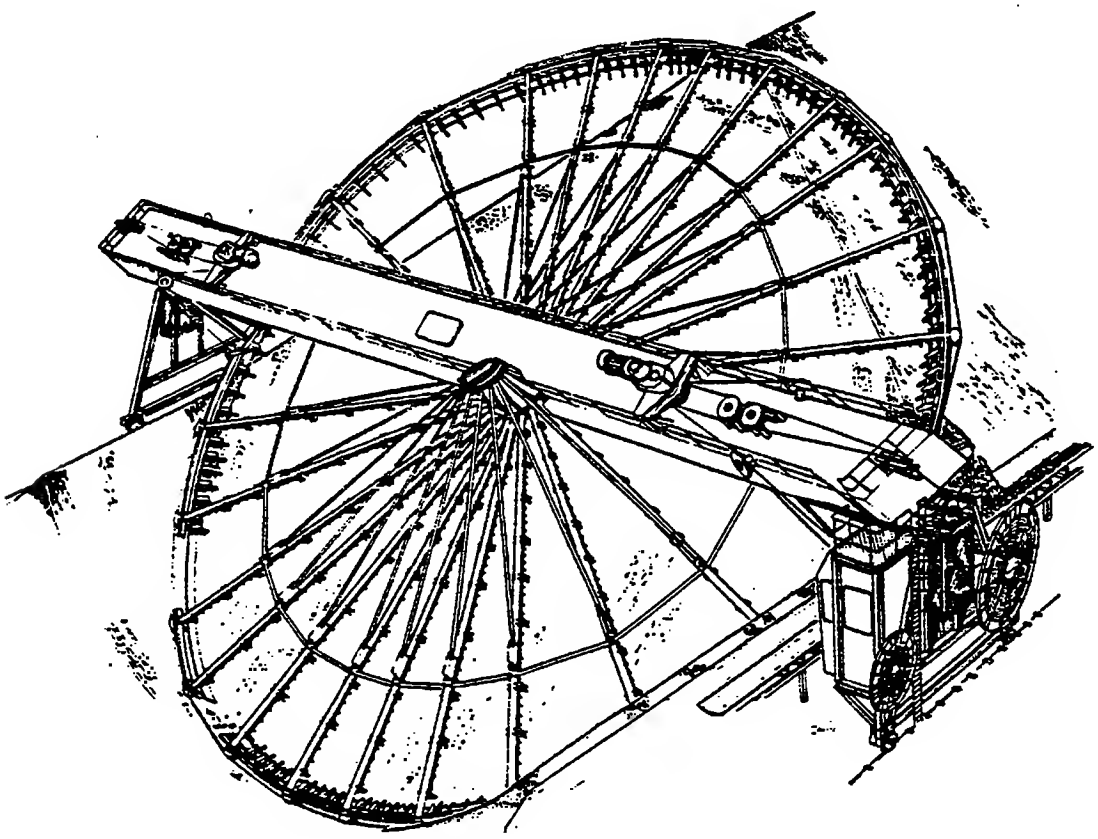
3.4.1.6 Disc Reclaimer

This type of reclaiming machine is an entirely new concept of a reclaimer. It only has in common with the other reclaimers that it acts also on the front of the pile (see fig. 22).

Working principle:

A rotably disc structure supported by a bridge spanning the width off the bed agitates, transports and discharges the material continuously over the rim of the pile onto the exit belt. The only parts of the machine that are in contact with the material to be reclaimed are the harrow teeth connected to the spokes, the bottom edge of the rim, the inside of the rim and the carriers along the rim. If the material has reached the rim of the pile it slides down the pile side over the reclaiming wall onto the collecting conveyor. The disc inclination can be adjusted on the slope of the pile face. The machine can reclaim in two opposite directions by simply tilting the disc from one position through the horizontal to the other position. If necessary the disc could discharge at either side by selecting the rotation direction.

Figure 22 Disc Reclaimer



3.4.2 Side Reclaimers

Side reclaimers acting from the side of the preblending bed are equipped with a boom mounted scraper chain that can be lowered. They work either on the front of the preblending pile or on the side.

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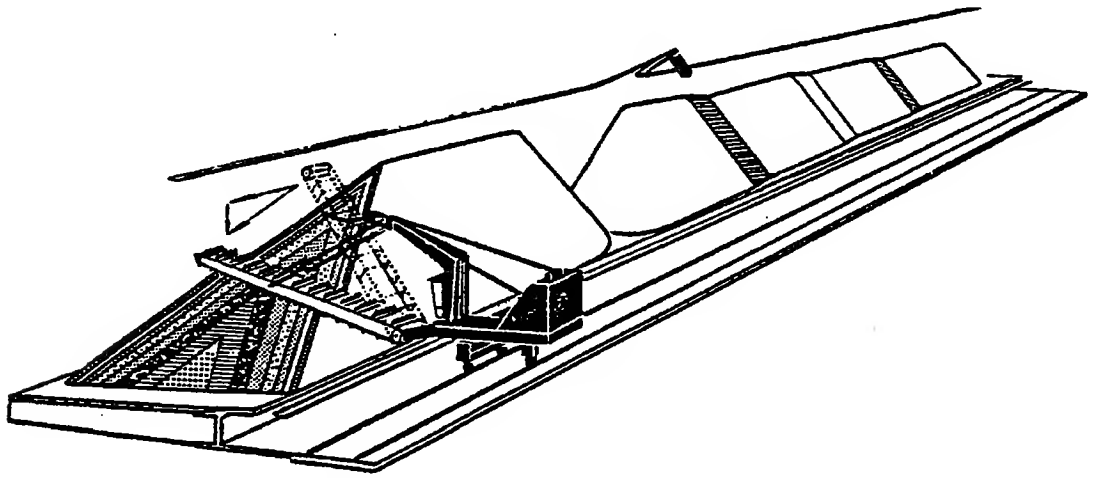
3.4.2.1 Side Reclaimers Working on the Front of the Pile

Reclaiming of the material from the pile is done by lowering and lifting the scraper chain on the pile face simultaneous forwards and backwards movement of the reclaimer carriage (see fig. 23).

This method of reclaiming has severe drawbacks and should not be installed anymore:

* Complicated sequence control of boom and carriage and thus high maintenance requirement
* Irregular material discharge flow requiring large belt capacities (fluctuations up to 50 %).

Figure 23 Front (Pilgrim Step) Reclaiming



3.4.2.2 Side Reclaimers Working on the Side of the Pile

The lowering boom reclaims the material from the side of the pile while the carriage travels the whole length of the pile (see fig. 24).

This working method is only applied in conjunction with Conical shell stacking. Their respective advantages and disadvantages can be summarized as follows:

Advantages:

- Several material components stored on the same preblending bed can be served by a common reclaiming machine (e.g. additives stored on the same preblending bed).
- Suited for fairly sticky materials
- Low investment costs

Disadvantages:

- A diminished homogenizing efficiency is obtained since not all layers are blended simultaneously (Conical shell stacking).
- Suited only for homogeneous raw materials (additives) with fluctuations $s \leq 2$ % $CaCO_3$.
- Preferred only if the available preblending bed concept demands a simple automatic material storage with no or less homogenizing requirements.

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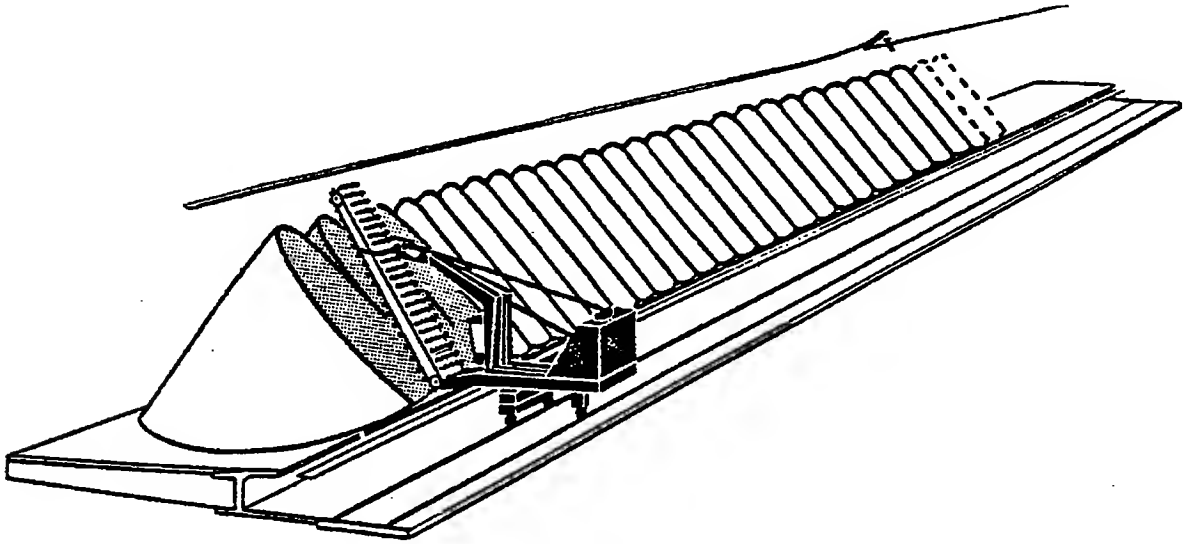
Figure 24 Lateral (Side) Reclaiming



3.4.3 Reclaiming Machines for Sticky Materials

In the cement industry it is often necessary to deal with materials that tend to stick. In such cases difficulties are liable to arise with some types of machine described in the foregoing, because in certain circumstances the material refuses to be easily dislodged and come tumbling down to the tow of the pile. The side-acting scraper can still be used for such materials. The scraper blades are normally provided with ripper teeth in that case, and devices for cleaning the blades when dealing with extremely sticky material have been developed. Alternatively, an easily modifiable version of the slewing-boom bucket-wheel reclaimer can be employed for the purpose. Here the raking device for dislodging the material from the slope is omitted, and the bucket wheel is raised to reclaim directly from the stockpile. Unfortunately in both cases the blending effect achieved by such machines is diminished.

A third possibility, without loss of blending effect, is provided by the homogenizing pit with bucket chain reclaimer.

Because of the high cost of construction the pit to receive the material, these blending installations are expensive. Under favorable conditions it is possible to construct the pit by blasting it out of solid rock and thus saving money. In comparison with stockpiles of triangular cross-section the space required by such tanks is substantially less. Under cramped conditions in a specific case it will therefore be the most economical one even in a case where the material to be dealt with is not particularly sticky. The application of this type of installation is also indicated for small storage capacities where the space for a Chevron bed becomes too large (see fig. 9 and fig. 25).

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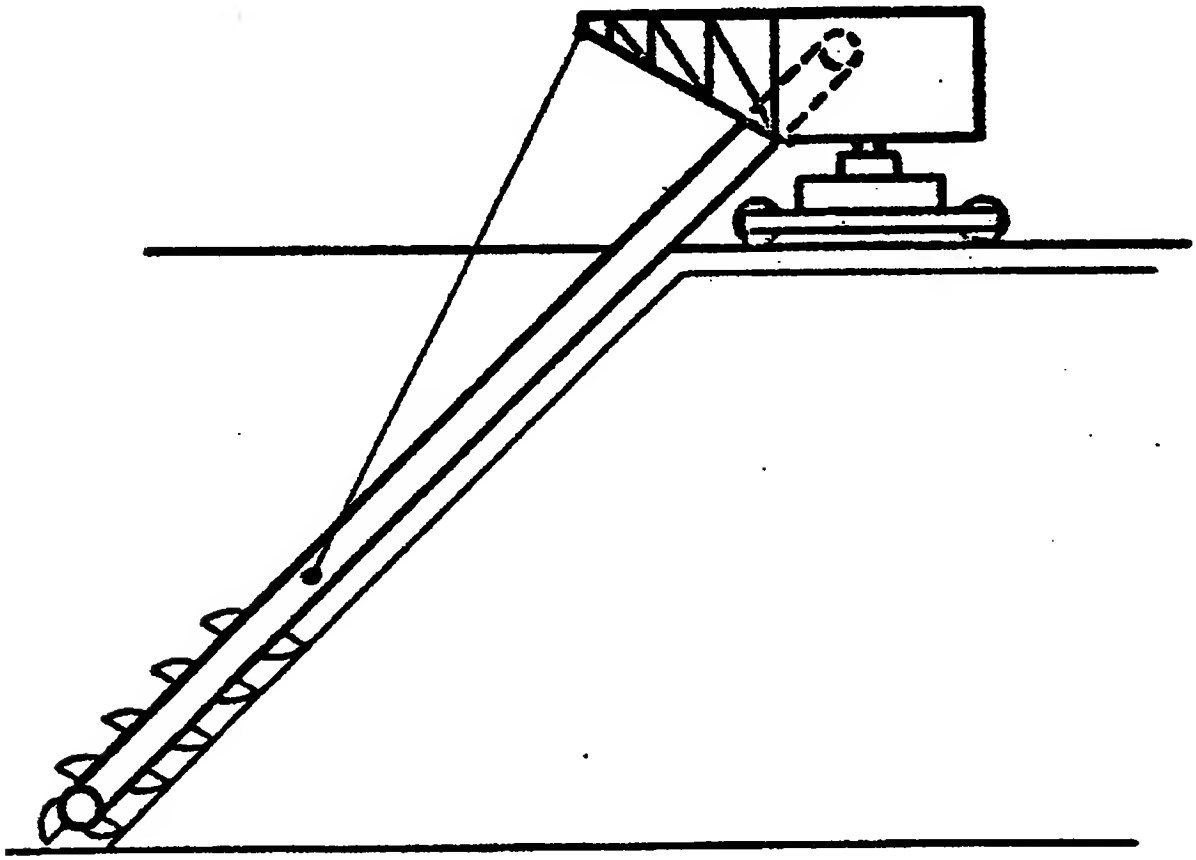
Figure 25 Homogenizing Pit with Bucket Chain



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4. CIRCULAR PREBLENDING BED

Principally two different types of circular preblending beds can be distinguished:

Batch-wise preblending by forming two or more pile according to the Chevron method covering two or more sections of the complete circumference. The piles are linked together (see fig. 27) or separate (see fig. 26).

Averaging preblending by forming one continuous pile according to the so called Chevcon stacking method. This method is explained in chapter 4.1.2.

The chemical fluctuations are differently damped by both methods. While batchwise stacking enables the achievement of a certain fixed mean value, averaging blending yields a continuous floating mean value (see fig. 28, 29, 30).

Figure 26 Separate Piles

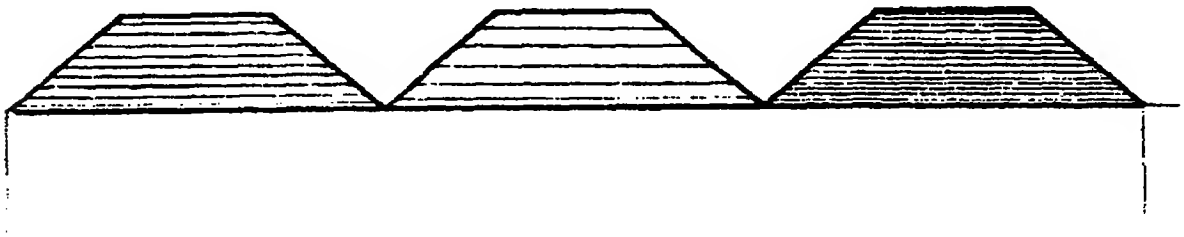


Figure 27 Adjacent Piles

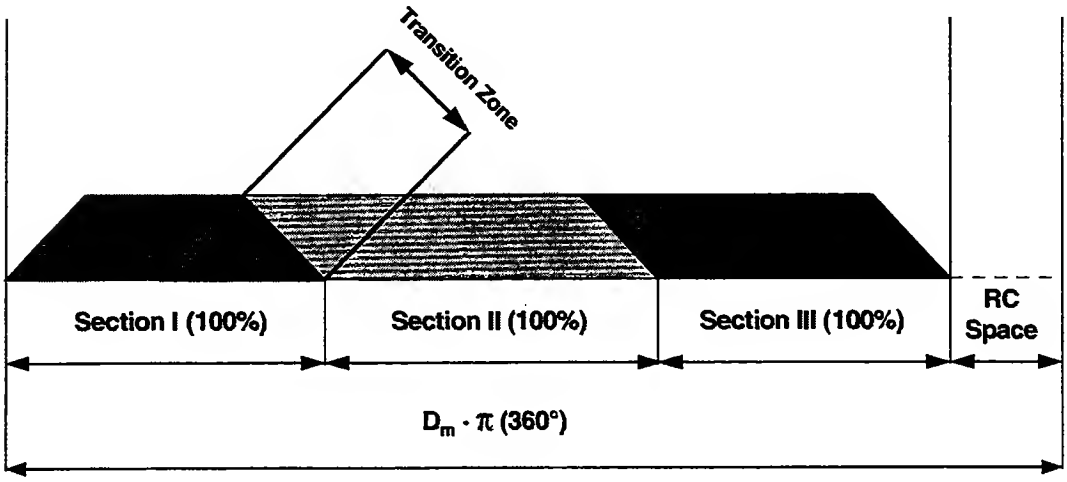


Figure 28 Batchwise Stacking with Separate Piles

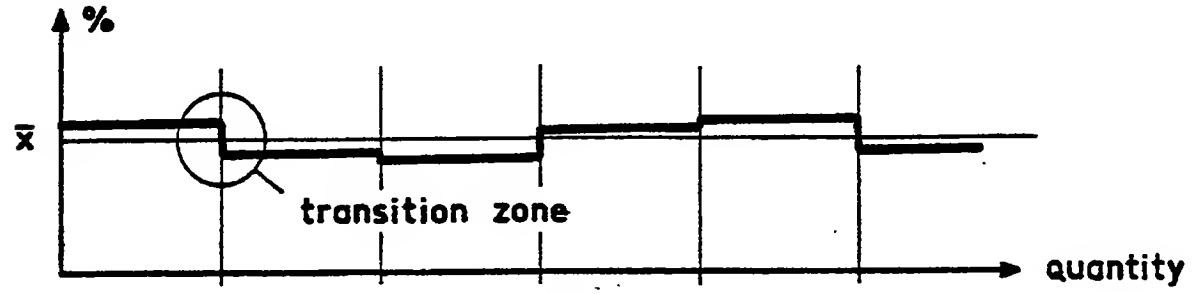


Figure 29 Batchwise Stacking with Adjacent Piles

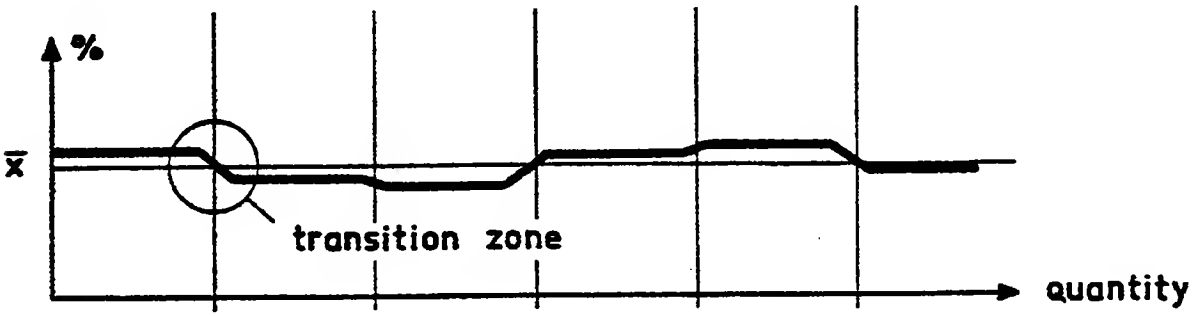
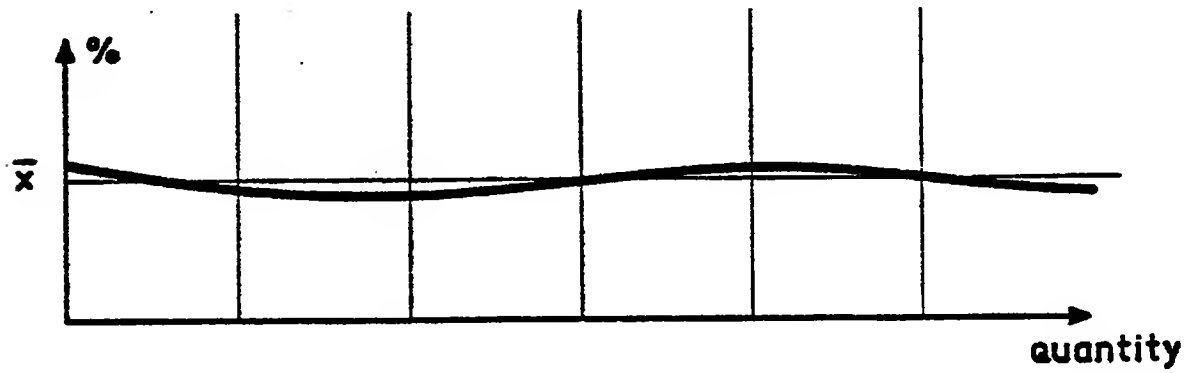


Figure 30 Averaging Stacking by Continuous Stacking



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4.1 Working Principle

4.1.1 Circular Preblending Bed with Batchwise Stacking

The batches of the pile are stacked according to the known Chevron method. The Conical shell method is also possible but not recommended due to the reduced blending efficiency.

The circular batch type stacking procedure can either be performed in separate (fig. 26) or adjacent piles (fig. 27).

Separate piles need too much space between single sections and are not recommended.

Adjacent piles gain a better space utilization but a diminished or heavy controllable homogenizing efficiency in the transition zone between single piles occurs.

In order to guarantee a sufficient operating flexibility a minimum number of three segments is recommended which distributes the total circumference of 360 ° in 60 ° minimum allowable space reservement for the reclaiming machine and 100 ° for each segment. More pile segments could improve the operating flexibility but adverse would reduce the homogenizing volume.

Before starting a new pile at least one segment and the space reservement for the reclaimer has to be free.

The ratio between stacking and reclaiming volume amounts in an optimal case with three segments to 2:1 while longitudinal piles always correspond to a ratio of 1:1.

If ever possible the continuous Chevcon stacking should be applied to overcome the diminished and nearly uncontrollable homogenizing efficiency in the transition zone of adjacent batch type pile segments.

4.1.2 Circular Preblending Bed with Continuous Stacking (Chevcon Method)

This relatively newly developed method of stacking is a superimposing of the Conical shell and Chevron method. As already mentioned above, the Chevcon stacking represents a continuous blending contrarily to the methods which yield batch blending. Continuous blending provides the following advantages:

- Continuous operation
- maximum space utilization
- variable storage volume according to the homogenizing requirements
- no segregation problems

The following working principle applies for the Chevcon stacking:

Stacking the material in the blending bed is effected over a preselected path length, corresponding to certain angle of slew, which in the course of each forward and reverse motion of the stacker is advanced a distance equivalent to the volume of material stacked. The number of layers deposited on the bed depends on the slewing speed of the stacker and on its material handling rate. Superimposed on the slewing is the raising / lowering motion, so that during each pass the stacker is raised from its lowest to its highest point, and vice versa. An electronic control system ensures that the stacker is kept lower over the stockpile, so that the height of fall of the material in building up the pile is always suitably small. The preselected slewing angle progressively shifts round the circumference, and by continually the stacker builds up an annular pile of material. The smaller the angle of slope of the pile is chosen, the longer becomes each individual layer deposited.

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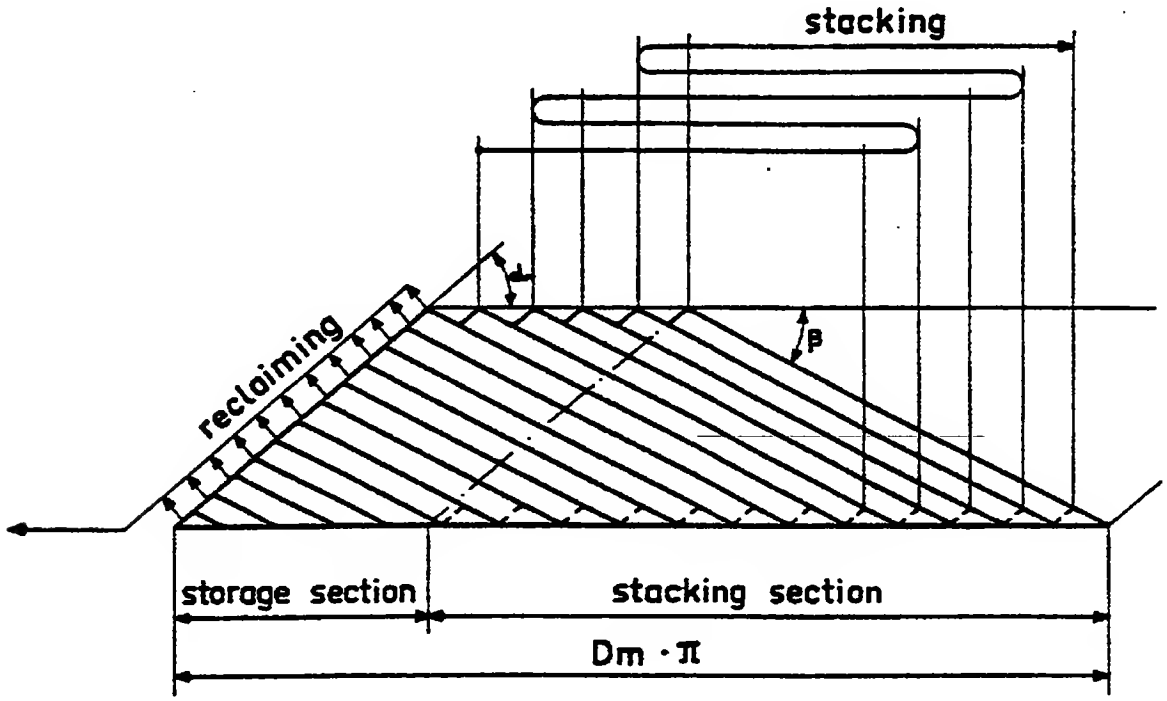
Figure 31 Chevcon Stacking



Note: The total homogenizing volume (see fig. 33) is a function of the length or the inclination of the layers. Chemical fluctuations within this fictive volume corresponding to approx. 5 days are yet homogenized. The homogenizing efficiency within the homogenizing volume depends on the number of layers, which is a function of the stacker velocity, the input mass flow and the pile cross section.

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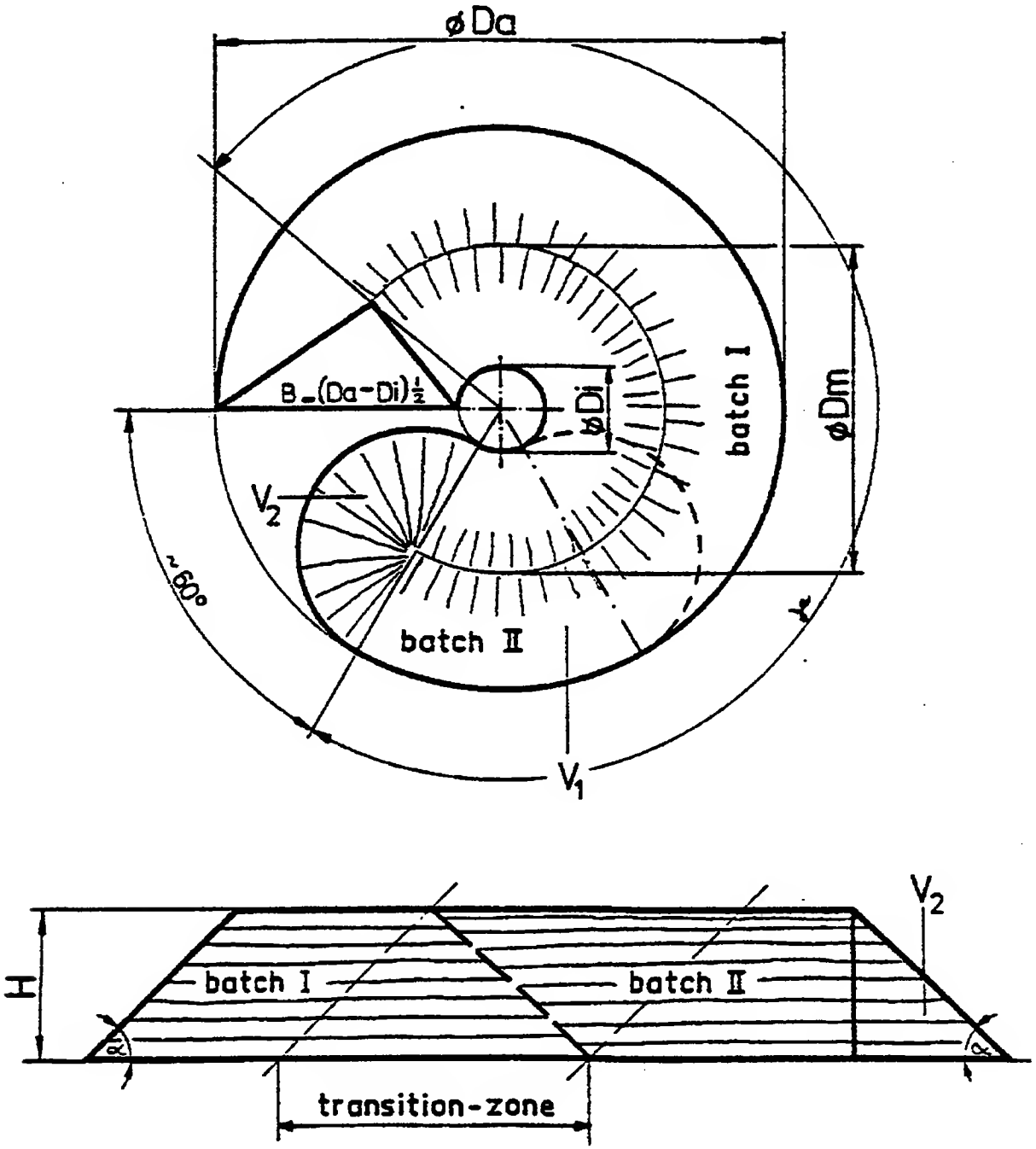
4.2 Pile Geometry

The circular preblending bed geometry differs for the two stacking types.

4.2.1 Pile Geometry for 'Batchwise Stacking'

In the following the geometry of the batchpiles linked together are considered only since separate piles spoil too much space.

Figure 32 Pile Geometry for Batchwise Stacking



Volume Calculation Terms

Pile width: $$B = \frac{D_a - D_i}{2}$$

Pile height: $$H = \frac{B}{2}\tan\alpha$$

Cross section: $$F = B \cdot \frac{B}{2}\tan\alpha\frac{1}{2} = \frac{(D_a - D_i)^2}{16}\cdot\tan\alpha$$

Mean diameter: $$D_m = \frac{D_a + D_i}{2}$$

Prismic part of pile: $$V_1 = \frac{D_m \cdot \Pi \cdot F\gamma}{360^o} = \frac{(D_a + D_i)\cdot\Pi\cdot\gamma}{2\cdot 360^o}\cdot\frac{(D_a - D_i)^2}{16}\cdot\tan\alpha$$

Cone end part: $$V_2 = \frac{1}{2}\cdot\left[\left(\frac{D_a - D_i}{2}\right)^3\cdot\frac{\Pi}{24}\cdot\tan\alpha\right]$$

Total volume: $$V_{tot} = V_1 + 2\cdot V_2 = F\cdot\gamma + 2\cdot V_2$$

4.2.2 Pile Geometry for 'Averaging Blending'

Using the averaging blending method there is only one continuously growing pile. But only one part of this pile is fully homogenized and ready to reclaim.

Considering the maximum available volume (V) of the pile (Dm · π · F), three subvolumes can be differentiated (see fig. 33):

- the storage volume ready to reclaim (V_1)
- the total homogenizing volume (V_2), consisting of the effective stacked volume (V_{St}) and the theoretical homogenizing volume (V_{theor}).
- The empty space reserved for the reclaiming machine (V_3).

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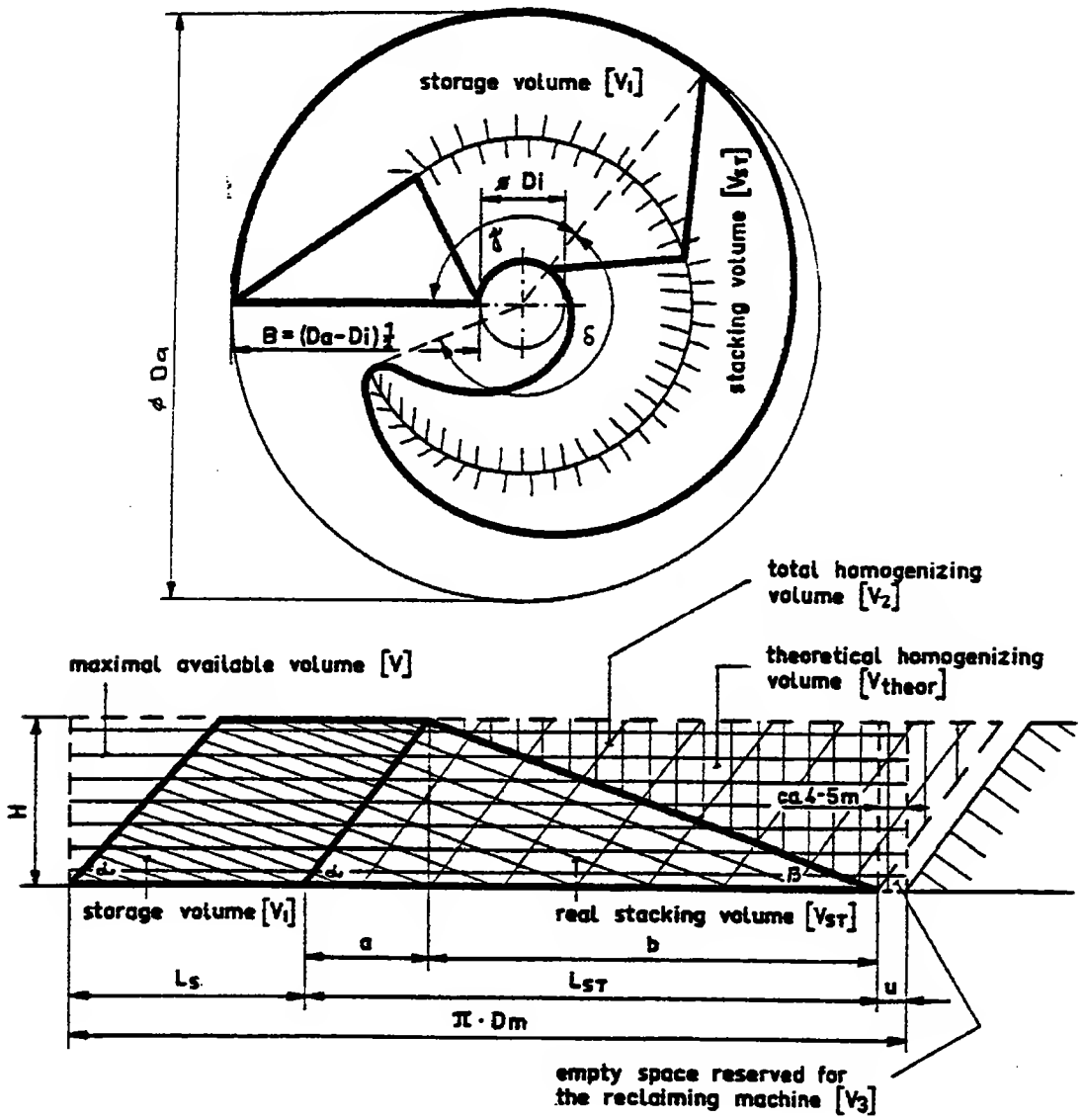
Figure 33 Pile Geometry for Chevcon Stacking



Chevcon Stockpiling

Volume calculations

Pile width: $$B = \frac{D_a - D_i}{2}$$

Mean diameter: $$D_m = \frac{D_a + D_i}{2}$$

Pile height: $$H = \frac{B}{2}\tan\alpha$$

Cross section: $$F = \frac{B \cdot H}{2} = \frac{(D_a - D_i)^2}{16} \cdot \tan\alpha$$

Maximum available volume (V)

This volume represents the maximum possible volume of the circular preblending bed.

$$V = F\ Dm \cdot \pi$$

Storage volume (V_1)

This volume represents the homogenized volume ready to reclaim. It is normally chosen such as to last approx. 1 week or at least over a longitudinal weekend (3 days).

$$V_1 = F * D_m * \pi * \frac{\gamma}{360} = F * L_3$$

Note: (L_s) is equal to the distance on the mean pile circumference and (γ) represents the belonging angle.

$$L_s = \frac{V_1}{F}$$

$$\gamma = \frac{V_1}{F * D_m * \pi} * 360°$$

The homogenizing volume (V_2)

The total homogenizing volume composes of a theoretical part and a really stacked part. Both together influence the homogenizing efficiency of the pile since raw material fluctuations with this fictive volume V_2 are homogenized.

$$V_2 = V_{ST} + V_{theor.}$$

$$L_{ST} = \frac{V_2}{F}$$

$$S = \frac{V_2 * 360°}{F * D_m * \pi}$$

Theoretical homogenizing volume (V_{theor})

The theoretical homogenizing volume represents the volume which has to be added to the real stacking volume to form the total homogenizing volume (V_2).

$V_{theor} = V_2 - V_{ST}$

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Real stacking volume (V_{ST})

The real stacking volume represents the tail of the pile being in progress:

$$V_{ST} = F\left(\frac{B}{3}+\frac{b}{3}\right), \text{ or with } F = N\frac{q}{V} \Rightarrow V_{ST} = N\frac{q}{V}\left(\frac{B}{3}+\frac{b}{3}\right)$$

$$V_{ST} = N\frac{q}{V}\left(\frac{L_{ST} - {}^{H}/_{\tan\alpha}}{3}+\frac{B}{3}\right)$$

Note: (L_{ST}) is equal to the distance on the mean pile circumference and (δ) represents the belonging angle.

Other parameters

Number of layers in the pile cross section

$$N = F_{pcs}\frac{V}{q}$$

Number of layers in the pile face

$$N = \frac{N * \sin(\alpha+\beta)}{\sin\alpha * \sin(90°-\beta)}$$

Preblending bed hight

$$H = \sqrt{N * \tan\alpha * \frac{q}{V}}$$

Tail angle

$$\beta = \arctan\frac{{}^{B}/_{2} * \tan\alpha}{L_{ST} - {}^{B}/_{2}}$$

Angle of the slewing boom to form a complete layer: (δ')

$$\delta' = \delta - \frac{\alpha}{D_m\pi}360°$$

$$\delta' = \delta - \frac{H}{\tan\alpha}360°$$

$$\delta' = \frac{L_{ST} - \frac{H}{\tan\alpha}}{D_m\pi}360°$$

Progress angle of the slewing boom after each completed layer: ($\Delta\delta$')

$$\Delta\delta' = \frac{L_{ST}/N}{D_m\pi}360°$$

Empty space reserved for the reclaiming machine (V_3)

The empty space reserved for the reclaiming machine represents the empty volume between the tail end and the beginning of the pile.

Note: U is equal to the distance on the mean pile circumference representing the empty space reserved.

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4.3 Stacking Machines

In general the same machineries as introduced on the longitudinal preblending bed do also apply for the circular piles. The tripper carriage or side stacker of the longitudinal pile is replaced by a slewing stacker boom turning around the preblending bed center column.

4.4 Reclaiming Machines

Front acting reclaimers moving the material to the center discharge by means of a bridge mounted scraper chain are normally used. The center discharge consists of a funnel from where a tunnel underneath the pile withdraws the material and conveys it to the surface.

5. MATERIAL SEGREGATION IN THE PREBLENDING BED

The mathematical models to calculate the homogenizing efficiency, the capacity and the number of layers of the preblending bed, are based on the assumption that the material layers are stacked in an ideal manner one upon the other.

The longitudinal pile as well as the circular pile (separate pile) stacked according to the Chevron method have at both extremes of the stockpile two semi-conical ends with unideal stratification and material segregation. When the reclaimer takes its first slices out of the end cone it picks up only coarse material.

There are various ways and means of improving these adverse end-cone conditions:

- By adopting a high length / width ratio for the pile the volume of the end cones is reduced. A minimum value of L/B = 3 (see fig. 34) should be aimed at for this ratio, which corresponds to a volumetric proportion of about 15 % for the end cones. For L/B = 2 this proportion is about 20 %.
- One end cone at the 'far end' of the pile can always be left standing or be only partly reclaimed. Of course, this does not alter the situation at the 'near end' cone; besides, there is a loss of efficiency stockpile volume.
- The material in the end cones can be recirculated to the second stockpile. This procedure is time- and energy-consuming, however, and should therefore be applied only in special cases.
- The reversal points of the stacking machine can be staggered in relation to the height attained by the stockpile during the course of building it up. In this way the segregation at the 'near end' cone can be reduced.
- These end cone problems are obviated if a circular stockpile with continuous stacking is adopted for the blending bed.

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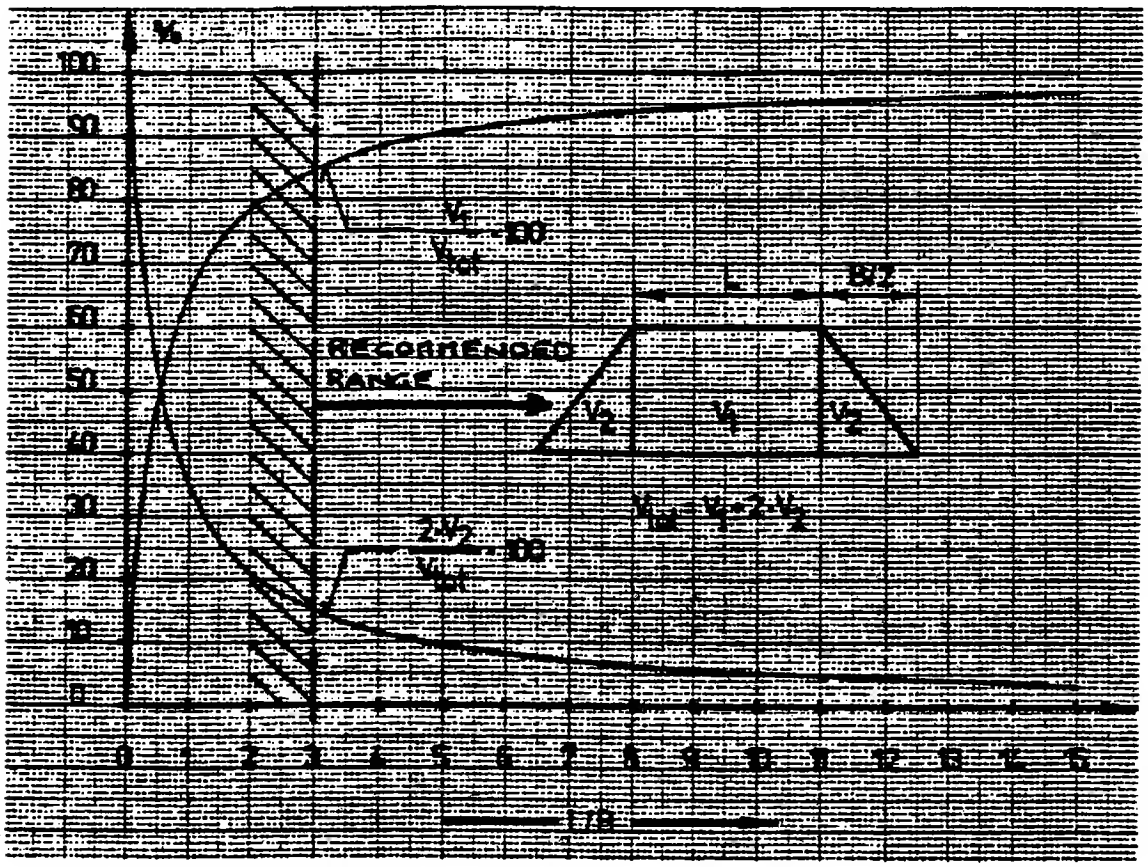
Figure 34 Percentage of the End Volume in Function of the L/B Ratio



5.1 Determination of the Reversal Points of the Stacking Machine

If the stacker is designed with only two fixed reversal points the non ideal layers according to fig. 35 develops. Fig. 36 illustrates the layering by variable stacker reversal points, where segregation is effectively reduced.

The reversal points are calculated as follows:

At the near end:

$$R_n = \left[n\sqrt{n} - \left((n-1)\sqrt{n-1}\right)\right]\left(1-\frac{\pi}{6}\right)\frac{H_1}{\tan\alpha}$$

$$H_1 = \sqrt{N_1 \tan\alpha \frac{q}{V}}$$

where:

R_n = reversal position (n) at the near end of the pile

n = number indicating the package of layers on which the reversal positions have to be adusted

N_1 = number of layers per layer package on which the reversal position is adjusted

H_1 = height of the first layer payckage on which the reversal position is adjusted

α = angle of repose

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At the far end:

$$R_n = \left[n\sqrt{n} - \left((n-1)\sqrt{n-1}\right)\right]\left(1-\frac{\pi}{6}\right)\frac{H_1}{\tan\alpha}$$

$$H_1 = \sqrt{N_1 \tan\alpha \frac{q}{V}}$$

R_n = reversal position (n) at the far end of the pile
all the other parameters remain as indicated above

Figure 35 Fixed Stacker Reversing Points

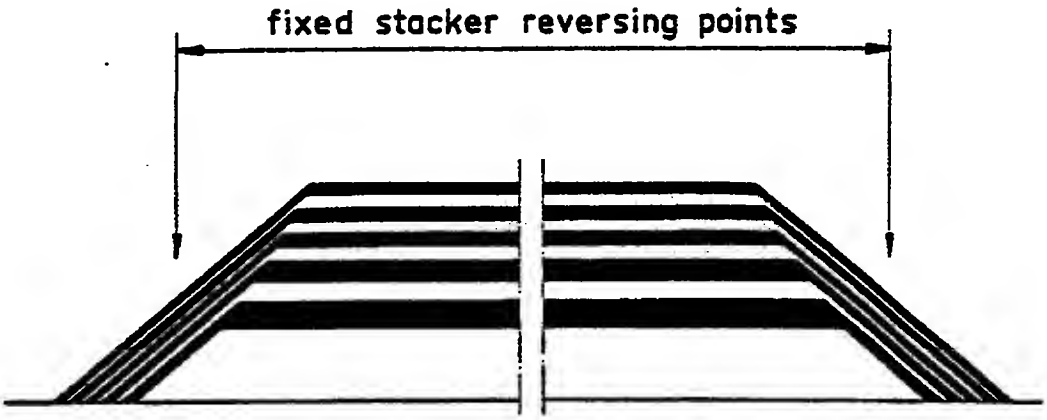
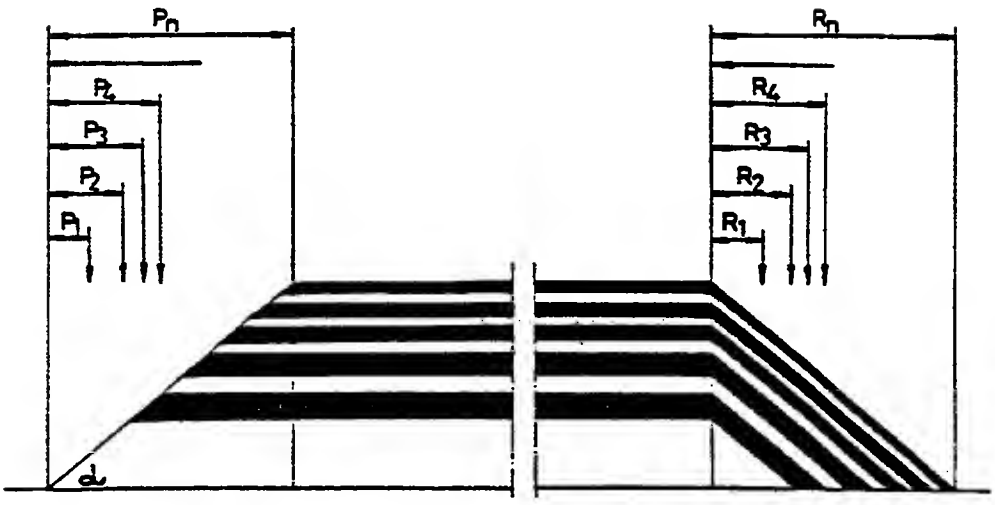


Figure 36 Variable Stacker Reversing Points



Comment

Theoretically, the number of variable stacker reversing points should be equal to the number of layers. This is however practically not feasible. Generally it can be said as higher the number of reversing points is chosen, the better will the blending results be. An appropriate number of stacker reversing points which are practically realizable are assumed with n = 10 - 15 for a pile height of approx. 10 m.

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The above determined variable stacker reversing points are only theoretical values which are based on geometrical relations at the pile shape and on the assumption of ideally stacked layers formed with a constant input quantity resp. stacker speed.

Since the input quantity varies in practice not the number of layers per stacker reversing point is important but the quantity of height related to individual stacker reversing points is the decisive measure.

6. PREBLENDING BED LAYOUTS

Since a stockpile destined to serve as a blending bed has to be built up first before the material can be reclaimed from it. It holds true for a longitudinal preblending bed where the material is stacked batchwise, but not for circular preblending beds with continuous stacking. On batchwise longitudinal stacking it is always necessary to operate with two piles to ensure uninterrupted delivery of material from the blending bed. One pile is being built up while reclaiming takes place from the other. The two piles may either be arranged parallel side by side or in line with each other.

The circular stockpile offers two other possibilities. In the one case the material is deposited by means of a revolving stacker attached to a central column. The material is placed in Chevron layer formation directly against the end cone of the pile already built up or by separate piles in sections. In the other case continuous stacking according to the Chevcon method is performed as dealt with in chapter 4. End cone problems are thus obviated and in addition a saving in space can be effected.

The homogenizing pit as dealt with in chapter 3.4.3. offers a further possibility suited for inhomogeneous, sticky material.

In selecting the layout for a blending bed installation the following important points should be given consideration:

* How much space is available?
* Does the layout fit in with the layout of the other parts of the plant and with the future extension scheme for the plants?
* Can good adaptation to the topography of the site be achieved?
* Is the choice of layout affected by subsoil conditions?

Of course, a detailed cost comparison will have to be made for the respective variants in order to arrive at the most economical solution. Their respective advantages and disadvantages can be summarized as follows:

6.1 Parallel Stockpiles

Advantages:

- Moderate length / width ratio fits in easily with the layout scheme for a cement work.
- Storage capacity can be increased.

Disadvantages:

- Reclaimer has to be changed over from one pile to the other.
- Slewing stacker or stacker with two booms.
- Large number of belt conveyors and transfer points.
- Long roof spans for shed.
- Extra space required for change-over of machine.
- End-cone problems.

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6.2 In-Line Stockpiles

Advantages:

- No change-over of machine.
- No slewing stacker required.
- Only two belt conveyors.
- Short roof spans for shed.
- Capacity can be increased.

Disadvantages:

- Long sheds.
- End-cone problems.
- High length / width ratio requires extremely long shed to accommodate high-capacity blending bed. Awkward to integrate into cement works layout.

6.3 Circular Stockpile

Advantages:

- Very short belt conveyors.
- Simple roof construction for shed, with central column to support it.
- No end-cone problems.
- Reclaiming output rate can easily be kept constant
- Area on plan about 40 % less than for straight stockpiles.
- No change-over of reclaiming machine.

Disadvantages:

- Sometimes difficult to fit into the cement works layout.
- Sticky material may choke the chutes in the central column.
- Ground-water may make it impossible to construct the discharge tunnel for removing reclaimed material.
- Capacity can be increased only by setting up a second pile.

6.4 Homogenizing Pit

Advantages:

- Very good space utilization.
- Windrow stacking.
- No end-cone problems.
- Roof of simple and light construction.

Disadvantages:

- Expensive machinery.
- Tank is expensive to construct
- Dust thrown up by material falling from great height.

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Figure 37 Longitudinal Preblending Bed Layouts

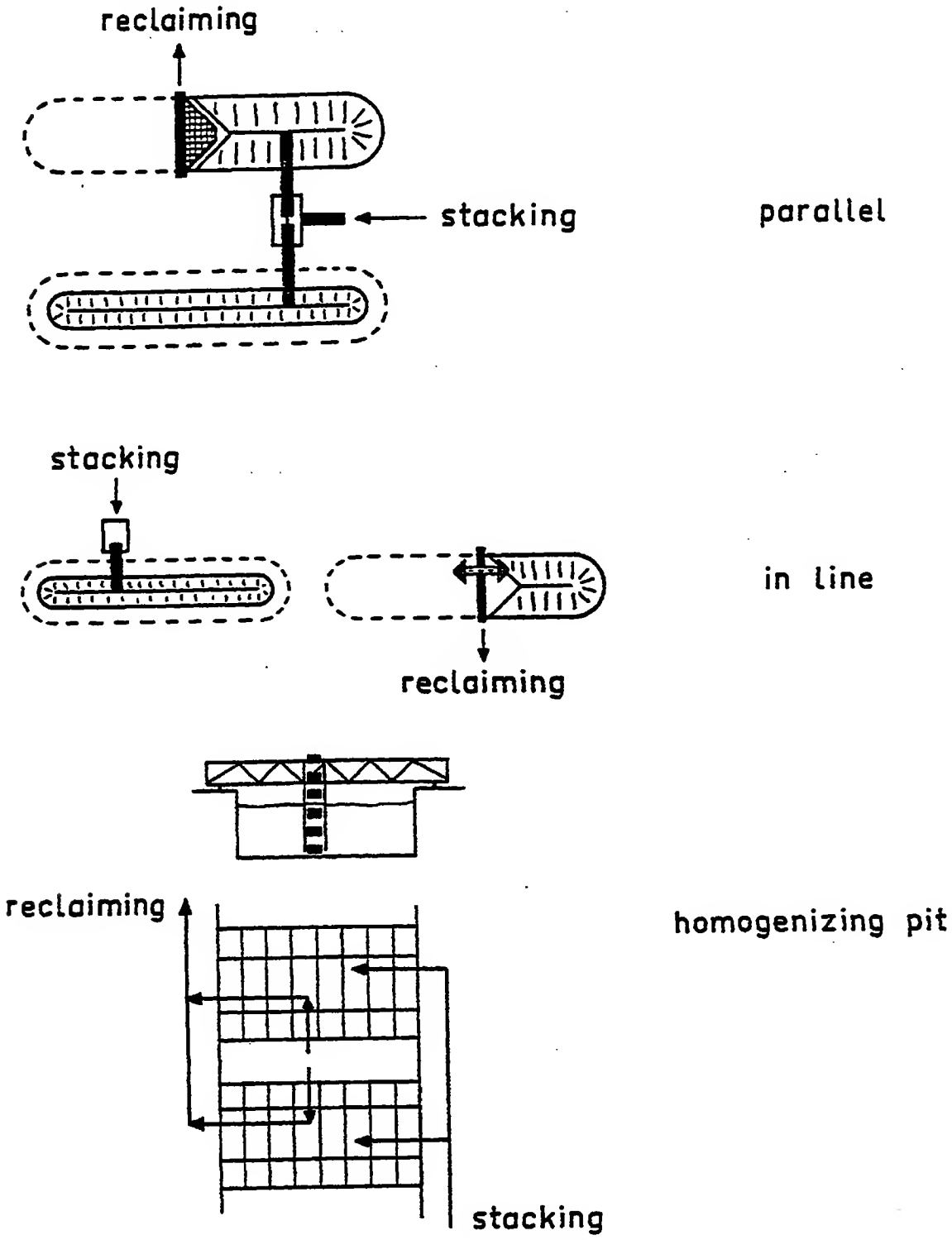
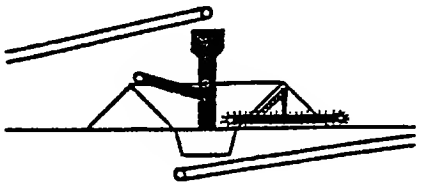
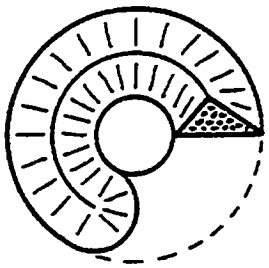


Figure 38 Circular Preblending Bed Layouts

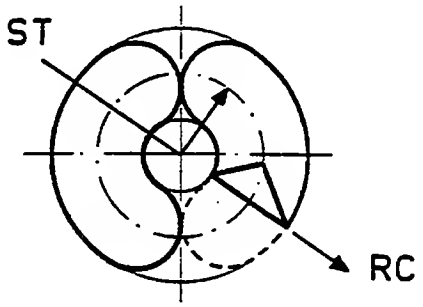
circular preblending bed



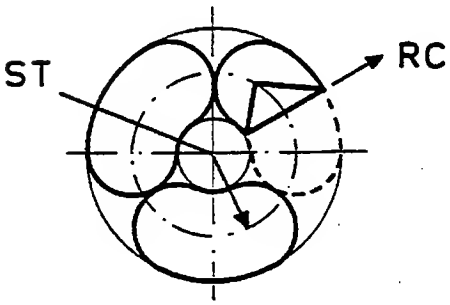
legend :
ST = stacking
RC = reclaiming



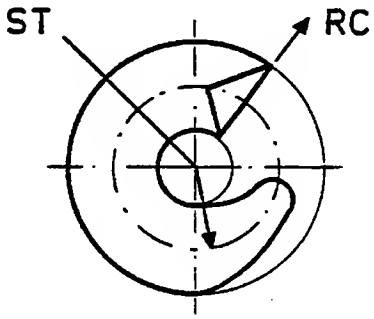
continuous stockpiling
(step wise)



two sectional pile
chevron method



three sectional pile
chevron method



chevcon method

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7. PREBLENDING BED APPLICATIONS

The selection of the appropriate preblending bed is dependent on many factors as listed below:

* Plant layout
* Material properties (stickiness, grain size, etc.)
* Environmental influences (weather conditions)
* Feeding capacity
* Storage capacity
* Raw mix proportioning
* Storage capacity expansion

7.1 Longitudinal Preblending Bed

The longitudinal preblending bed is best suited if a more component or integrated preblending bed is chosen, since a defined chemical value within one pile can be obtained.

However, material segregation in the end cones have an unfavorable influence on the homogenizing efficiency of the preblending bed.

The roofed preblending bed is a simple construction. Instead of a side stacker with a movable undercarriage the stacker can then be installed under the ridge of the roof.

But especially the unroofed preblending bed is considered favorable, since it is the cheapest solution.

7.1.1 Longitudinal Preblending with Side Reclaimer

The side reclaimer is a relative simple machine which is low in investment and low in maintenance cost. It is well suited also for very sticky materials.

The side reclaimer represents a cheap solution if several materials are stored on the same preblending bed in line, since the different piles can be served by one common side reclaimer.

The achievable homogenizing efficiency is rather poor, since not all layers are reclaimed simultaneously.

7.1.2 Longitudinal Preblending Bed with Front Acting Bridge Reclaimer

The bridge reclaimers should favorably be applied in the following cases:

* For materials which are not extremely sticky.
* If a good homogenizing efficiency is required.

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7.2 Circular Preblending Bed

Normally circular preblending beds are slightly cheaper because of less civil costs for roofing. However this statement depends much on the local condition. If the preblending bed is not covered this argument is inapplicable.

The respective advantages and disadvantages can be summarized as follows:

Advantages:

- No material segregation at the end cones if the next pile is formed adjacent to the foregoing or by continuous stacking.
- Good space utilization; this argument becomes only true for adjacent piles or continuous stacking.
- The continuous stacking method allows the adaptation of the homogenizing volume in comparison with the storage volume.

Disadvantages:

- When having an integrated preblending bed stacking is rather preferred in two or more segments (batchwise) than continuous. However stacking in segments has the drawback that before starting a new segment first a material free segment plus the space reservement for the reclaimer has to be free.
- The central discharge arrangement can become problematic if the material is sticky and difficult to handle.
- Machine parts as central slewing boom stacker bearing and lifting device are sensitive units exposed to excessive wear and therefore need special attendance.
- The extraction tunnel is exposed to 'water break in'.
- Expansion of the existing preblending bed is not possible.

7.3 Homogenizing Pit

The usual applied preblending bed type for sticky materials is the homogenizing pit. This type of preblending bed represents an expensive solution since excessive civil work is implied. As the material is stacked in longitudinal or transversal layers according to the windrow method a good homogenizing efficiency can be expected.

8. PREBLENDING BED CONCEPTS

The preblending bed concept depends very much on the raw material situation. The required number of raw material components, their homogeneity as well as the applied quarry concept are the basis for the elaboration of a suited preblending bed concept.

The following four concepts can be distinguished:

* Automatic storage
* Single component preblending bed
* More component preblending bed
* Integrated preblending bed

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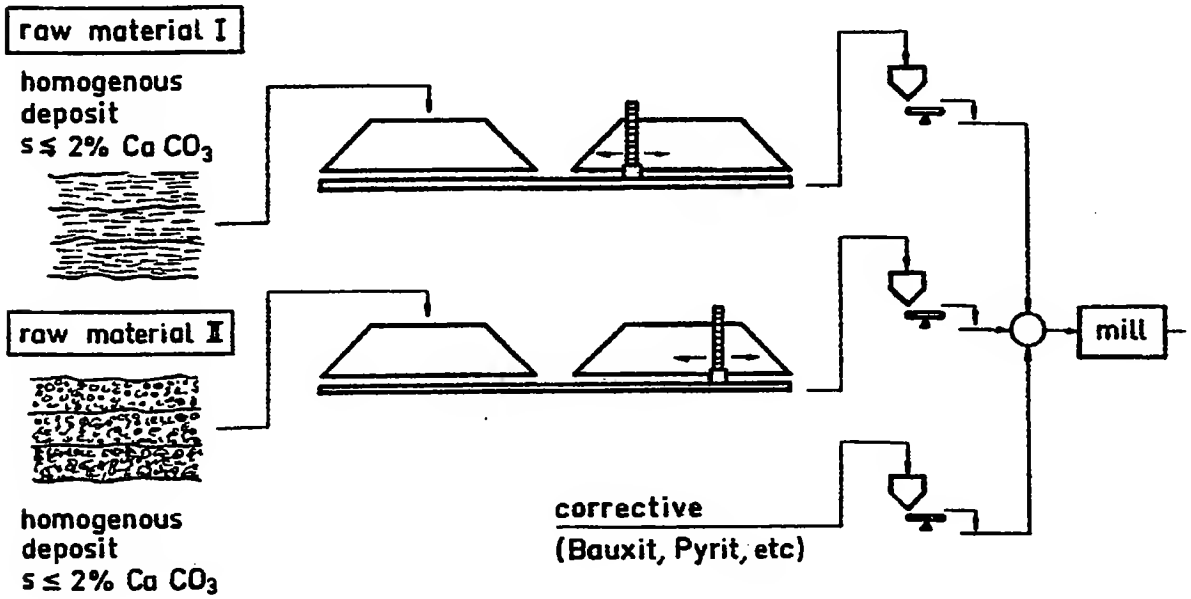
8.1 Automatic Storage

Assuming a raw material situation with homogeneous raw material components ($s \leq 2\ \% CaCO_3$) the preblending bed simply serves as an automatic storage deposit. No special caution with respect to raw material homogenization has to be given. In this case the simple and less expensive preblending beds with side reclaimers are applied (see fig. 39).

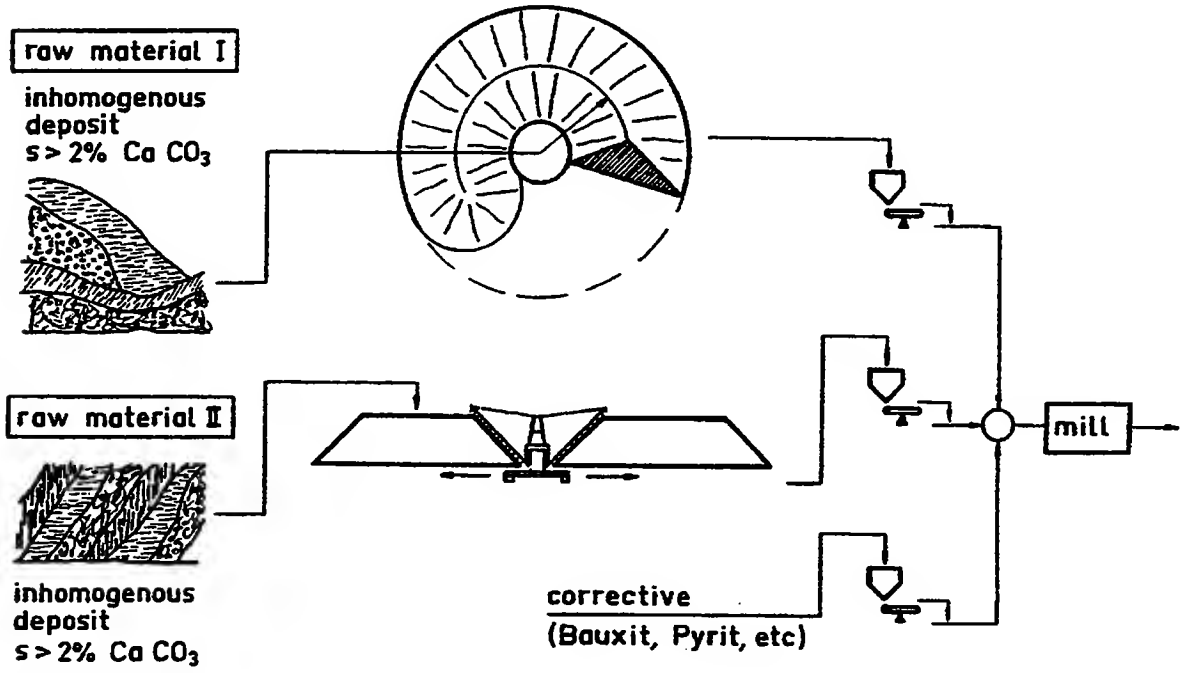
Figure 39 Automatic Storage



8.2 Single component Preblending Bed

The preblending bed is installed to effectively prehomogenize one single raw material component. This represents the 'classic' application of the preblending bed. It is applicable if the raw material component is a clearly defined raw material which is always above respectively below the desired average composition of the raw mix. Due to the less expensive roofing and the possibility to apply the Chevcon stacking. Circular preblending beds are in common cases preferred if the storage capacity is larger than 10'000 t (see fig. 40).

Figure 40 Single Component Preblending Bed



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8.3 More Component Preblending Bed

It is often the case that in the same quarry various raw materials are deposited. In this situation it can be advantageous to put all material qualities in the same preblending bed (no problems with separating the material during exploitation).

In order to guarantee a right proportioning of the raw mix, several solutions to apply the 'more component preblending bed' are possible:

- Providing two 'more component preblending beds' one being below the chemical target value of the raw mix, the other being above target (see fig. 41).
- Installing one 'more component preblending bed' followed by two large corrective silos which allow the final correction to the target value (see fig. 42).
- Controlling the 'more component preblending bed' in a way that is average chemical concentration is always below respectively above target so that the installation of one corrective silo only is sufficient (see fig. 43).

Figure 41 More Component Preblending Bed - 'High' and 'Low' Component

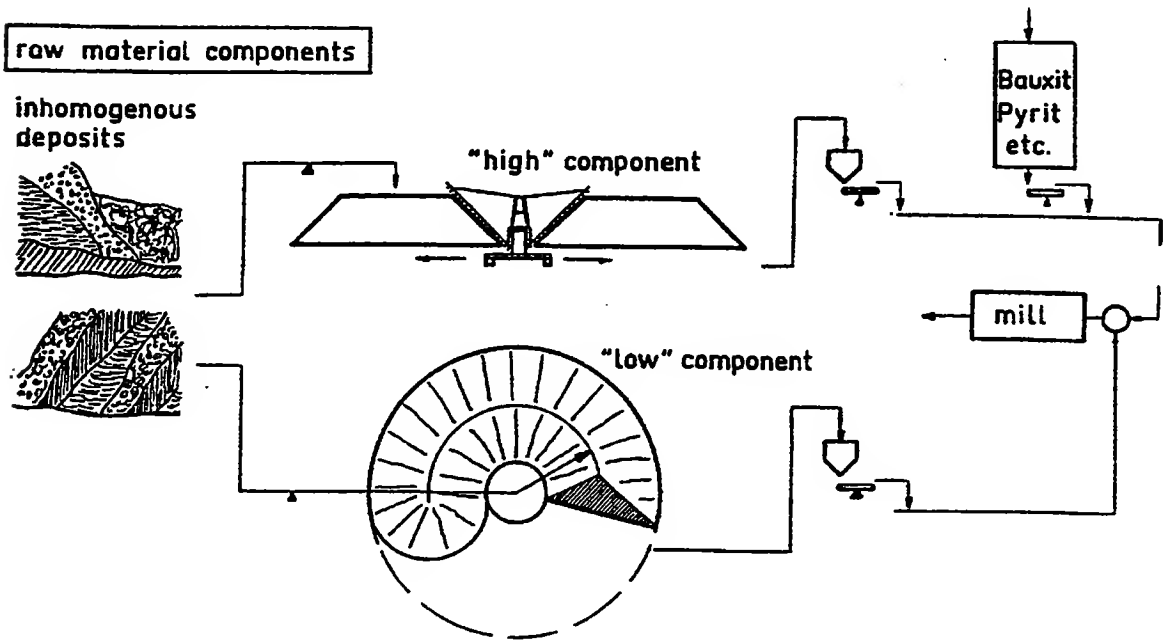


Figure 42 More Component Preblending Bed - 'High' and 'Low' Corrective Silo

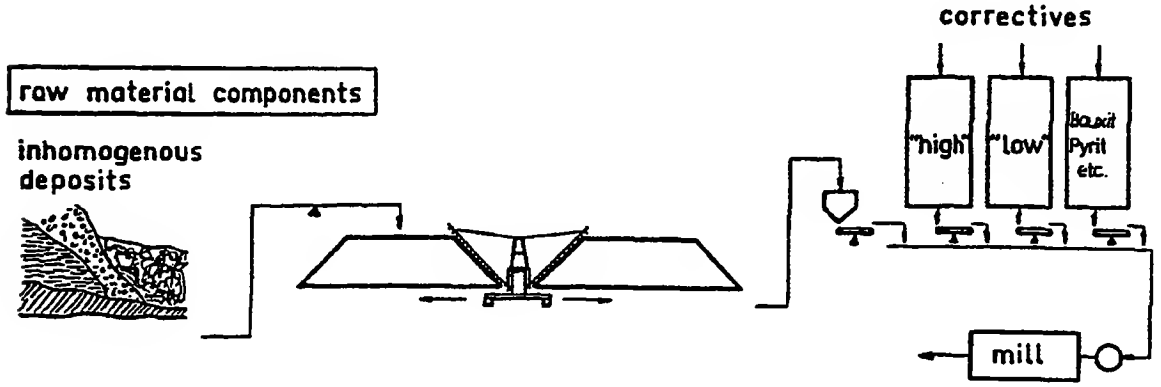
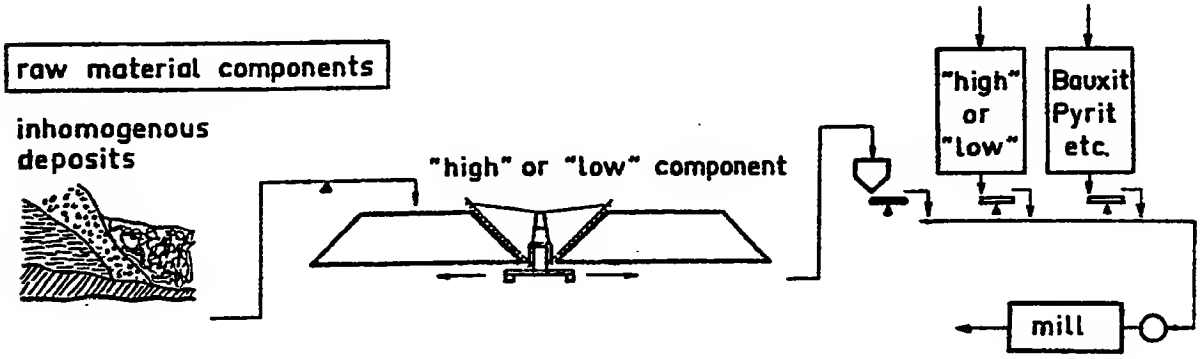


Figure 43 More Component Preblending Bed - 'High' or 'Low' Corrective Silo



8.4 Integrated Preblending Bed

Integrated preblending beds are 'more component preblending beds' which are closely controlled in their average pile composition by an automatic sampling station.

Today there is an increasing tendency to chose the integrated preblending bed as it implies some obvious advantages, e.g.:

- Complete utilization of a raw material deposit by controlled and selective quarrying.
- Stable kiln operation since the preblended material corresponds fully or nearly to the raw mix composition so that the chemical fluctuations of the kiln feed are very small.
- Possibility of storing a sticky raw material component together with non sticky materials so that material handling becomes easy (e.g. common crushing etc.).
- Only one preblending bed is required.

However, the advantages are faced to some not negligible drawbacks as listed below:

- The production line is dependent on a sampling station which requires high attendance.
- The required selective quarrying and controlling of the pile composition necessitate a qualified laboratory personnel.

Working principle

In an integrated preblending bed different materials (two ore more components) are dumped on the same pile in a controlled manner to attain a predesired chemical value. To reach this

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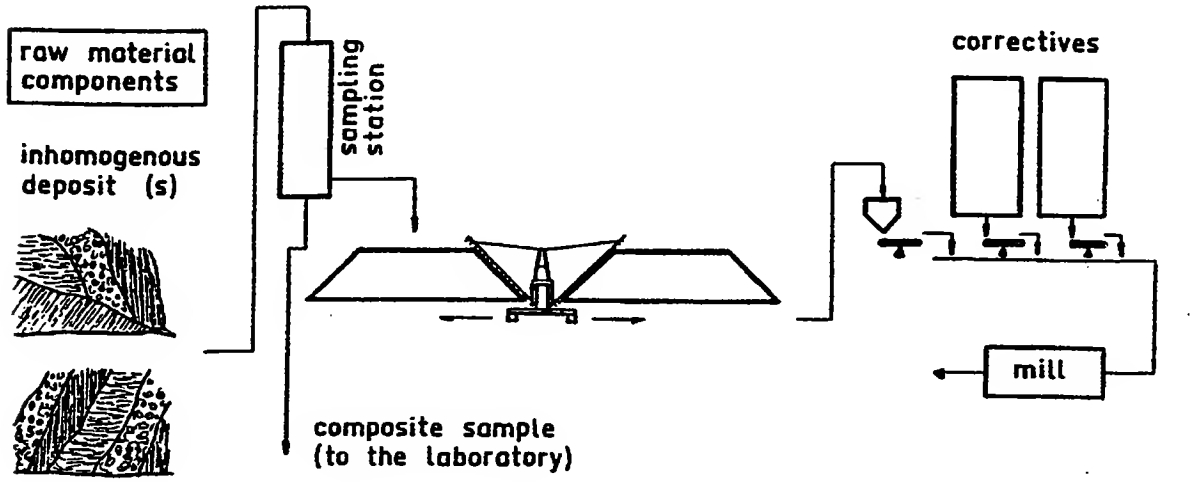
goal samples of about 1 % of the total material stream are continuously branched off in front of the preblending bed and prepared in an automatic sampling station.

Regularly (say hourly) composite samples are analyzed and together with the stacked material quantities the mean chemical composition of the pile is controlled.

Practice has shown that in most cases the integrated preblending bed can be controlled in such a way that no or only very little corrective material is required.

Figure 44 Integrated Preblending Bed



In fig. 44 the integrated preblending bed arrangement is schematically presented.

Application of the integrated preblending bed

The task to reach a predetermined chemical value within one pile is best obtainable by the batch type, longitudinal pile. The circular stockpiling is not well suited since the mean value is floating and needs considerably more careful supervision than the batch type piles.

8.4.1 Sampling Station

The proper functioning of the integrated preblending bed is to a far degree dependent on the sampling station. Only a regular and representative sampling of the material stream to the preblending bed guarantees to arrive at the exact desired chemical value. A sampling station has to fulfill generally the following requirements:

* Branch off of a defined sample quantity from the main material stream to the preblending bed
* Sample crushing
* Sample dividing
* Sample drying
* Transport to the laboratory
* Laboratory analysis

The sampling procedure as well as the selection of the special machines has to be adjusted to the individual case.

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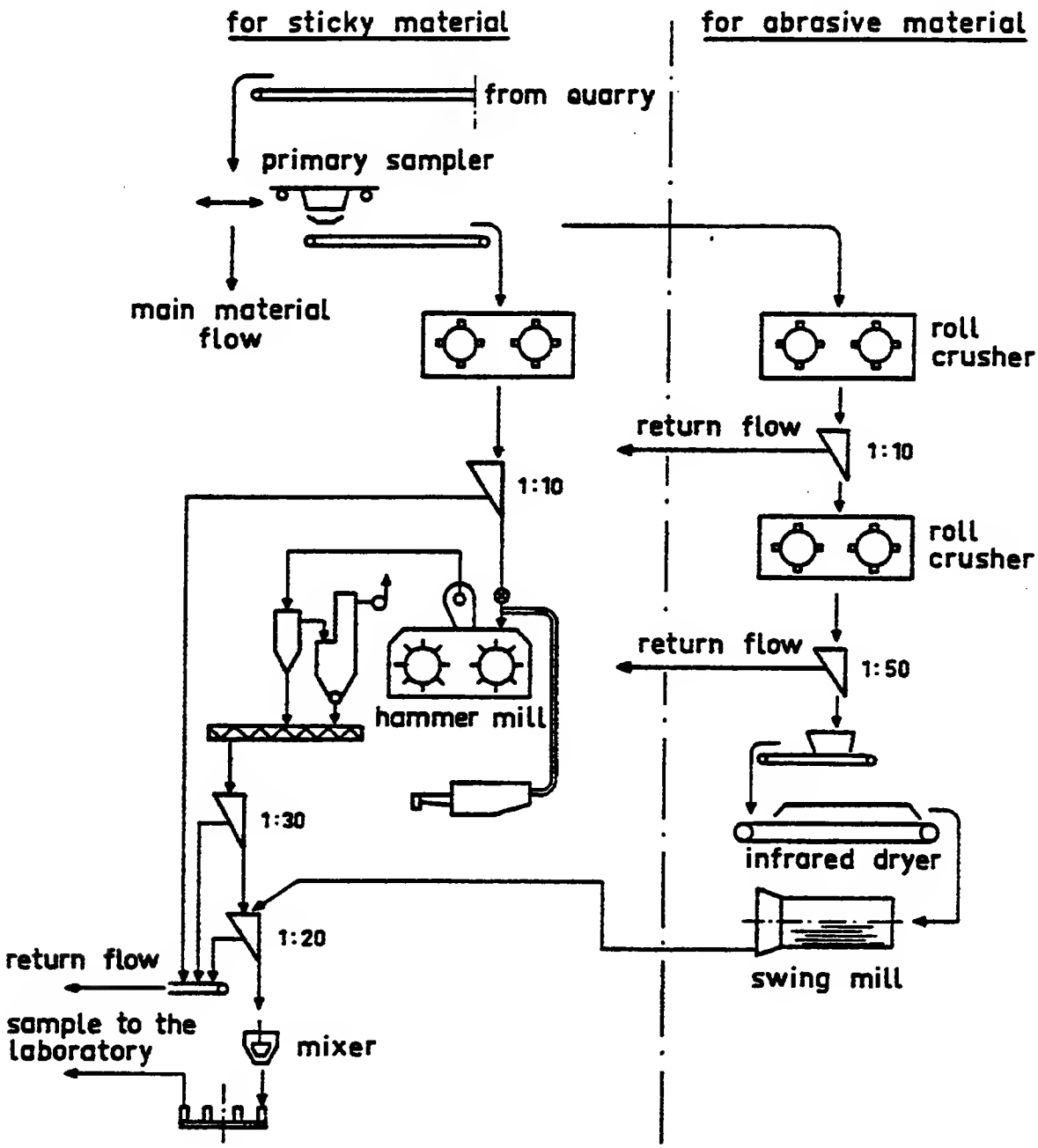
Typical applications for sticky and abrasive materials are illustrated in fig. 45.

Figure 45 Sampling Station



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Design of a sampling station

The sampling station aims to determine the mean chemical composition of the integrated preblending bed, given by

$$\bar{x} = \frac{\sum x_i m_i}{\sum m_i}$$

where:

x_i = analysis of the (hourly) integrated sample No. i
m_i = material quantity in the preblending bed, represented by x_i

Accuracy of the mean value (X)

The accuracy ε of the determined mean value (X) depends on factors as:

* Representativity of the sampling station
* Number of samples taken
* Sampling quantity

The following calculations are based on a confidence interval of 95 % corresponding to an accuracy of ε = +/- 2s (see 'aspects of raw material homogenization').

According to the accuracy ε, within the target mean value of the preblending bed, the type and quantity of corrective material must be chosen.

Representativity of the sampling station

The type of machines of the sampling station have to be selected by the criterion that each piece of raw material has the same chance to be sampled, i.e.

* Sampling of the whole cross section of the material flow
* Special design of the sampler (speed, slot width)
* Special design of the sample divider (speed, slot width)
* Reducing the grain size before the sample is divided
* Division rate of the samples not too large in one step

Required number of samples

For calculating the required number of samples the following relation will hold (see fig. 46):

$$n = \frac{s^2 k^2}{\varepsilon^2}$$

where:

n = required number of samples
s = standard deviation of sampled material
k = 2 (confidence factor for 95 % probability)
ε = required accuracy of the average bed composition

Based on n, the sampling frequency (f) is calculated as follows:

F = n/T

where:

the stacking time $T = \frac{bed\ \text{capacity}}{\text{stacking rate}}$

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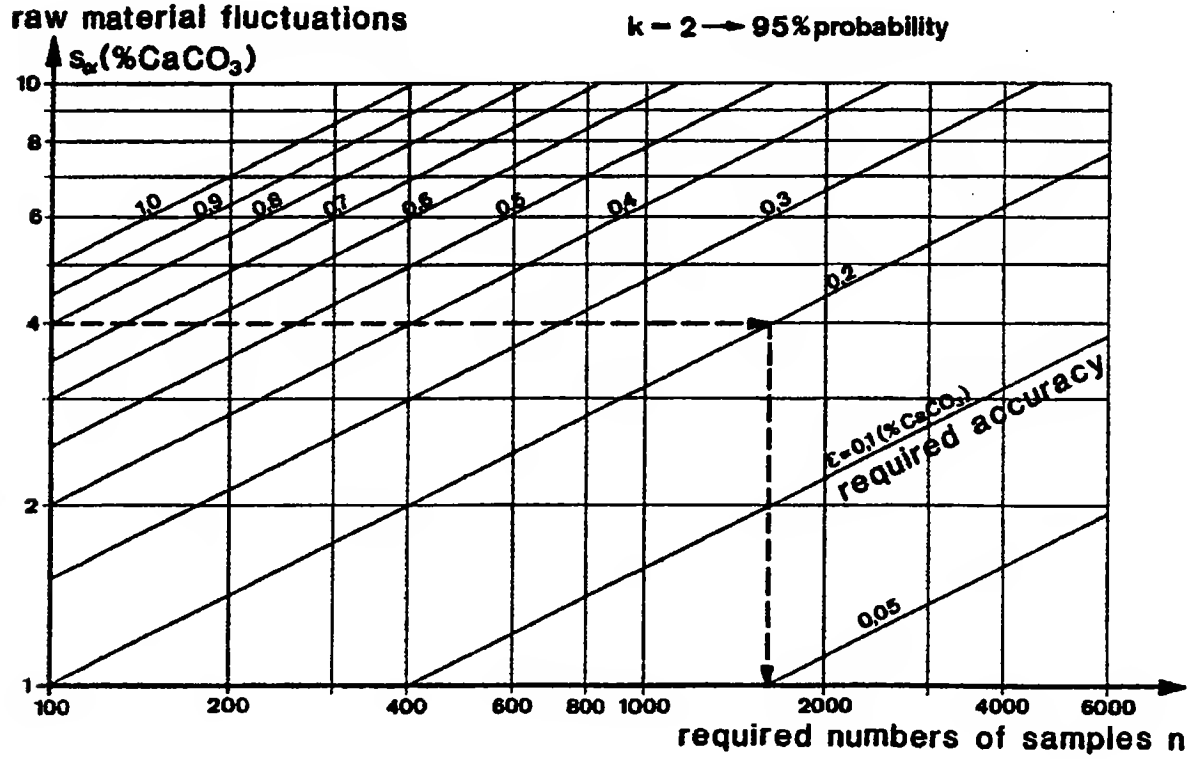
Figure 46 Required Number of Samples



Sampling quantity

Slot width of the sampler:

A sampler traveling across the material stream should be sufficiently wide that each grain has the same chance of being sampled, i.e.

B = 3d + 10 mm

where:

B = slot width (mm)
d = maximum grain size (mm)

Individual (primary) sample quantity (see fig. 47):

g = M B/W

where:

g = individual (primary) sample quantity (kg)

M = mass flow rate (kg/sec)

w = travelling speed of sampler (mm/sec)
(w) is normally choosen as 300mm/sec

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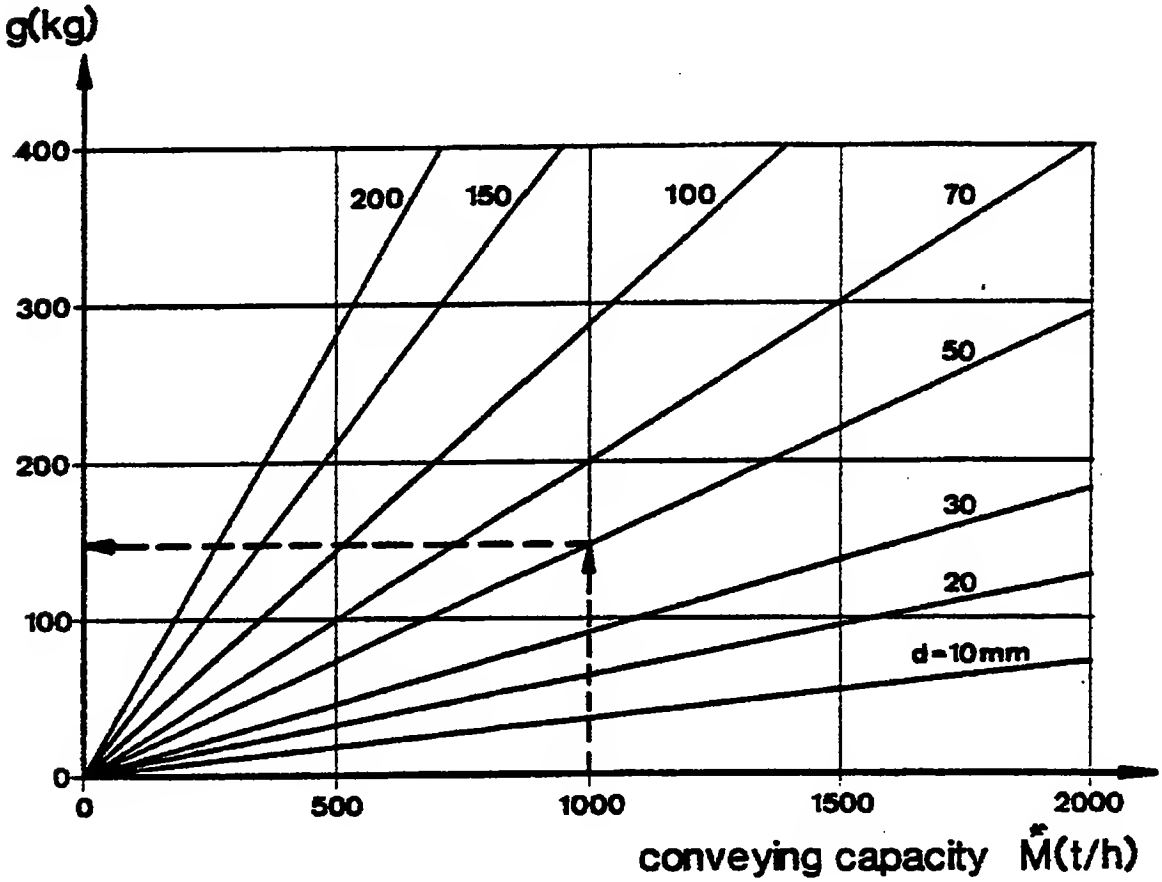
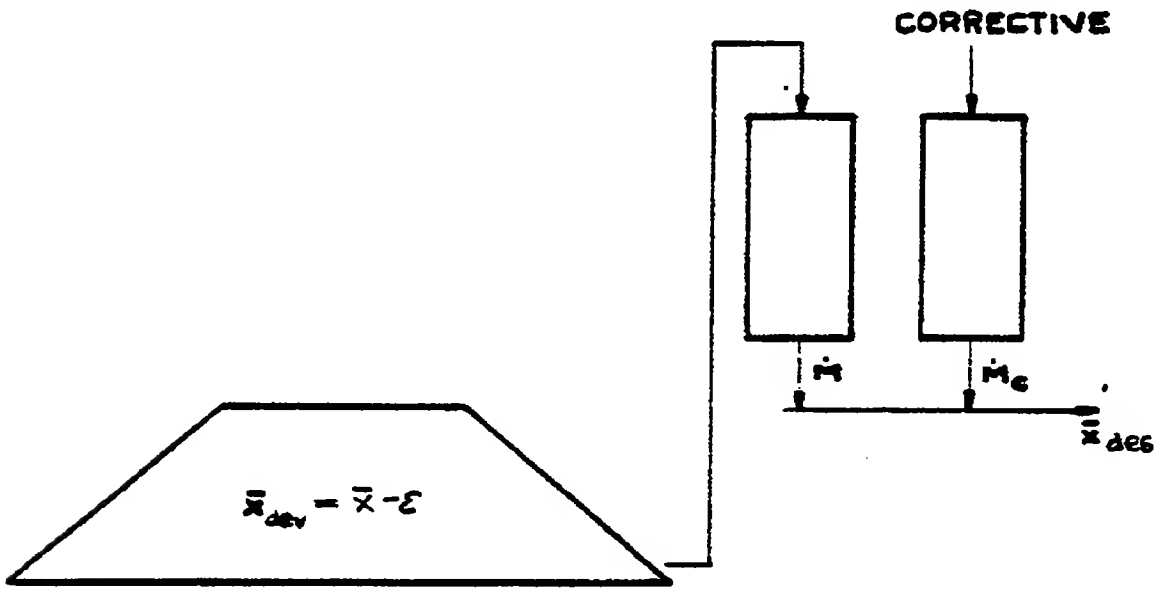
Figure 47 Required Sampling Quantity



Quantity of corrective material

Figure 48 Proportioning of Correctives



Proportioning formula:

$$x_{dev} * \dot{M} + x_c * \dot{M}_1 = (\dot{M} + \dot{M}_1) x_{des}$$

By setting $\varepsilon = [x_{des} - x_{dev}]$ the required amount of corrective material is calculated by the following formula:

$$\dot{M}_c = \frac{\varepsilon * \dot{\bar{M}}}{[x_c - x_{des}]}$$

or expressed in percentages (P):

$$P = \frac{\dot{M}_c}{\dot{M} + \dot{M}_c} * 100 = \frac{\varepsilon}{[x_c - x_{dev}]} * 100$$

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9. PREBLENDING BED DIMENSIONING AND DESIGN

9.1 Dimensioning of the Preblending Bed

- Estimation of the required homogenizing efficiency (see chapter 2).

$$e = f(s_\beta / s_\alpha, \rho_\alpha, T/M, N)$$
$$1) \quad 2) \quad 3)$$

1) Standard deviation of the preblended material $s_\beta \leq 1$ % $CaCO_3$
2) Determination of the chemical input fluctuations.
3) Calculation of the number of layers by assuming a certain capacity of the preblending bed.
   For operation and storing reasons the capacity of the preblending bed should normally correspond to approx. 1 week production capacity.

- Choice of the type of preblending pile: longitudinal pile (side or front reclaimer, homogenizing pit) or circular preblending bed (see chapter 7).
- The required number of layers (N) determined in (a) leads to a certain cross-sectional area of the pile or to the width (B) resp. (Da - Di) of the pile:

$$N = F\frac{v}{q}, \quad F = \frac{B^2 \tan\alpha}{4} \text{ or } \frac{(D_a - D_i)^2}{16} * \tan\alpha$$

where:

F	=	cross-sectional area
v	=	stacker velocity
q	=	mass flow rate
B	=	pile width for longitudinal pile
$\frac{D_a - D_i}{2}$	=	pile width for circular pile

The nomograms 1,2,3 resp. 4 allow a quick determination of (B) resp. (D_a - D_i).

- In case of a longitudinal bed, check whether the assumed preblending bed capacity has a minimum length to width ratio of L/B ≥ 3.
  In case of a circular preblending bed, check whether the resulting (D_a - D_i) allow to incorporate a storage capacity as assumed in (a). In case of Chevcon stacking check also the available homogenizing volume.
  Enlarge (D_a - D_i) if the desired bed capacities cannot be incorporated.
- Determine the measures to be taken against cone segregation, normally
  - adjustable reversal points of the stacker
  - dead stock at the rear ends

(see chapter 5)

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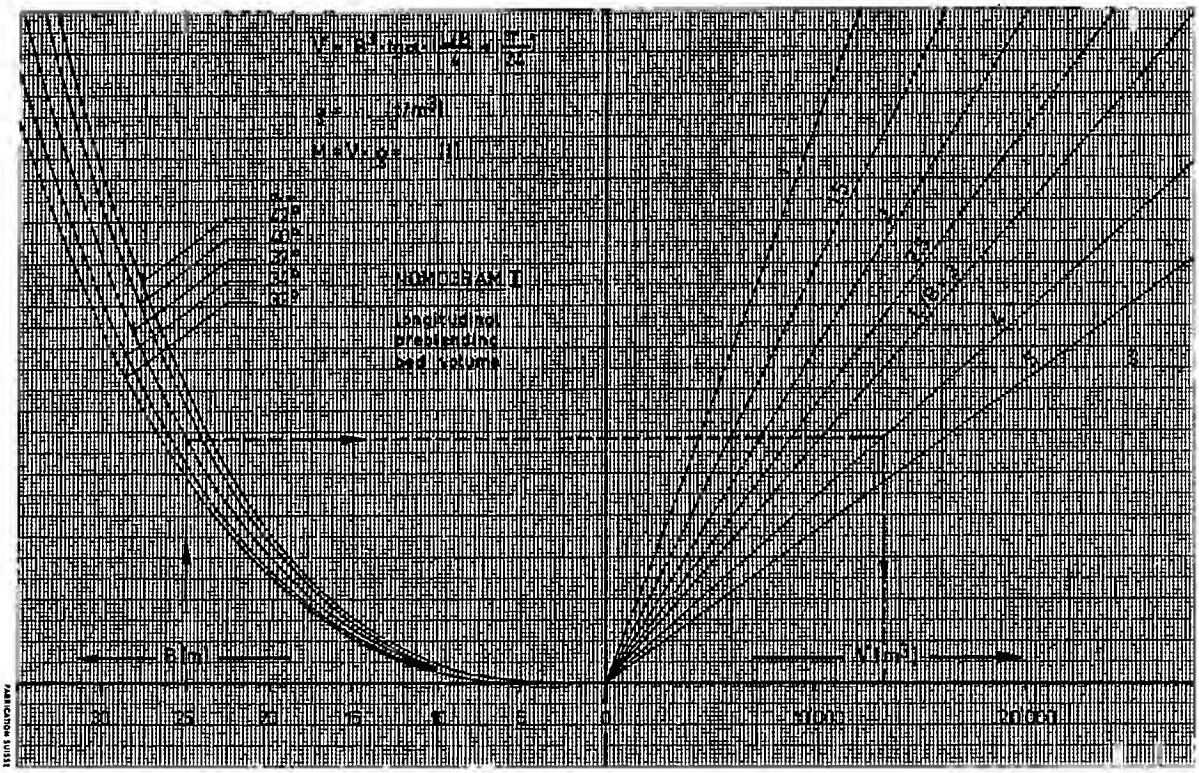

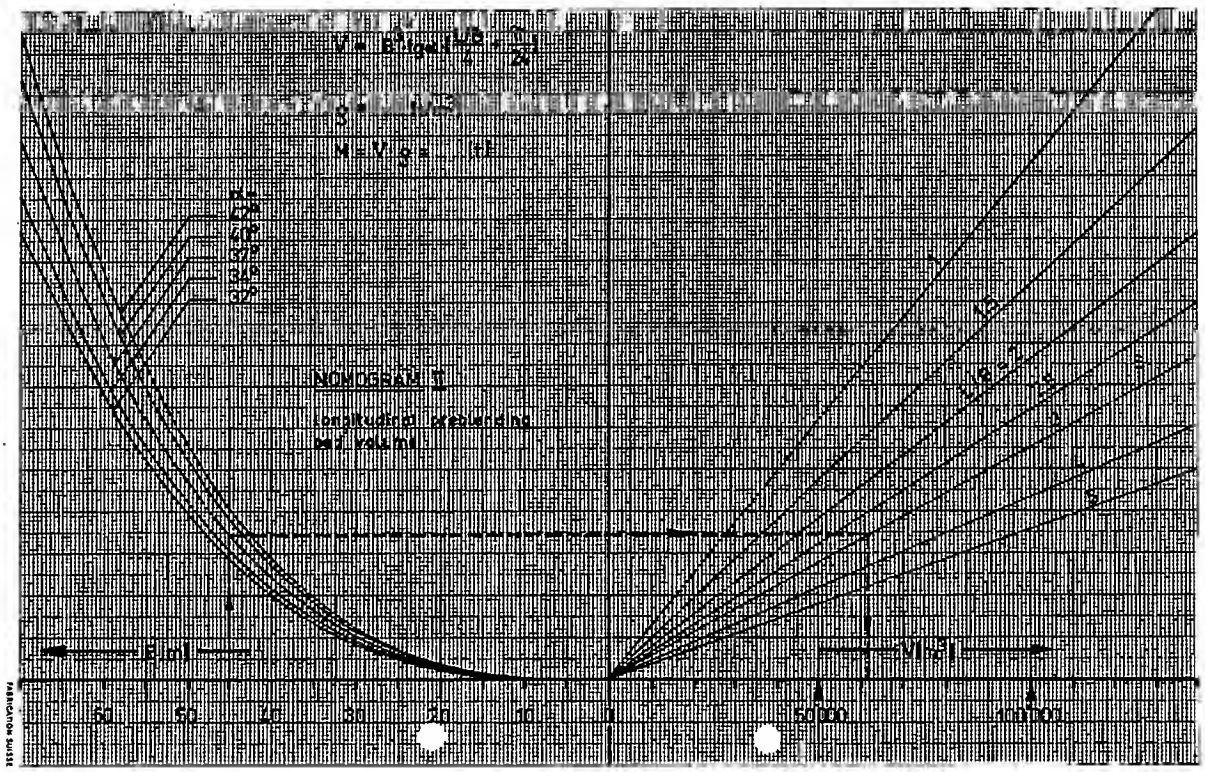

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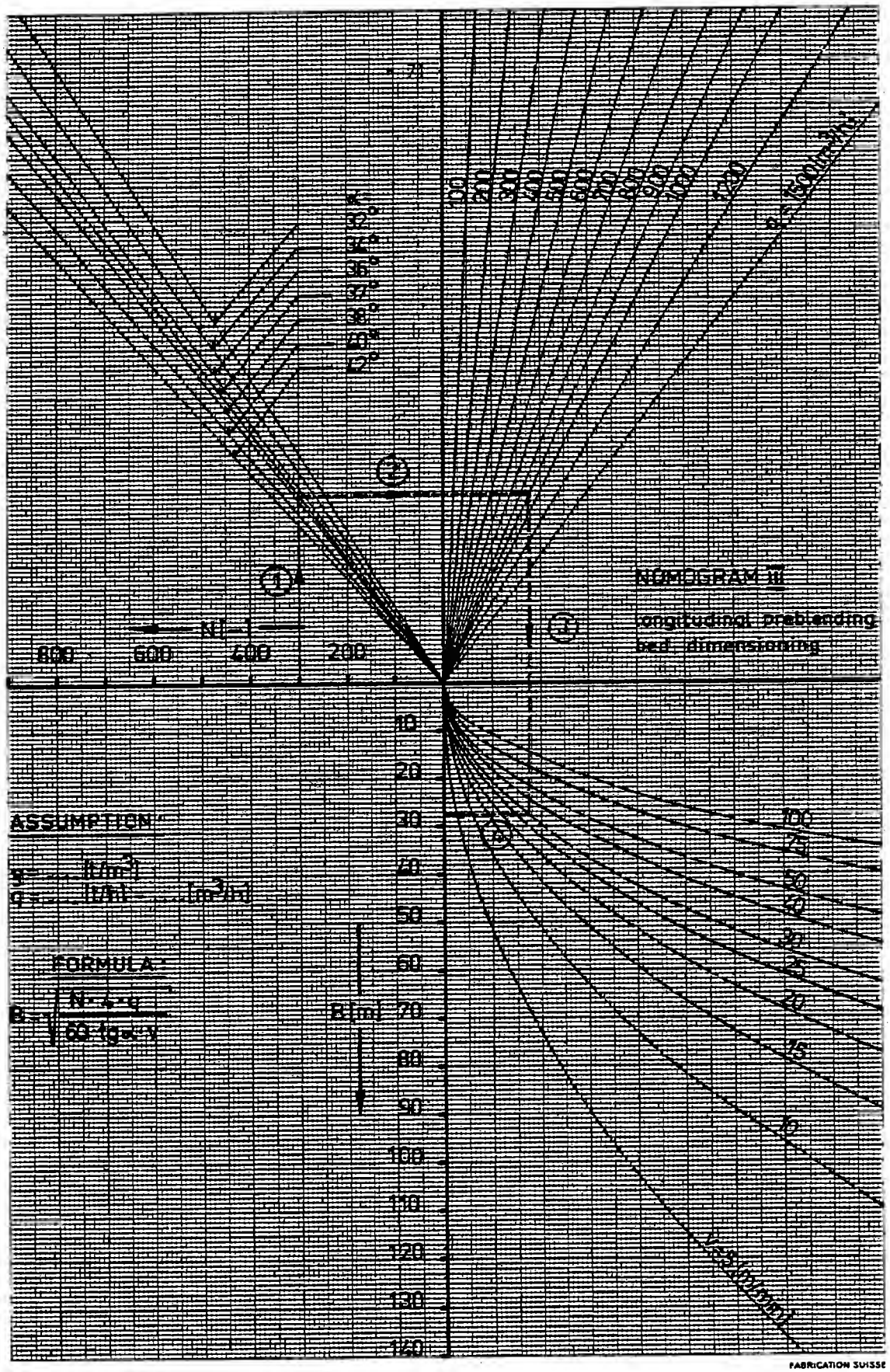
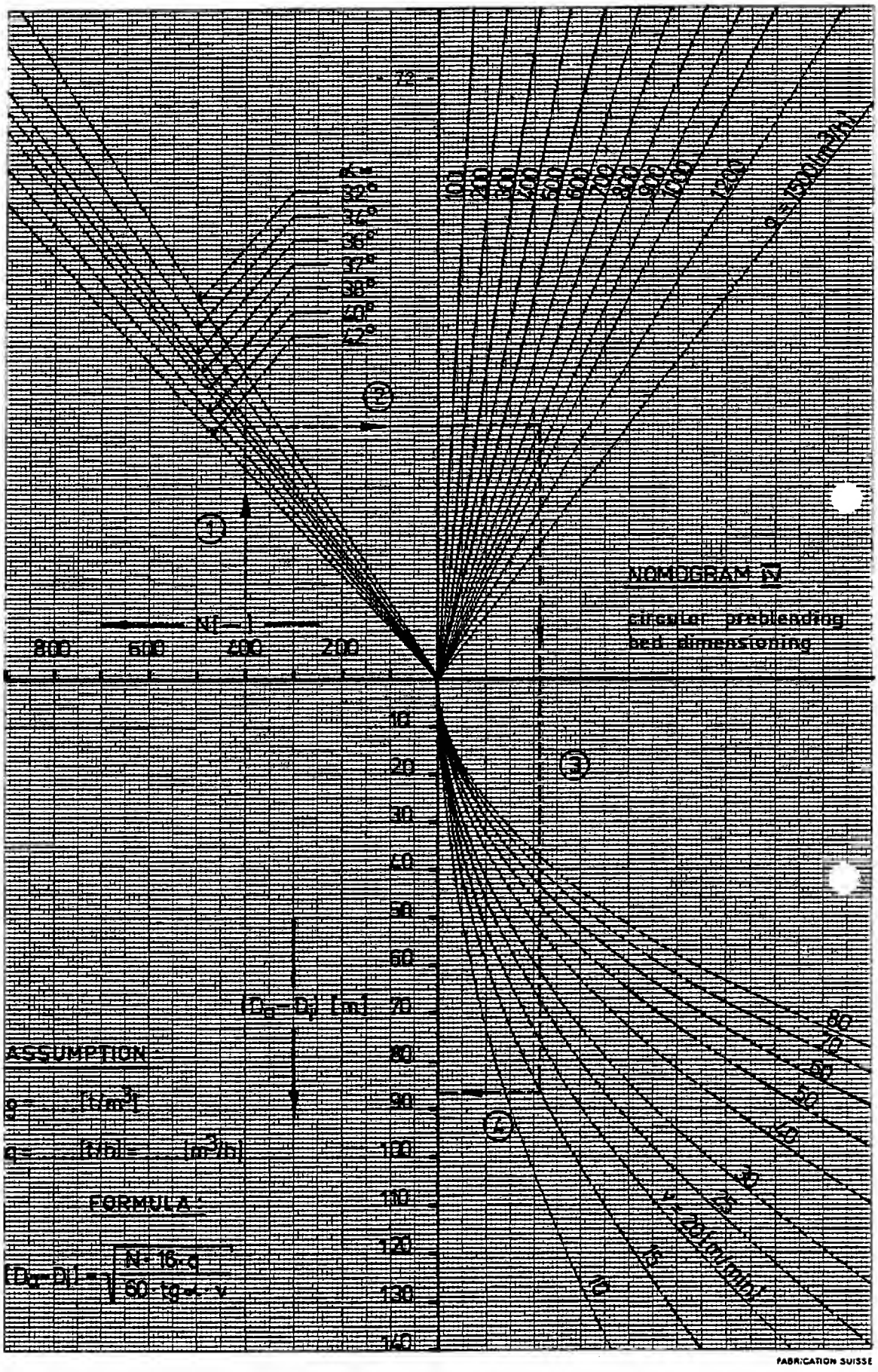

NOMOGRAM III
longitudinal preblending bed dimensioning
ASSUMPTION
FORMULA
B [m]
FABRICATION SUISSE



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9.2 Design Aspects of the Preblending Bed

On selecting the proper machine equipment attention to the below listed facts should be given.

- Power rates and speed range of all travel drive systems (e.g. stacking machine, reclaiming machine with scraper chain and raking device for dislodging the material).
- Rakes wiping the whole cross-section in few minutes time, equipped with renewable teeth.
- Synchronized speed control of the reclaimer drive to compensate skew-running tendency.
- Angle of repose not more than 38° for safety reasons.
- Lowerable boom to avoid dust.
- Scraper chain design (e.g. horizontal, inclined, block link or roller chain).
- Blade spacing and size
- Material of the blades
- Material of the chain
- Blades equipped with replaceable wear pieces and ripper teeth.

9.2.1 Machineries Especially for Circular Piles

Check points as mentioned above concerning the stacker, reclaimer bridge and scraper chain.

The center discharge as well as the feed point of the stacker are exposed to clogging problems when dealing with sticky materials and should therefore be examined carefully in that respect.

The central slewing boom stacker bearing is considered to be a delicate machine part and needs special attendance.

The lifting device of the stacker which is exposed to excessive wear oblige special observance.

9.2.2 The Below Announced Figures should Assist as Guide Values:

Stacking machine:

The stacker speed should be limited at max. 30 m/min.

Reclaiming machine:

The front acting bridge reclaimer should be equipped with a fast traveling speed of 10 m/min. to dislodge the machine to the next pile and an operating speed of 0.005 to 0.05 m/min. The **scraper chain** speed should be in the range of 0.5 m/s.

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10. ACCEPTANCE TEST

The proof of proper functioning of a preblending bed under most extreme circumstances as expected during operation shall be subject to an acceptance test.

Extreme test conditions are:

- maximum moisture content
- maximum input fluctuations
- maximum input material flow

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Chapter 5

Raw Meal Homogenization

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A Survey on Homogenising and Blending Silos and their Operation

F. Bucher
PT 98/14361/E

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Summary

In Cement Industry raw meal blending or homogenisation is always done in silos. It is the last beneficiation step in the line of the raw mix preparation processes installed with the aim to reduce the residual (relatively short-term, high frequent) compositional variations observed for the raw meal produced in the raw mill. The raw meal reclaimed from such blending or homogenisation silos will then be fed to the kilns without further beneficiation. Therefore it is a challenge for all cement plant operators to achieve for the raw meal ex blending or homogenising silo a quality that meets narrow uniformity specifications regarding chemical composition and physical characteristics. This as a prerequisite for achieving steady process conditions for the kiln.

The paper gives a survey on the silo concepts used in raw meal beneficiation as well as their valuation.

The beneficiation efficiency achieved with such silos is not only function of the selected silo configuration but also of the kind of compositional disturbances produced in raw mix composition. The major lesson to learn is that the raw meal beneficiation silos can not be blamed for all the errors committed in raw mix composition. Hints regarding optimum operation of raw meal silo systems are given.

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1. INTRODUCTION

In Cement Industry raw meal blending or homogenisation is always done in silos. It is the last beneficiation step in the line of the raw mix preparation processes (acc. Fig. 1) installed with the aim to reduce the residual (relatively short-term, high frequent) compositional variations observed for the raw meal produced in the raw mill. The raw meal reclaimed from such silos will then be fed to the kilns without further beneficiation. Therefore it is a challenge for all cement plant operators to achieve for the raw meal ex blending or homogenising silo a quality that meets narrow uniformity specifications regarding chemical composition and physical characteristics. This as a prerequisite for achieving steady process conditions for the kiln.

Besides final compositional beneficiation blending and homogenising silos serve as intermediate stores separating the two continuous processes raw grinding and clinker burning which are not necessarily operated at similar rates.

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2. COMPOSITIONAL UNIFORMITY AND BLENDING FACTOR

In cement industry it is common to specify kiln feed uniformity

- in terms of variations in clinker compounds (%CaO; %CaCO3 (titration)) or
- in terms of clinker moduli (%LSF; %C3S; etc)

using simple statistical terms such as the Average and the Standard Deviation.

The simplest and most common statistical measure is the Average or Mean. Given a set of N measurements, X_1, X_2, , X_N the mean value X is given by

$$\overline{X} = \frac{1}{N}(X_1 + X_2 + ... + X_N) = \frac{1}{N}\sum_{i=1}^{N} X_i \qquad (1)$$

Deviations from the mean are expressed in terms of the Standard Deviation S, given by

$$S = \sqrt{\frac{(X_1 - \overline{X})^2 + (X_2 - \overline{X})^2 + ... + (X_N - \overline{X})^2}{N-1}} \qquad (2.1)$$

or

$$S = \sqrt{\sum_{i=1}^{N} \frac{(X_i - \overline{X})^2}{N-1}} \qquad (2.2)$$

The difference between each measurement Xi and the mean value X are squared so that positive and negative fluctuations above and below the mean do not cancel each other. The square root of the sum of the squared variations is then divided by the number of measurements N to obtain an average measure of variation, having the same units as the measured quantity.

The Standard Deviation S allows for the following simple interpretation:

- the characteristic tested of 68 % of all samples will fluctuate within a range of ± 1S
- the characteristic tested of 95% of all samples will fluctuate within a range of ± 2S
- the characteristic tested of 99 % of all samples will fluctuate within a range of ± 3S

The natural and induced blending which occurs at a particular beneficiation stage may be expressed by a Blending Factor BF defined as the ratio of the incoming and discharge Standard Deviations:

$$BF = \frac{S_{in,corr}}{S_{out,corr}} = \frac{\sqrt{S^2_{in,measured} - S^2_{in,error}}}{\sqrt{S_{out,measured} - S^2_{out,error}}} \qquad (3)$$ as blending/homogenising factor

The Standard Deviations as measured need correction for sampling and analysis errors for which the silo can not be blamed. For determining the sampling and analysis error Merks[1] double sampling method is proposed. Evaluation of the test is given in Annex 1.

[1] JW Merks, Sampling and Weighing of Bulk Solids, Trans Tech Publications 1985

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In Cement Industry daily practice demonstrate that compositional variations of the kiln feed have an adverse effect on kiln operation (coating formation, temperature profile, encrustation due to unstable evaporation of the circulating elements (SO_3, Alkalis, Cl^-), etc.) and thus on brick life and kiln availability. The problem gets even more complex by the fact that not all industrial raw meals need to be uniform to the same extent. Easy burning raw mixes tolerate fluctuations in a wider range than difficult burning raw mixes.

Nevertheless it is useful to have some guide values regarding tolerable compositional fluctuations at hand. In Cement Industry it is generally accepted that no further improvement of raw meal quality can be expected by additional blending/homogenisation for the kiln feed variations given in below table.

Characteristic		**analytical error excluded [standard deviation s]**	**analytical error included [standard deviation s]**
$CaCO_3$	%	≤ 0.2	≤ 0.25
$CaCO_3$ max.	%	≤ 0.3	≤ 0.35
CaO	%	≤ 0.11	≤ 0.15
CaO max.	%	≤ 0.17	≤ 0.2
SiO_2	%	≤ 0.1	≤ 0.15
Al_2O_3	%	≤ 0.07	≤ 0.12
Fe_2O_3	%	≤ 0.04	≤ 0.04
LSF	%	≤ 1.0	≤ 1,5
SR	%	≤ 0.04	≤ 0.06
AR	%	≤ 0.06	≤ 0.08

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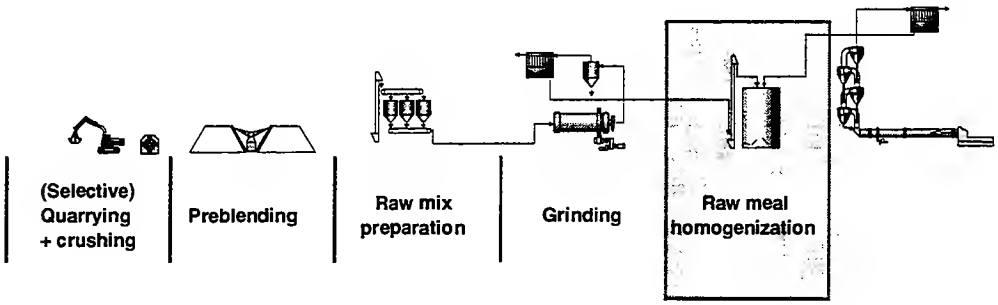
Figure 1: A cement works blending / homogenizing sequence



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3. SILO CONCEPTS FOR RAW MEAL BENEFICIATION

The silo concepts used in raw meal beneficiation can be classified according to the kind of the working principle applied into the following categories:

- Air-fluidised systems,
- Aerated gravity systems,
- Non-aerated gravity systems.

3.1 Air-fluidized Systems

3.1.1 Process Concept

Air-fluidised homogenising systems aim at raw meal beneficiation prior to be stored. A typical system set-up is the two-storey silo concept with the homogenising silo installed on top of a storage silo (Fig. 2).

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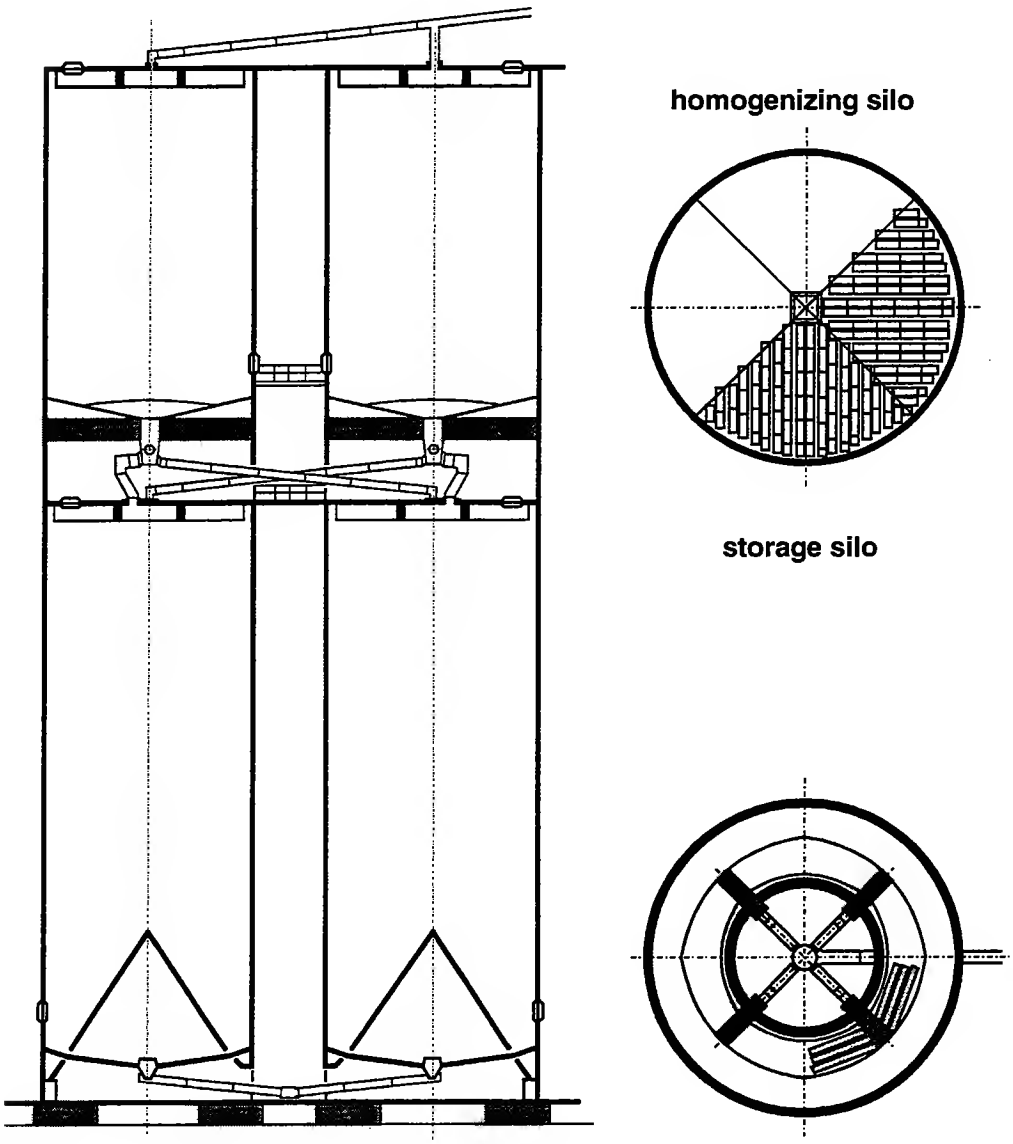
Figure 2: Air Fluidized silo systems - Batchtype, tow storey arrangement



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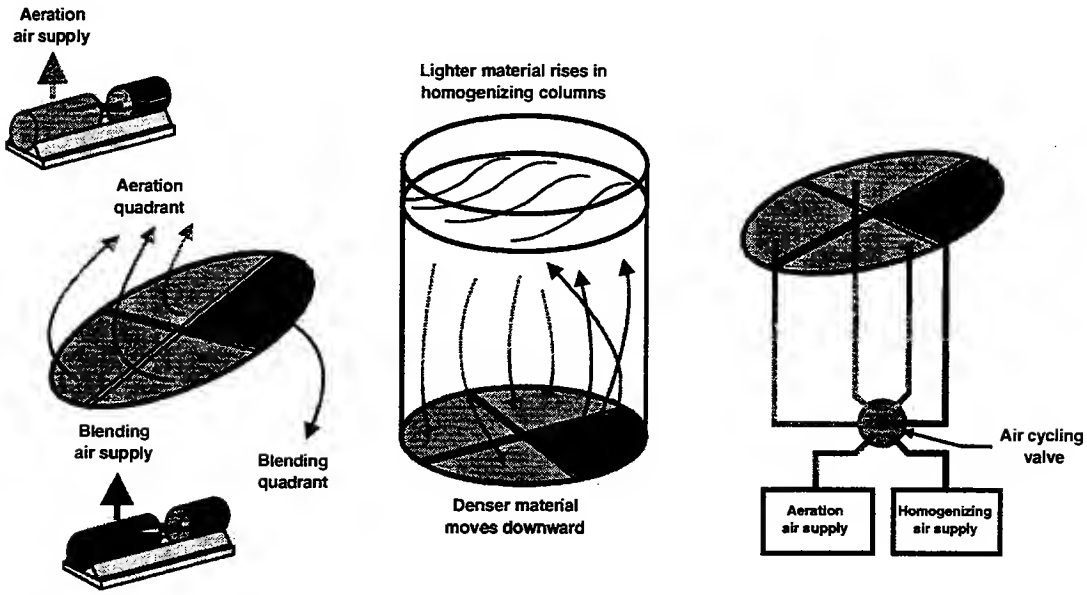
Such systems follow the concept of the fully agitated mixer. For that purpose compressed air is introduced through a permeable media covering the silo's bottom. Aeration causes the cement raw meal to behave as a liquid. By variation of the airflow through the raw meal bed the individual particles are forced to move relative to each other what result in efficient homogenisation (Fig. 3).

Figure 3: Air fluidized silo systems - The Quadrant System



As to achieve a variation of the airflow through the raw meal bed the silo bottom is divided into segmented areas, typically into quadrants or octants. Typically two air compressors are installed for aeration air supply. Homogenising air is supplied at a high rate and a high pressure into the selected homogenising sector (one quadrant or octant) and creates by this an extremely active, low-density column of raw meal. At the material surface this upward moving blending column spills onto denser, downward moving material located over the aeration sectors. Air supply to the aeration sectors is at much lower rate and at lower pressure. Active homogenising aeration is switched systematically at regular time intervals by means of a special valve sector by sector.

The design of storage silos is quite similar to the design of continuous blending silos, which will be discussed in Para 3.2.

3.1.2 Design concepts

Fluidised homogenising silos are designed according two concepts:

- as batch-type silos or
- as continuous-overflow type silos.

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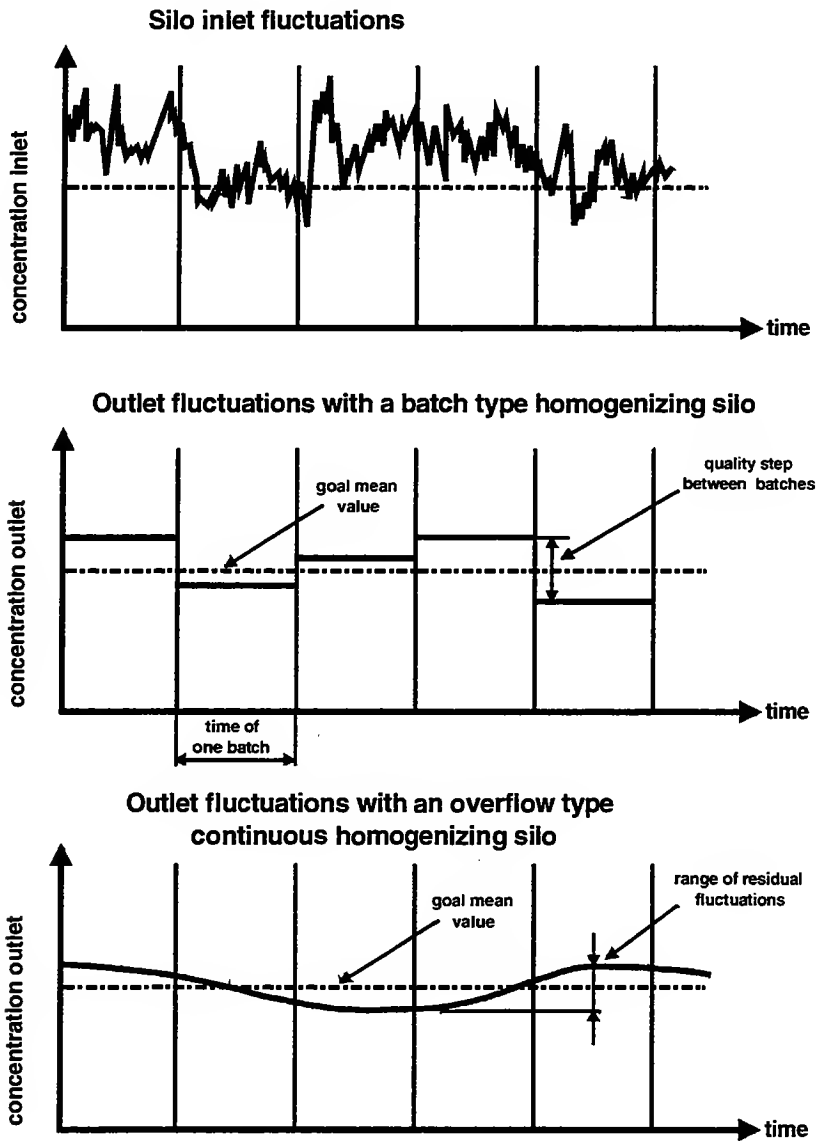
3.1.2.1 The batch-type silo concept

- Homogenising effect

The homogenising effect of air-fluidised silo system when operated in batch-wise mode may be as high as 15:1. Air fluidised silo systems are thus most efficient in raw meal beneficiation but one has to accept small compositional differences between the single batches (Fig. 4). These differences become the smaller the closer the goal value for raw mix composition is achieved by component adjustment at mill inlet.

Figure 4: Homogenizing / blending effect of raw meal silos



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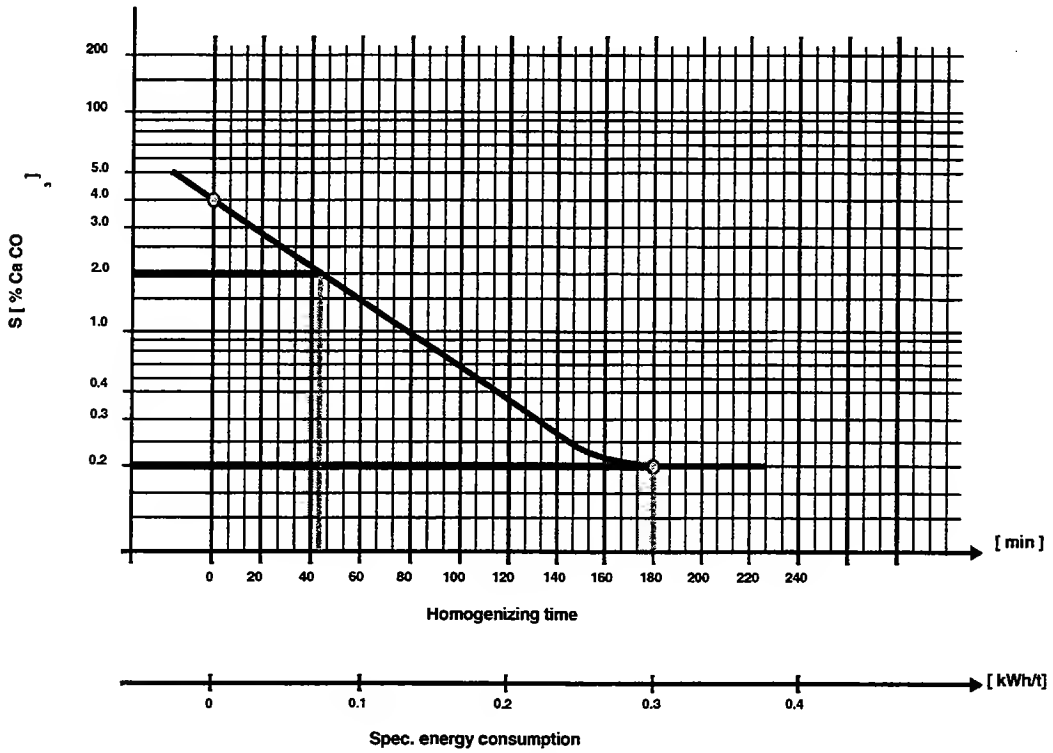
- Energy consumption

Energy consumption with air-fluidised silos is in a range of 0.7-1.0 kWh/t, thus important energy consumers. Nevertheless batch type operation of homogenising silos may be optimised regarding power consumption just by limiting compressor run-time to the minimum required for achieving a sufficient compositional uniformity. It is often observed that compressor run-time is excessive without that raw meal quality can be further improved (Fig. 5)

Figure 5: Air fluidized silo systems - The Varioflow System



Suppliers:

- BMH-CPAG
- IBAU
- Fuller-Kovako

3.1.2.2 The continuous-overflow concept

- Homogenising effect

The continuous-overflow operation mode was developed as to overcome the step type quality changes common for batch-type silo systems (Fig.4). Nevertheless one has to accept a slightly reduced homogenising efficiency due to short-circuit product leaving the silo immediately without being homogenised.

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- Energy consumption

  With the compressors permanently running energy consumption of overflow-type silos is in a high range of 1-1.5 kWh/t, thus significantly higher compared to the batch-type silos

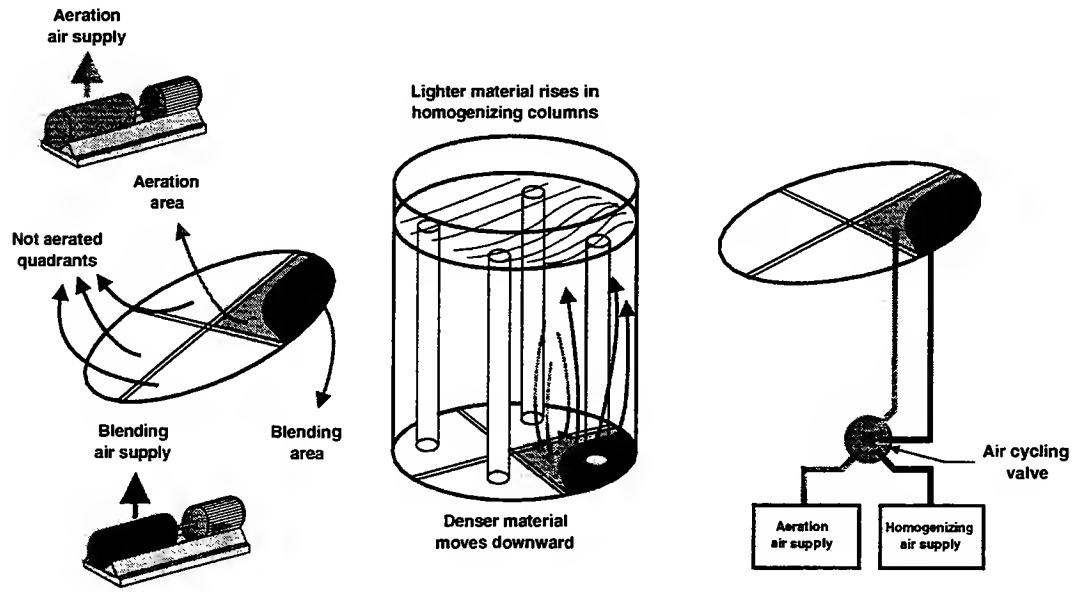
Suppliers:

- BMH-CPAG
- IBAU
- Fuller-Kovako

3.1.2.3 The Vario-Flow concept

The Vario-Flow silo concept (Fig. 7), a further development of the overflow silo concept, was developed as to reduce system energy consumption. Again the silo bottom is divided in quadrant areas but each of the quadrants is in addition divided into a homogenising and an aeration area. By this air supply may be reduced what results in reduced system energy consumption.

Figure 6: Air fluidized silo systems - The Varioflow System



- The homogenising effect is similar as with a conventional overflow system.
- Energy consumption is about 0.9 kWh/t.
- Supplier:
- BMH-CPAG

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3.1.3 General Design Characteristics:

Design characteristics		Homogenising Silo	Storage Silo
capacity (hours of mill operation)	h, d	10-12 h	1-3 d
height/diameter ratio	-	1.2-1.5 : 1	up to 2.5:1
net aeration area (% of bottom area)	%	50-70	35-50
active aeration air rate (specific)	$m^3/minm^2$	1.5-2.0	
active aeration air pressure	bar	up to 2.5	
aeration air rate (specific)	$m^3/minm^2$	1.0-1.5	1.0-1.5
aeration air pressure	bar	0.6-0.8	0.6-0.8
energy consumption (specific)	kWh/t	0.9-1.5	0.1-0.3
range of fluctuations that can be reduced	h	10-12	
homogenising effect	-	up to 15:1	

3.1.4 Valuation of the air-fluidised silo concept

+ most efficient raw meal homogenising system
- high energy consumption, above all when operated in a continuous-overflow mode
- application limited to capacities of about 2000 t corresponding to 3000 t/d clinker production lines
- high investment

Development of the air-fluidised silo concept has to be seen in context with the introduction of the dry process in Cement Industry. At that time efficient raw meal blending was indispensable as a consequence of the non-availability of efficient raw material preblending systems. The air-fluidised silo systems lost ground in favour of continuous blending silos (i.e. inverted cone type gravity systems) with the introduction of efficient preblending systems. New air fluidised homogenising silos will hardly be installed anymore in Cement Industry.

Still operational air fluidised homogenising silos need not to be modified at all costs. But there operating cost may in many cases be reduced by optimising the homogenising sequence. In that respect batch-wise operation at shortest homogenising time is preferred to a continuous-overflow operating mode.

3.2 Aerated gravity Systems

3.2.1 Process Concept

Aerated gravity systems aim at raw meal beneficiation and intermediate storage in one common silo.

The system follows the concept of a blender. For that purpose the raw meal is fed into the silo in horizontal layers. When reclaiming meal from the silo a funnel will form on top of the discharge point at the product surface. The declining funnel surface cause blending of particles originating from different layers when sliding down the slope to into the transport channel. (Fig. 8).

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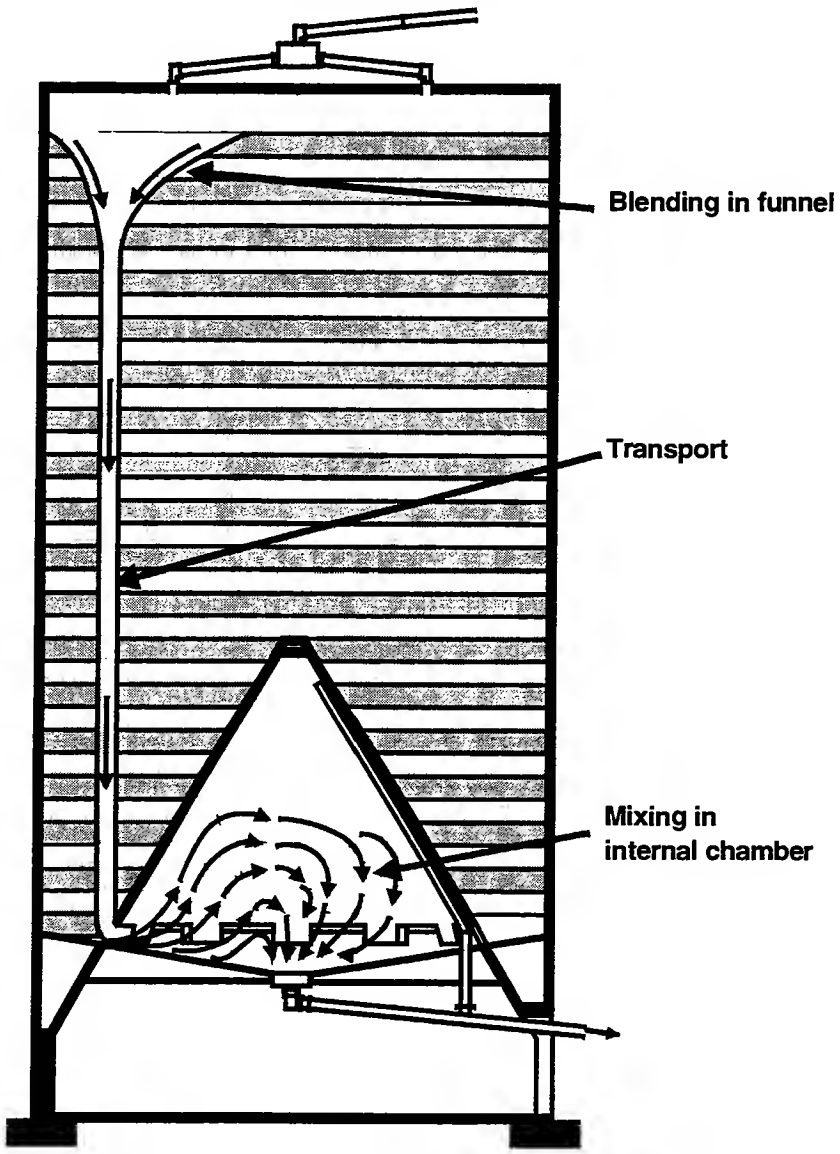
Figure 8: Aerated gravity systems



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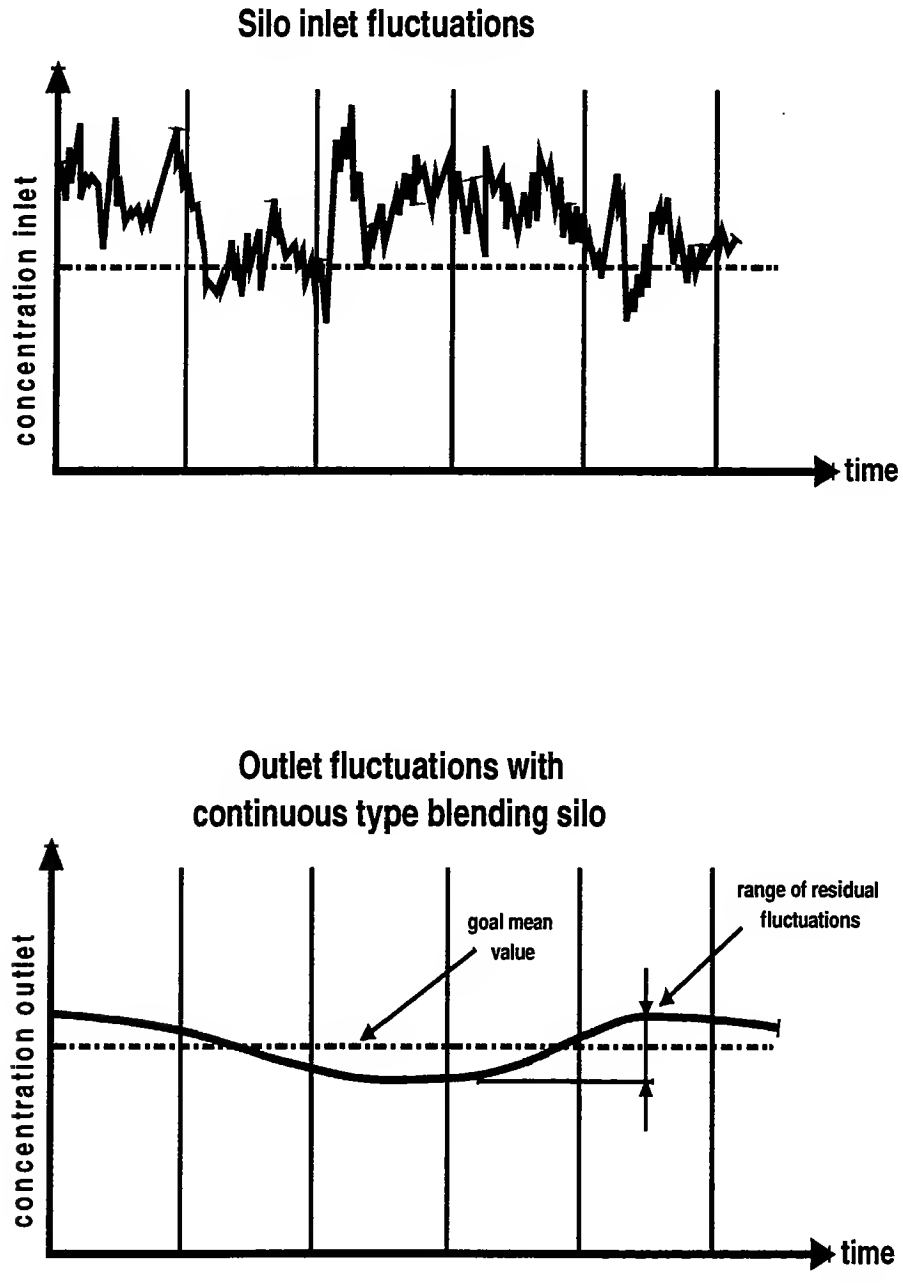
Horizontal layering of the raw meal is achieved while feeding the silo via a spider-type air slide system. Cement raw meal activation for discharge is achieved by slight aeration. For that purpose compressed air is introduced through a permeable media covering the silo bottom. The silo bottom itself is divided into segmented areas, the number of which is a function of the silo diameter. Aeration air is supplied at a low rate and a low pressure into the selected aeration sector for raw meal activation. This air leaves the silo together with the activated raw meal; it will not penetrate into the raw meal column on top of the activated sector. Aeration is switched systematically by means of a special valve sector by sector.

The blending potential of the aerated gravity silos is limited compared with that of air-fluidised homogenising silos (Fig. 13).

Figure 13: Homogenizing / blending effect of raw meal silos



3.2.2 Design Concepts

Aerated gravity silo systems are designed according to three concepts:

- as inverted cone silos or
- as central chamber silos,
- as multiple-outlet silos.

3.2.2.1 The inverted cone concept

The inverted cone silo (Fig. 9.1) represents the pure concept of the aerated gravity silo. The silo is, as said by its name, equipped with a huge inverted cone covering most of its centre bottom area. The remaining annulus is divided into segmented areas that are covered by open airslides. Each sector is equipped with its own outlet. Raw meal is activated predominantly at the silo's circumference by sequential air supply to the individual sectors, avoiding by this the formation of huge zones of stagnant product.

- Blending effect: max. 5:1
- Energy consumption: 0.1-0.2 kWh/t

System suppliers:

- BMH-CPAG
- IBAU
- Fuller-Kovako
- Krupp-Polysius tangential silos

3.2.2.2 The central chamber silo concept

The central chamber silo configuration (Fig. 9.2+3) refers to a concept that uses the inverted cone as a centre chamber for additional reduction of residual compositional short-term variations. The annulus is divided into segmented areas that are covered by open airslides as for the inverted cone configuration. Again raw meal is activated by sequential air supply to the individual sectors. Design of the central chamber differs in shape (conical or cylindrical) and volume. Compressed air is introduced for air-fluidisation through a permeable media (open airslides) covering the chamber bottom that is typically divided into quadrants. The aeration sequence for the central chamber is similar to that of air-fluidised silos as discussed in Para 3.1.

The blending potential of a central chamber silo is slightly better compared to that of a simple inverted cone silo.

- Blending effect: 5:1
- Energy consumption: 0.3 kWh/t

System suppliers:

- BMH-CPAG central chamber silos homogenising chamber silos
- Joh. Möllers

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Figure 9: Continuous blending - type silos

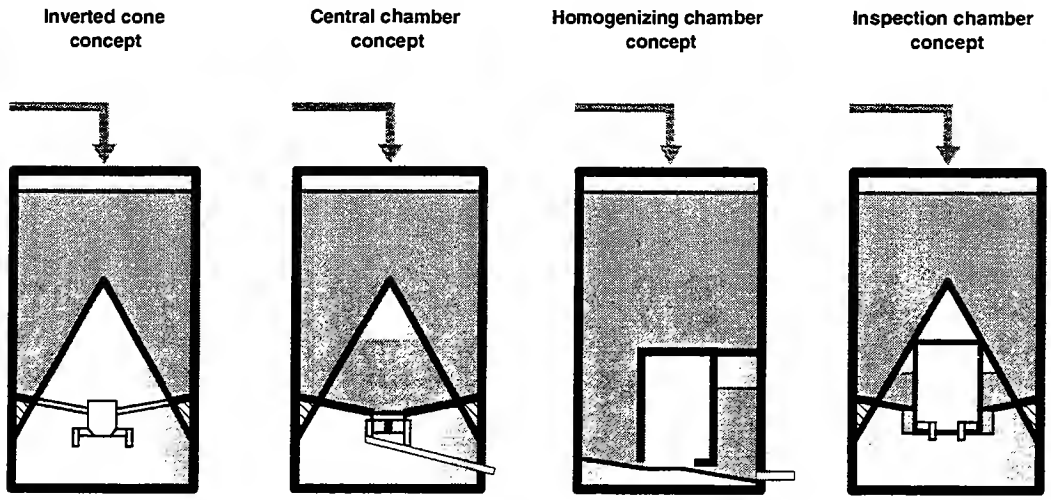

Fig. 9.1 Fig. 9.2 Fig. 9.3 Fig. 9.4

3.2.2.3 The multiple-outlet silo concept

Multiple-outlet type gravity silos follow the concept of blending the raw meal while it is discharged via different outlets at different rates.

- Blending effect: 5:1
- Energy consumption: 0.3 kWh/t

This type of silos is available in various configurations:

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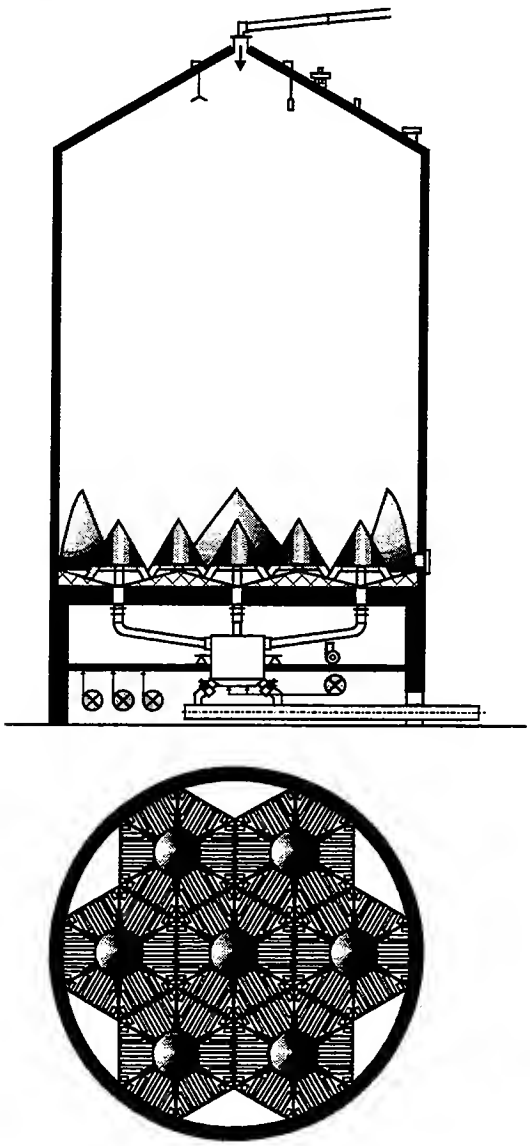
3.2.2.3.1 The FLS' Controlled Flow (CF) silo concept

The silo bottom of the CF silo (Fig.10) is divided into seven identical hexagonal sectors, each of which has its centre outlet covered by a pressure relief cone made of steel. Each of the hexagonal sectors is subdivided into six triangular segments all equipped with open aeration boxes. Raw meal extraction follows a sequence where three segments positioned at three different outlets are aerated at a time. From the outlets it is conveyed at different rates to the central mixing tank installed below the silo. The aeration sequence is cyclic in a way that all the 42 segments will be activated once within about 15 minutes.

Supplier:

- FLS

Figure 10:



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3.2.2.3.2 The Fuller-Kovako Random Flow silo concept

Again the silo bottom of the Random Flow silo is divided into a number of sectors. In addition these sectors are subdivided in six discharge zones and equipped with a closed collecting airslide. Each discharge zone has its pick-up point to the collecting airslide from which the raw meal is transferred to the centre mixing tank installed below the silo. Selective aeration is the means by which raw meal is discharged out of three different silo areas at a time. Again aeration sequence is cyclic.

The concept can easily be applied for upgrading existing fluidised blending silos.

Suppliers:

- Fuller Kovako
- Krupp-Polysius Multiflow silo

3.2.3 General Design Characteristics:

Silo design characteristics		**Continuous Blending Silo**
silo capacity	t	2'000-20'000
height/diameter ratio	-	up to 2.5:1
net aeration area (% of bottom area)	%	25-35(-50)
aeration air rate (specific)	$m^3/minm^2$	1-1.5
aeration air pressure	bar	0.6-0.8
energy consumption (specific)	kWh/t	0.1-0.3
range of fluctuations that can be reduced	h	up to 5
homogenising factor	-	up to 5:1
Internal Chamber		
net aeration area (% of bottom area)	%	35-50
active aeration air rate (specific)	$m^3/minm^2$	1.0-1.5
active aeration air pressure	bar	0.8
aeration air rate (specific)	$m^3/minm^2$	1.0- 1.5
aeration air pressure	bar	0.6-0.8

3.2.4 Valuation of the aerated gravity silo concept

+ applicable for wide capacity ranges
+ low energy consumption
+ low investment
- limited raw meal beneficiation potential compared to the air-fluidised homogenising silos
- insufficient reduction of long-term, peak or step-type fluctuations

A comparison of aerated gravity systems versus air-fluidised systems is given in Fig.12.

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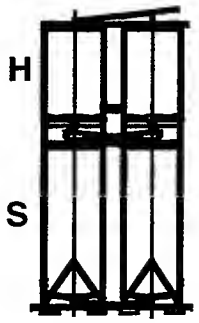
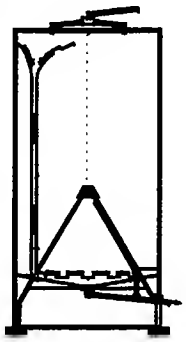
Figure 12: Comparison of silo concepts: Homogenizing versus Blending Silos

Silo configuration	 Two storey		 Single silo
	H	S	
Design Data			
• Capacity	10-12 h	1-3 d	0,5 - 3 d
• Height/diameter ratio	1,2 : 1	2,2 : 1	Up to 2,5 : 1
• Net aeration area (% of bottom area)	Up to 70	25-35	25-35-(50)
Investment cost	100%		50%-60%
Aeration			
• Air rate per net aeration area	1,5 - 2,0m ^{3}/min m ^{2}		1 - 1,5m ^{3}/min m ^{2}
• Air pressure	Up to 2,5 bar		Up 0,6 to 0,8 bar
• Spec energy consuption	0,7 - 1,5 kWh/t		0,1 - 0,3 kWh/t
Performance			
• Range of fluctuations that can be reduced	10-12 h		Up to 4 h
• Homogenizing/blending effect	Up to 15:1		Up to 5:1

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Development of the aerated gravity silo concept started with the introduction of raw material preblending systems in the cement process out of the need to reduce power consumption for cement raw meal homogenisation. The increasing efficiency of such preblending systems went along with a gradual reduction of the size of blending silos. While sizing of blending silos was typically for holding a three days stock in the area 1960 to 1980 it is now as low as for holding a one-day stock.

3.2.5 Limits in blending efficiency

3.2.5.1 The case of insufficient reduction of a peak disturbance

The blending efficiency of a continuous blending silo is commonly given by the ratio of the silo inlet and outlet Standard Deviations for the selected compositional characteristic:

$$BF = \frac{S_{in}}{S_{out}}$$

The diagram given in Fig. 15 shows the example of the LSF fluctuations at silo inlet and how these fluctuations are reduced in function of the time. In the first part of the observed interval the inlet and the outlet fluctuation have the following values:

	Inlet	**Outlet**
mean	96.0	96.0
Standard Deviation	2.11	0.24

The blending effect of the silo calculates to be

$$BF = \frac{S_{in}}{S_{out}} = \frac{2.11}{0.24} = 8.8$$

Unfortunately this efficiency factor do not means that all compositional peaks fed into the silo are reduced by this factor. The second part of the observed interval illustrates the case of a huge peak (LSF=112.0) that has been produced and fed into the silo. Reduction of this peak is much less efficient (LSF=103.5). The corresponding blending effect calculates to

$$BF = \frac{112.0 - 96.0}{103.5 - 96.0} = 2.1$$

Continuous blending silos obviously reduce suddenly appearing compositional peaks much less than more or less stochastic short-term fluctuations.

The example demonstrates that compositional fluctuations of different kind are reduced with different efficiencies. The simple blending efficiency as defined by the ratio of the silo inlet and outlet Standard Deviations gives thus just a very general indication on the silo's blending behaviour. As to get more specific in this respect a more in depth investigation is required.

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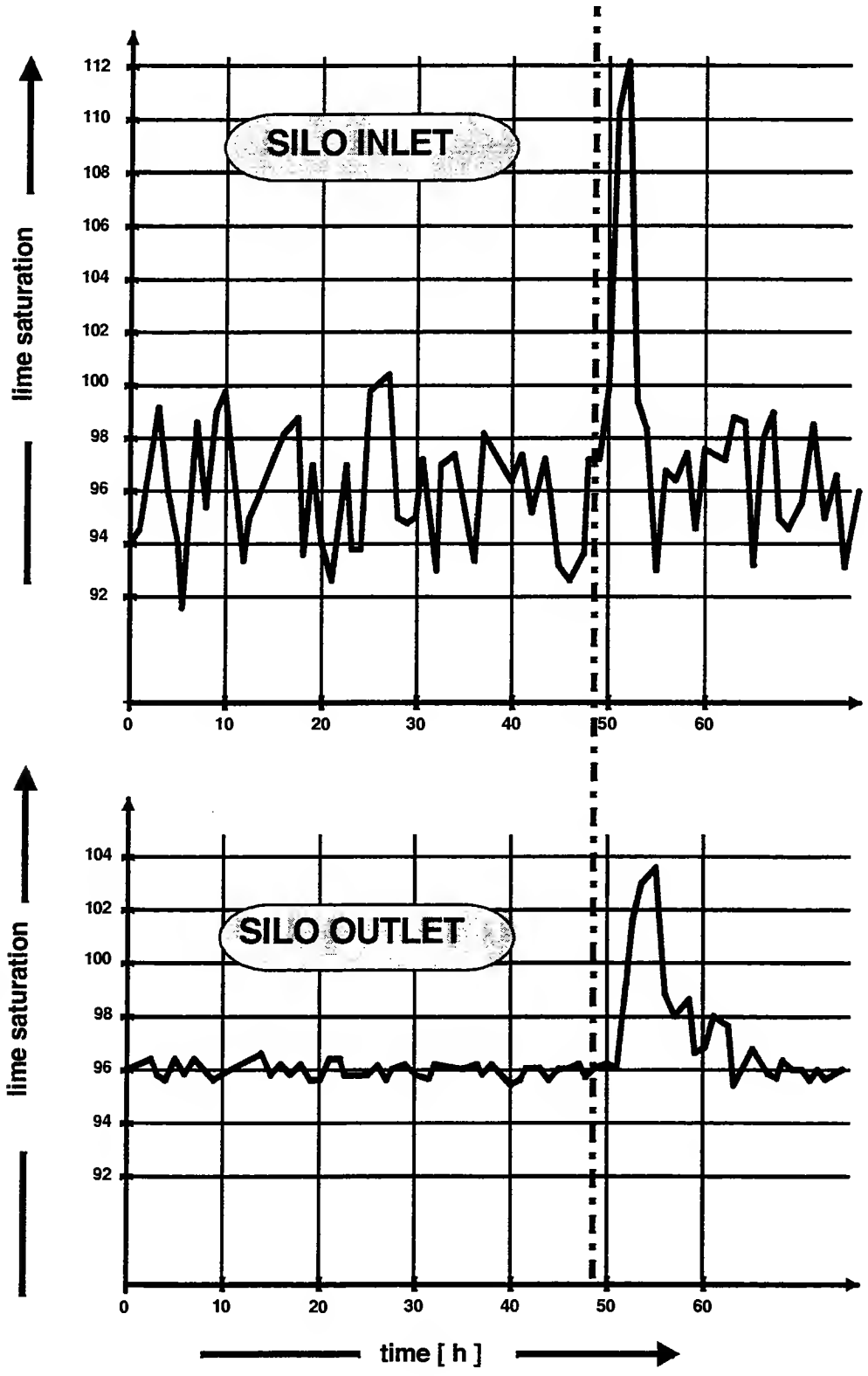
Figure 15: Homogenizing / blending effect of raw meal silos



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3.2.5.2 "Holderbank's" silo investigation

Investigations in a silo's blending behaviour are in that way complex as direct operation of the flow during silo operation is impossible. Again the only information available for evaluation is the information regarding the compositional variations in the silo feed and in the raw meal reclaimed. What we have done already some years ago was to test a silo's response on different types of on purpose created disturbances like

- suddenly appearing peak-type disturbances and
- periodic oscillations.

Suddenly appearing disturbances can mathematically be divided into a sum of single impulse functions. Knowing a silo's response on one inlet impulse function allows for predicting the silo outlet function of any suddenly appearing disturbance. (Fig. 16)

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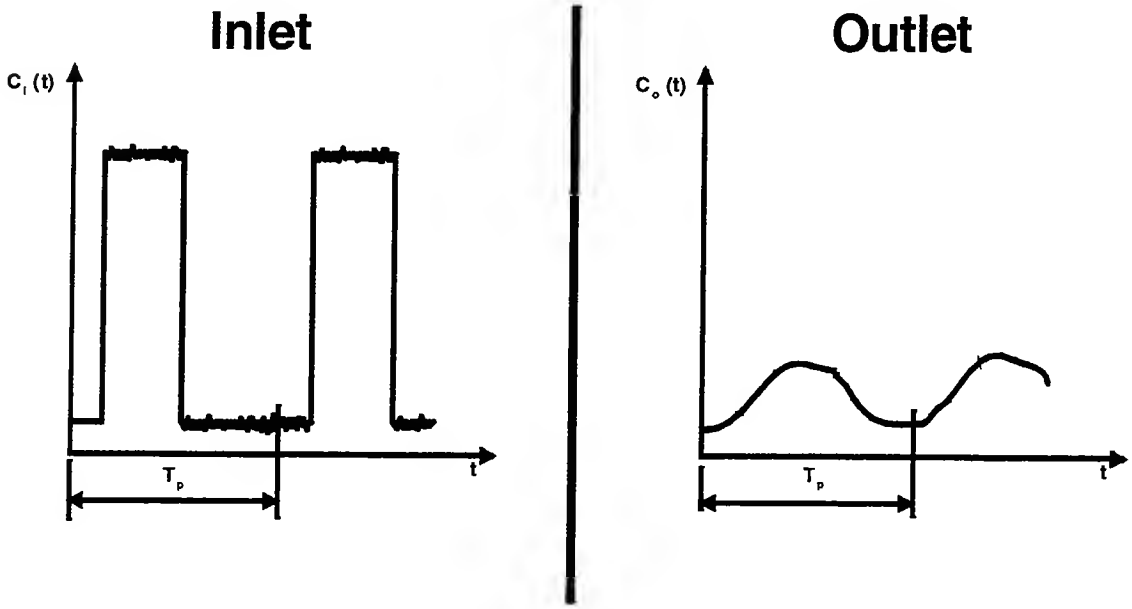
Figure 16: Silo response to a suddently appearing disturbance



$$C(t) = \frac{A_o}{2} + \sum_{n=1}^{\infty} [A_n \bullet \sin(n \bullet \omega_o \bullet t + \varphi_n)]$$

$$\text{with } \omega_o = \frac{2 \bullet \pi}{T_p}$$

n	A_n	φ_n
1	..	..
2	..	..
3	..	..

n	A_n	φ_n
1	..	..
2	..	..
3	..	..

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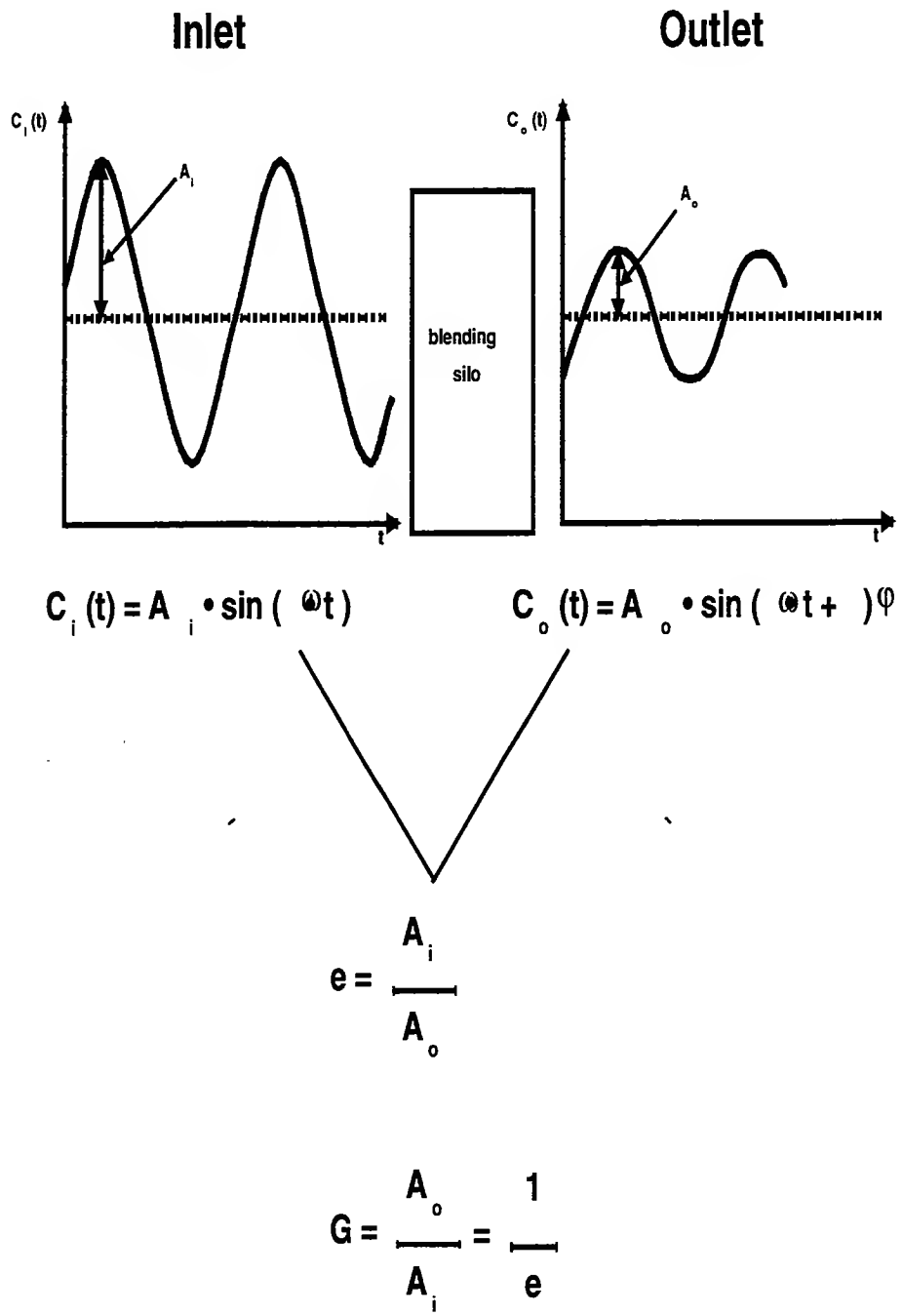
If raw meal's composition out of the raw mill varies like a harmonic oscillation (e.g. like a sinus function) the raw meal exit function (at silo outlet) will also be harmonic with the same frequency but with reduced amplitude and with a phase shift. Real periodic disturbances are not exactly harmonic functions but can according to the law of Fourier be divided into as um of harmonic functions. Again the silo answer to a harmonic inlet function can be calculated from the answer to one single impulse function (Fourier-Laplace Transformation). (Fig. 17)

Figure 17: Silo response to a harmonic oscillation



Hence it follows that with an impulse function as inlet disturbance the behaviour of the silo can be sufficiently investigated and the blending efficiency for all cases predicted. Applying Fourier Transformation on both the silo inlet and outlet functions, for each function a discrete spectrum of amplitudes is obtained (Fig. 18). Just by dividing the two spectra of amplitudes the silo's frequency response G can be calculated. Its bending factor is the reciprocal of G.

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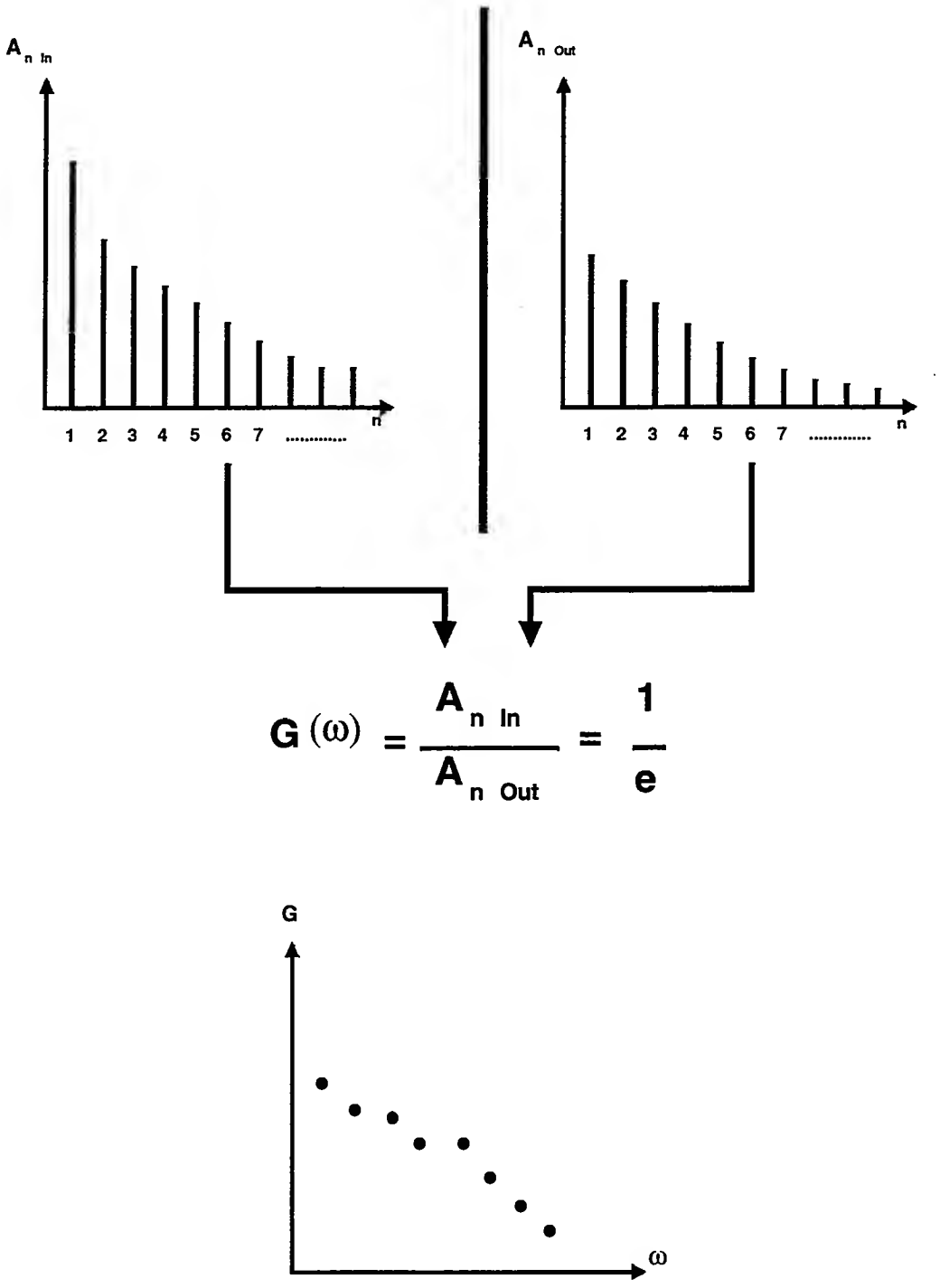
Figure 18: Spectrum of amplitudes - Frequency response



In a further step a mathematical model of second order (Fig. 19) was developed for the simulation of the real behaviour of a continuous blending silo.

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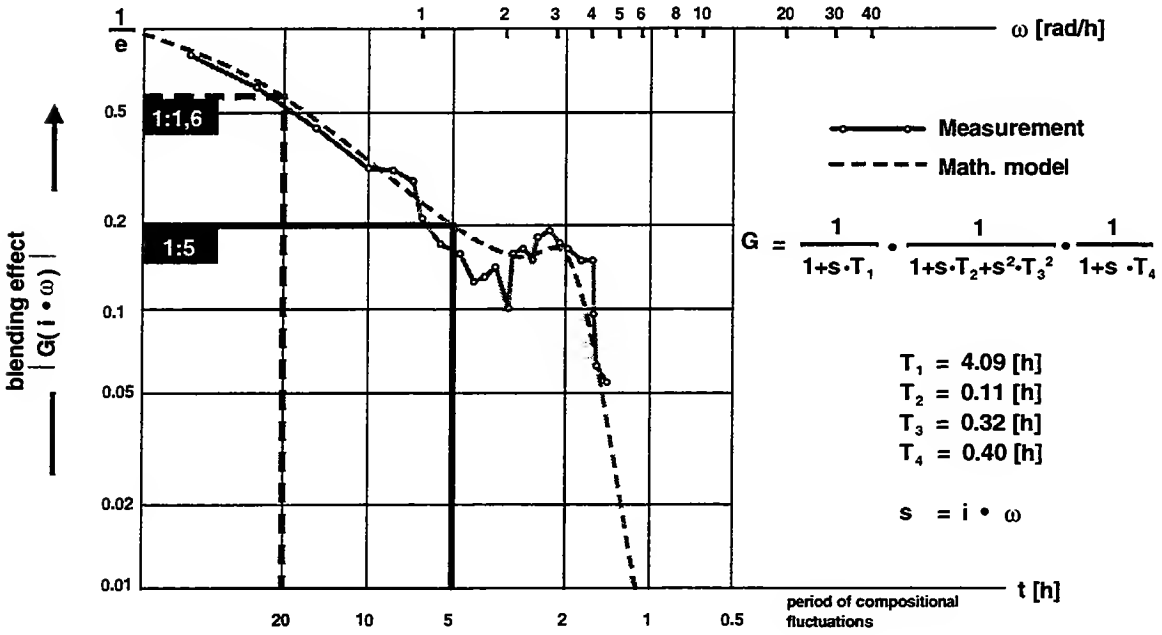
Figure 19:



The investigation showed the following results:

- Most efficient blending takes place at product surface in the silo when a funnel is formed. Feeding raw meal into the silo in horizontal layer is therefore prerequisite for a good blending effect.
- In the transporting channel no significant cross-mixing takes place. This part is of inferior influence on blending efficiency.
- Due to the raw meal's limited residence time in a central chamber contribution of this chamber to the meal beneficiation is limited to very short-term fluctuations.

3.2.5.3 Lessons to learn for silo operation

The diagram in Fig. 19 indicate that all fluctuations with a periodical time shorter than about 5 h are reduced with an efficiency better than 5:1. The first part in the diagram (Fig. 15) represents efficient reduction of such short-time fluctuations.

Long-term fluctuations with periodical times exceeding 5 h are not reduced sufficiently. A compositional oscillation for example with a periodical time of 20 h is reduced with a low efficiency of just 1.16:1. In this case the silo has no chance to achieve a sufficient blending result. The only way to improve the situation is to cut any long-term fluctuations by frequent adjustment of raw mix composition, i.e. by frequent adjustment of the weigh feeder set points.

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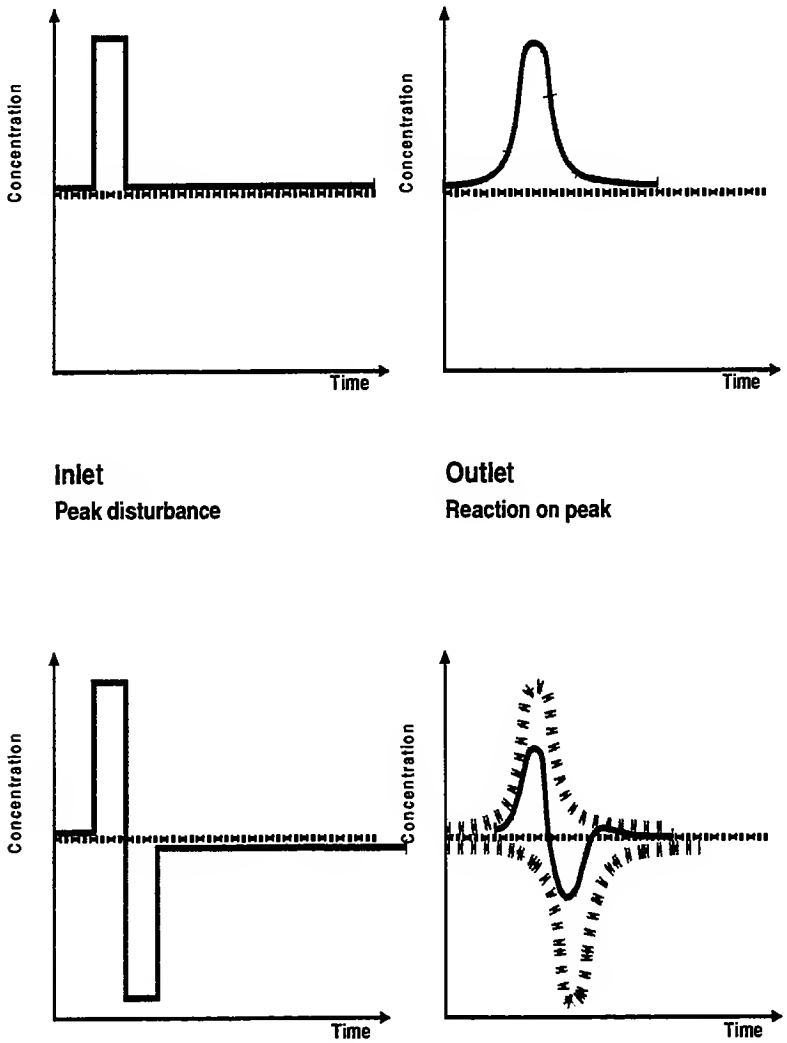
Different to oscillating disturbances are suddenly appearing peaks due to reasons such as

- cut of a new preblending pile,
- loss of one raw mix component,
- inadequate feeder adjustment,
- inadequate kiln dust handling when changing from a compound to a direct operating mode.

Continuous blending silos are not in a position to reduce the peak disturbances sufficiently. The only efficient measure to compensate for such peak disturbances is by creation of a defined counter-peak by making adequate adjustments to the raw mix composition (Fig. 20).

Figure 20: Effect of inadequate raw mix preparation



4. KILN DUST HANDLING

4.1 Compositional characteristics of kiln dust

Switching from a compound operation mode (kiln and raw mill in operation) to a direct operation mode (kiln only in operation with the raw mill stopped) or vice versa may result in an abrupt compositional variation of kiln feed, particularly of its Lime Saturation Factor (LSF). The reason for is that kiln feed and kiln dust are different in their chemical composition. Typical LSF data of some 4 stage preheater kilns are given in below table.

Plant	Kiln No.	LS kiln feed	LS recirculated dust
AL	1	95.7	119.5
AP	1	93.4	110.4
AP	2	93.4	110.1
AT	1	93.2	88.6
EC	3	92.1	81.3
GM	4	95.3	97.3
HD	2	96.1	93.3
KA	6	119.7	87.4
MI	3	93.9	97.4
OZ	4 + 5	93.1	93.9
PL	1	90.8	70.8
RE	3	93.1	103.9
RK	1	91.3	93.0
UV	3	90.2	81.3

As can be seen from above data LSF of kiln dust may differ from LSF of kiln feed quite significantly (in a ± 20 % range). There is no general rule allowing for predicting the LSF of kiln dust based on the compositional data of kiln feed.

A ± 1 % standard deviation of LSF in kiln feed is tolerated regarding stability of kiln operation. Abrupt changes in kiln feed LSF may in principle be compensated by adapting the burning conditions in the kiln. Nevertheless, all effort is made to keep the burning conditions unchanged

- as each variation result in unstable coating conditions in the kiln,
- as clinker quality may temporarily deteriorate due to insufficient control of free lime.

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4.2 Conceptual set-up of Kiln Dust handling systems

LSF variation in case of switching from compound to direct kiln operation also depend on the set-up of the kiln feed system and the manner in which this system is operated by the works personnel.

There is no problem in adding kiln dust to the raw meal when operating kiln and raw mill in a compound operation mode. Arrangement of the sampling station should be such that the compound material flow is sampled.

When switching from a compound to a direct operation mode or vice versa gas routing is adapted to the new operating conditions by the kiln operator but often not kiln dust routing. With the kiln in a direct operation mode kiln dust should never be fed into a raw meal silo. Neither the air-fluidised homogenising silo nor the continuous blending silo is fit to deal with kiln dust out of extended periods of direct kiln operation.The batch actually prepared in case of an air-fluidised homogenising silo will deteriorate under such conditions within a short period of time to an extent that its composition may not be corrected anymore. An important top-layer of kiln dust in an aerated gravity silo deteriorates its beneficiation effect as blending predominately takes place at product surface in such a silo.

Three conceptual set-ups have been developed for reasonable kiln dust handling:

- the separate kiln dust silo concept,
- the silo by-pass concept,
- the kiln dust dilution concept.

4.2.1 The separate kiln dust silo concept

In compound operation all the exhaust gases from the kiln pass through the raw mill. All the kiln dust is mixed with the raw mix fed to the mill. This mix is separated in the kiln dust collector and fed into the homogenising/blending silo. In addition a small portion of kiln dust out of the separate kiln dust silo is added to the raw meal ex mill prior to be fed into the homogenising/blending silo.

In direct operation the kiln exit gases are diverted directly to the kiln dust collector by-passing the raw mill (Fig. 21). At first the kiln dust separated by the dust collector is stored in a separate silo from which it is then added at a low rate to the raw meal reclaimed from the storage/blending silo. It is obvious that by doing so the kiln feed LSF is subject to a change.

When changing from a compound to a direct operation mode the concept allows for limiting the kiln feed LSF to an acceptably small variation provided the kiln dust recirculation rate is sufficiently low.

Valuation:

+ allow for keeping the compositional variations in a narrow range
- Expensive solution

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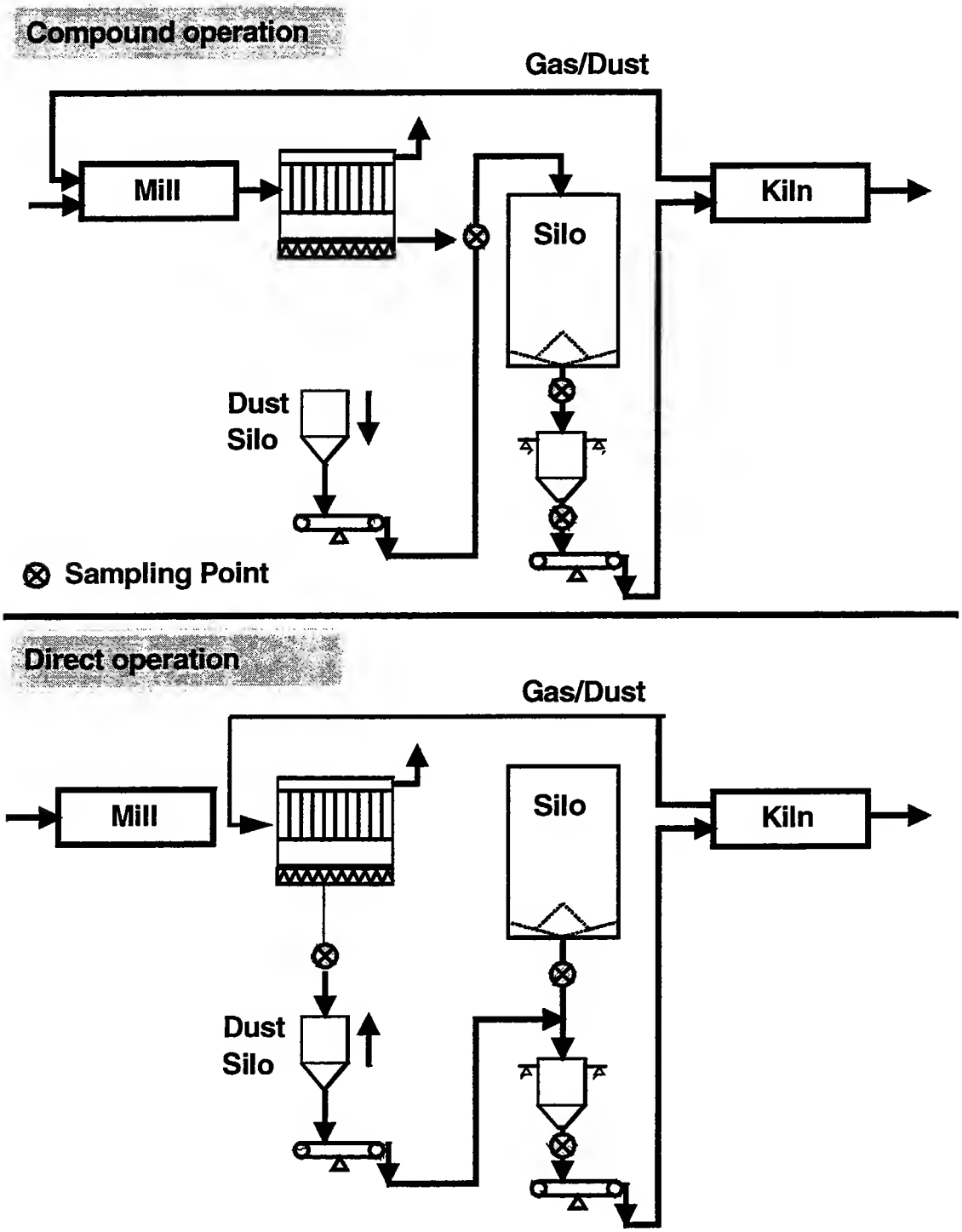
Figure 21: Kiln dust handling - By-Pass with separate dust silo



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4.2.2 The silo by-pass concept

In compound operation again all the exhaust gases from the kiln pass through the raw mill and the kiln dust collector. The raw mix separated in the dust collector is then fed into the homogenising/blending silo out of which the kiln is fed.

In direct operation the kiln exit gases are diverted directly to the kiln dust collector by-passing the raw mill (Fig. 22). The kiln dust separated by the dust collector is mixed with meal reclaimed from the blending silo for being fed to the kiln.

When changing from a compound to a direct operation mode it is obvious the kiln feed LSF is subject to a change. This change may be significant and cause operational problems. But provided the system is correctly operated the meal content of the homogenising/blending silo is not subject to a compositional change. For a numerical example see Annex 2.

Valuation:

+ simple inexpensive arrangement, standard arrangement
- may in rare case result in a compositional change that can cause problems with kiln operation

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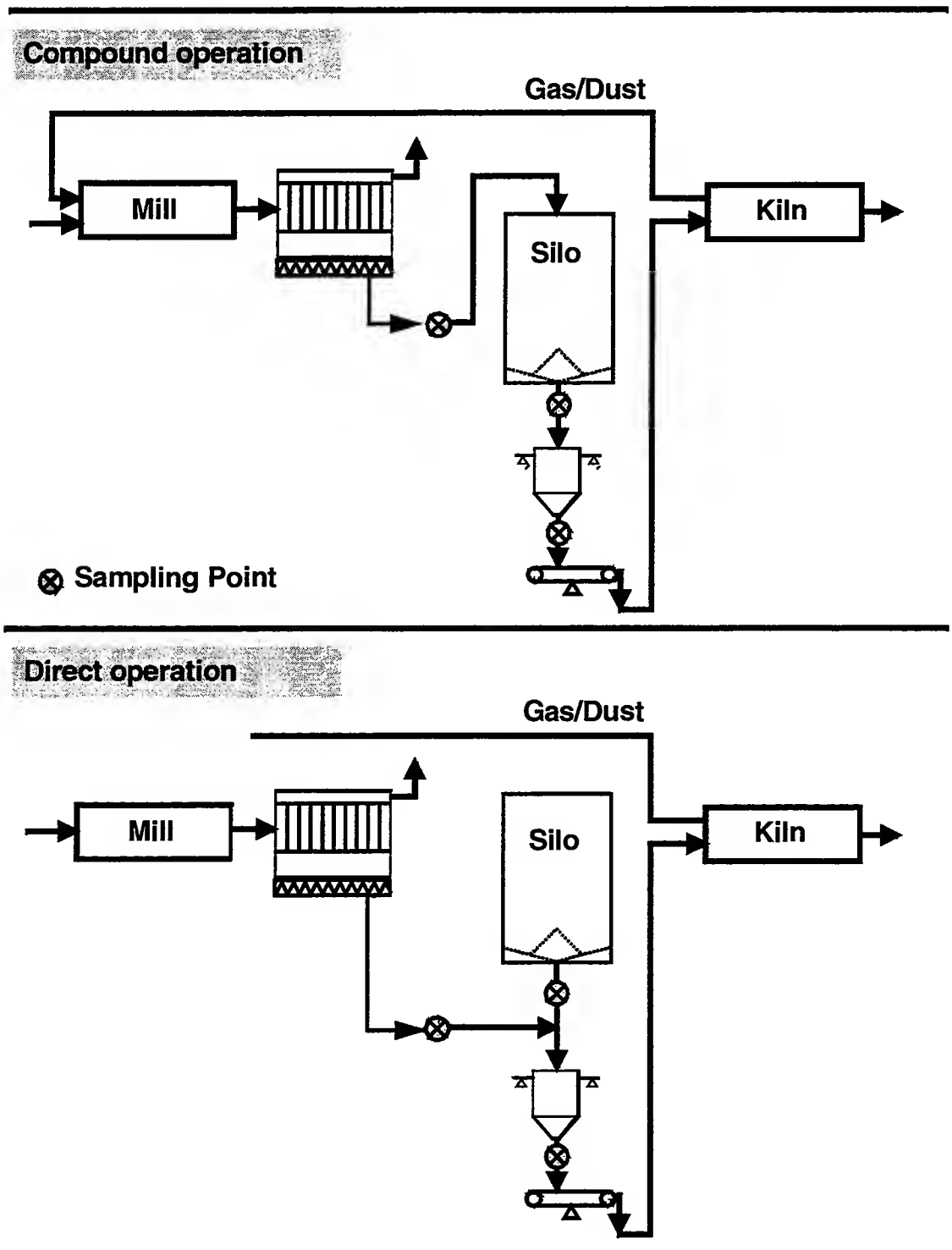
Figure 22: Kiln dust handling - By-Pass without separate dust silo



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4.2.3 The kiln dust dilution concept

Compound operation is similar as for the silo by-pass concept.

In direct operation the kiln exit gases are diverted directly to the kiln dust collector by-passing the raw mill (Fig. 23). A second silo outlet is activated The kiln dust separated by the dust collector is mixed with meal reclaimed via an additionally activated outlet from the blending silo for dilution and recirculation to the blending silo.

When changing from a compound to a direct operation mode kiln feed LSF will gradually change as will the raw meal contained in the blending silo.

Valuation:

+ simple arrangement

+/- result in a gradually changing composition of the hold raw meal stock

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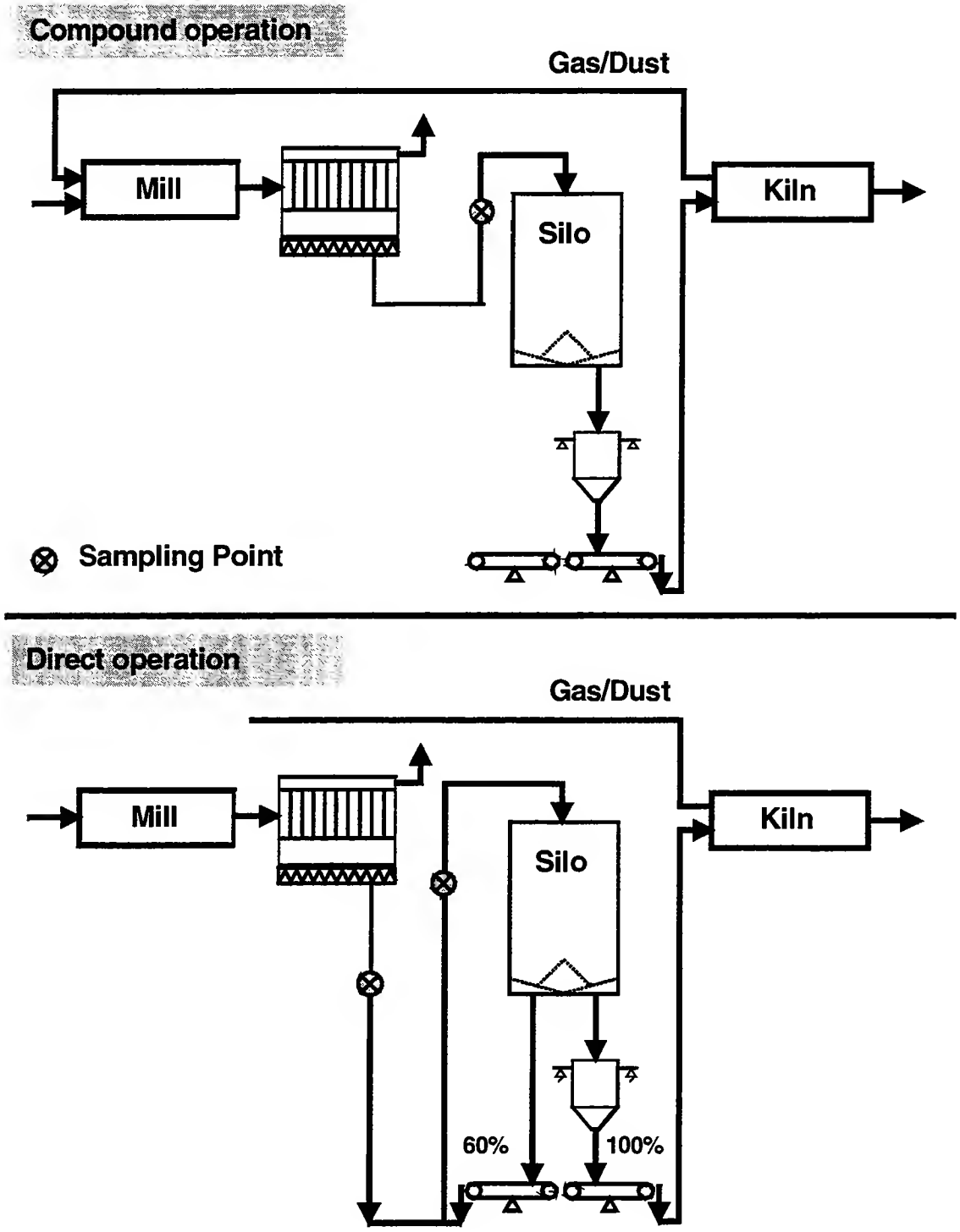
Figure 23: Kiln dust handling - Dust Dilution with Raw Meal



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5. ANNEX

5.1 Annex 1

Testing the Blending Factor BF of Blending/Homogenising Silos

Duration:	2 x 24 h
Permissible interruptions:	3 interruptions, but max. 90 min per test
Sampling the silo feed product:	double spot samples (2 x 100 g) once an hour
Sampling the silo outlet product:	double spot samples (2 x 100 g) once an hour
Analysis:	CaO by XRF

Test Evaluation:

- Blending Factor

$$BF = \frac{S_{in,corr}}{S_{out,corr}} = \frac{\sqrt{S^2_{in,measured} - S^2_{in,error}}}{\sqrt{S_{out,measured} - S^2_{out,error}}} \quad (1)$$ as blending/homogenising factor

- Correction for the sampling and analysis error acc. to Merks double sampling method

with

$$S^2_{measured} = \sum_{i=1}^{N} \frac{(X_i - \overline{X})^2}{N-1} \quad (2)$$ as variance of the measured values

$$X_i = \frac{1}{2}(X_{i,1} + X_{i,2}) \quad (3)$$ as mean concentration of a double sample

$$\overline{X} = \frac{1}{N}\sum_{i=1}^{N} X_i \quad (4)$$ as mean concentration of all double samples

$$S^2_{error} = \frac{\pi}{4}\left(\frac{1}{N}\sum_{1=i}^{N}|d_i|\right)^2 \quad (5)$$ as error of sampling and/or analysis (acc Merks)

$$d_i = (X_{i,1} - X_{i,2}) \quad (6)$$ as difference in concentration for one double sample

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5.2 Annex 2

Effect of kiln dust on kiln feed composition

Assumptions:

average LSF of the clinker (goal value)	95	%
LSF of kiln dust	110	%
external dust circuit (LOI free)	0.1	kg/kg_{cli}
operating time of mill (5 d/w, 20 h/d)	59.5	%

Calculation:

- specific mill performance (LOI free)

$$\dot{m} = \frac{100\%}{59.5\%} = 1.68 kg / kg_{cli}$$

- LSF of raw mix ex mill or kiln feed respectively (compound operation)

$$LSF_{CO} = \frac{1.98 \times 95\% + 0.1 \times 110\%}{1.98 + 0.1} = 95.8\%$$

- LSF difference on changing from compound to direct operation

$$\Delta LSF = LSF_{DO} - LSF_{CO} = (\dot{m}_{KD} \times LSF_{KD} + (1 - \dot{m}_{KD}) LSF_{RM}) - 1 \times LSF_{RM}$$
$$\Delta LSF = \dot{m}_{KD}(LSF_{KD} - LSF_{RM})$$
$$\Delta LSF = 0.1(110\% - 95.8\%) = 1.4\%$$

with:	LSF_{DO}	= Lime Saturation direct operation	%
	LSF_{CO}	= Lime Saturation compound operation	%
	LSF_{KD}	= Lime Saturation kiln dust	%
	LSF_{RM}	= Lime Saturation raw meal	%
	m_{KD}	=specific kiln dust rate (LOI free)	kg/kg_{cli}

- LSF of kiln feed (direct operation)

$$LSF_{CO} = \frac{1 \times 95.8\% + 0.1 \times 110\%}{1 + 0.1} = 97.1\%$$

- LSF in clinker during compound operation

$$LSF_{cli,CO} = 95.8\% - 1.4\% = 94.4\%$$

- LSF in clinker during direct operation

$$LSF_{cli,DO} = 95.8\%$$

The LSF of the clinker and the raw meal ex silo will be identical provided that the dust produced by the kiln is immediately returned to the kiln at the same quantity.

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5.3 Annex 3

Operation of Raw Meal Homogenising and Blending Silos

The efficiency of a homogenising/blending silo system is mainly impaired by

- insufficient working condition of the silo system,
- insufficient operation of the silo system,
- insufficient raw mix composition control.

1 Condition of the Silo

Problem: lump formation due to water ingress into the silo

Remedies:
- empty and clean the silo completely
- check silo roof and wall with regard to leaks, seal the leaks and coat the silo wall

2 Raw meal distribution

Problem: non-uniform distribution of raw meal in continuous blending silos (Fig. 24)

Remedies:
- install a guide plate in distributor box
- check installation (level) of distributor box
- check whether the feed spout is arranged in the centre of the distributor

3 Raw meal reclaim

Problem: raw meal lumps obstruct material flow from the silo

Remedies:
- check and clean outlet boxes, cut-off and flow control gates with regard to foreign matters, tramp iron, hard product lumps
- in case that product lumps check the possibility of water ingress (acc. Para 5.1.1) or install a lump breaker
- install dryer for the aeration air

4 Aeration air supply

4.1 Problem: reduced air supply to the aeration system

Remedies:
- check air aspiration filter on permeability and clean the filter
- check air distribution system on permeability and clean the system
- check pressure distribution in air distribution system

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4.2	Problem:	continuous blow-off of blower safety valve
	Reason:	inadequate tuning of blower capacity to silo operating conditions
	Remedies:	• relief blower safety valve by a reduction of its differential pressure • Measure volume of excess air, reduce air rate by this volume by a proportional reduction of the blower speed (exchange of pulleys)
4.3	Problem:	loss of aeration air
	Remedies:	check external air distribution systems with regard to leaks • empty and clean the silo completely as to check the aeration system on leaks
5	**Aeration air distribution**	
5.1	Problem:	the specific aeration air rate (m^3/m^2min) should be constant and not function of the silo diameter (Fig. 25.1)
	Remedies:	check opening of manual valves
5.2	Problem:	faulty operation of air distribution valves resulting more than one activated sector at a time (Fig. 25.2)
	Remedies:	check operation of air distribution valves
5.3	Problem:	asynchronous aeration sequence (Fig. 25.3)
	Remedies:	check setting of air distribution valves

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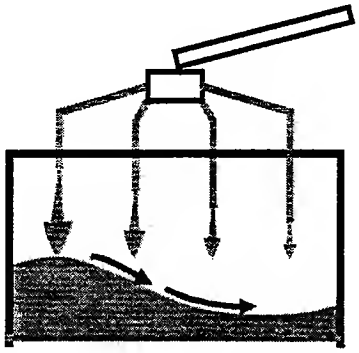
Figure 24: Continuous Blending silo

Problem

Segregation due to

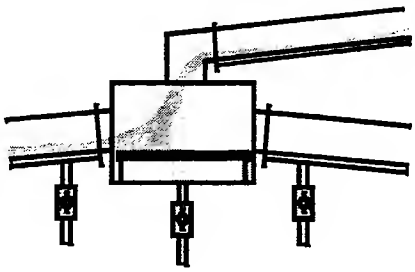
Not uniform

Raw meal distribution

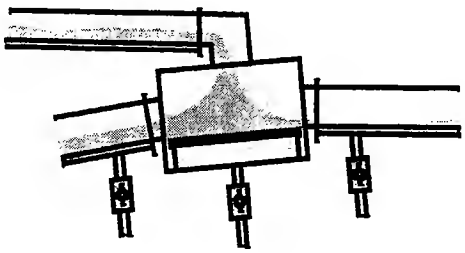


Reasons

Reduced feed rate



Unleveld distributor box



Excentric feed point

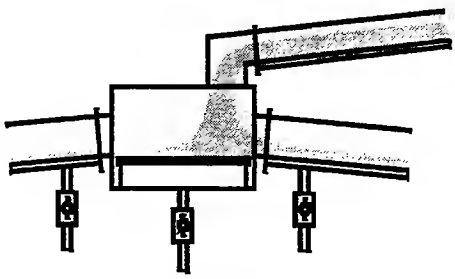
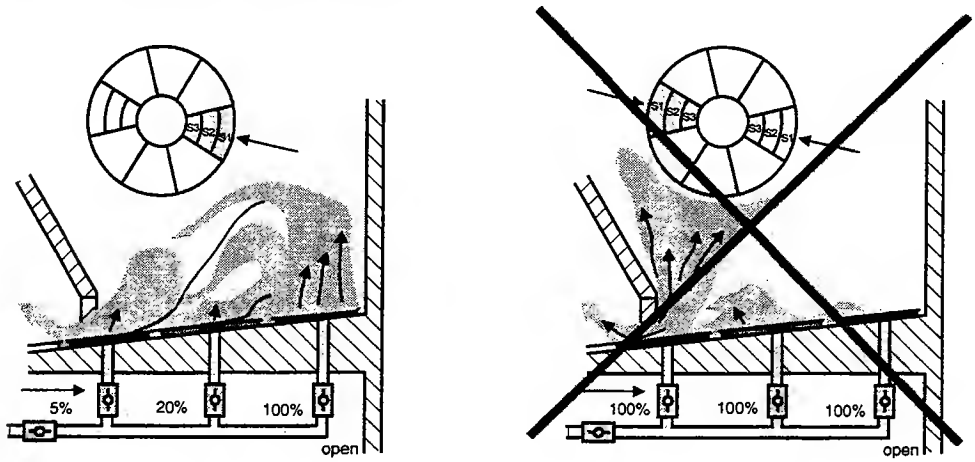


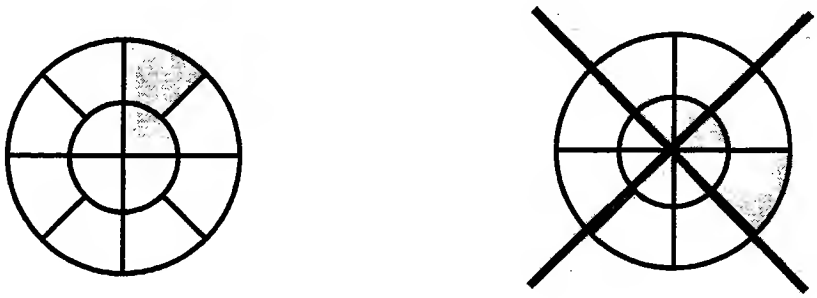
Figure 25: Continuous Blending Silos - Problems with aeration air distribution

1. Not uniform air distribution



2. Faulty operation of air distributor valves

3. Asynchronous aeration sequence



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Chapter 6

Comminution Engineering

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Comminution Engineering

M. Rüdlinger
VA 89/5589/E

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Summary

Comminution engineering describes the processes for the size reduction of bulk materials. The materials are crushed in one or several steps to the feed size required for the grinding process. In the grinding systems of a cement manufacturing plant the materials are ground to either raw meal or cement.

Crushing is characterized by the application of either compression or impact forces whereas grinding is effected with a combination of impact and friction forces.

Comminution theories have been developed in an attempt to determine the energy input required to crush or to grind a material of a given feed size to the required product fineness. These theories are only valid for a rather limited range of materials and of size-reduction-machine types and are therefore not suitable for the dimensioning of industrial machines.

Based on comminution theories and empirical findings several laboratory tests have been developed to measure the energy input required for a given size reduction.

The ground or crushed product can be characterized by its particle size distribution (PSD) and/or by the new surface produced in the comminution process.

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1. INTRODUCTION

Comminution engineering, also known as size-reduction engineering, is defined as the designing and analyzing of processes for the breakdown of bulk solids into smaller / finer particles, without changing their state of aggregation.

Comminution is a major unit operation in a cement manufacturing plant, accounting for about 50 % - 75 % of plant electrical energy consumption.

The comminution processes are divided, depending on feed-material size, into the crushing (breaking) processes and the grinding (pulverizing) processes.

In the following pages especially the processes, theories and laboratory tests relevant for grinding are dealt with. The crushing aspects will only briefly be mentioned.

2. COMMINUTION PROCESSES

The comminution processes in a cement production line are divided into four main groups depending on kind of material to be processed and feed material size. The four main comminution processes are:

1) Crushing of rock materials from the quarry (size 1 - 1.5 m) in one till three steps to an intermediate raw material state (size < 100 mm), used as input for the raw meal production process.
2) Grinding of the intermediate raw material to raw meal (< 2 % 12 200 µm), which degree of fineness is defined by the following kiln-firing process used.
3) Grinding (and simultaneously mixing) of clinker and additives (size < 25 mm) to the final product, cement, which degree of fineness is defined either by standards or customer requirements.
4) Grinding of raw coal (size 20 mm) to coal powder used for kiln firing.

It is obvious that the great difference between these four comminution processes will affect the electrical energy consumption, machinery design, machinery wear and process heat consumption.

The following approximate rates of specific power consumption per ton cement production shows the distribution of energy between individual comminution processes and gives an indication on where energy can be saved.

Raw material crushing:	1 - 2 kWh/t clinker
Raw material grinding:	5 - 15 kWh/t clinker
Coal grinding:	2 - 4 kWh/t clinker
Cement grinding:	30 - 60 kWh/t clinker
Cement plant total:	**90 - 130 kWh/t clinker**

A lot of attempts have been made to obtain a thorough knowledge about the energy required for comminution of a known material bed consisting of determined product particle size distribution (PSD), but till now nobody has been successful in stating a valid theory.

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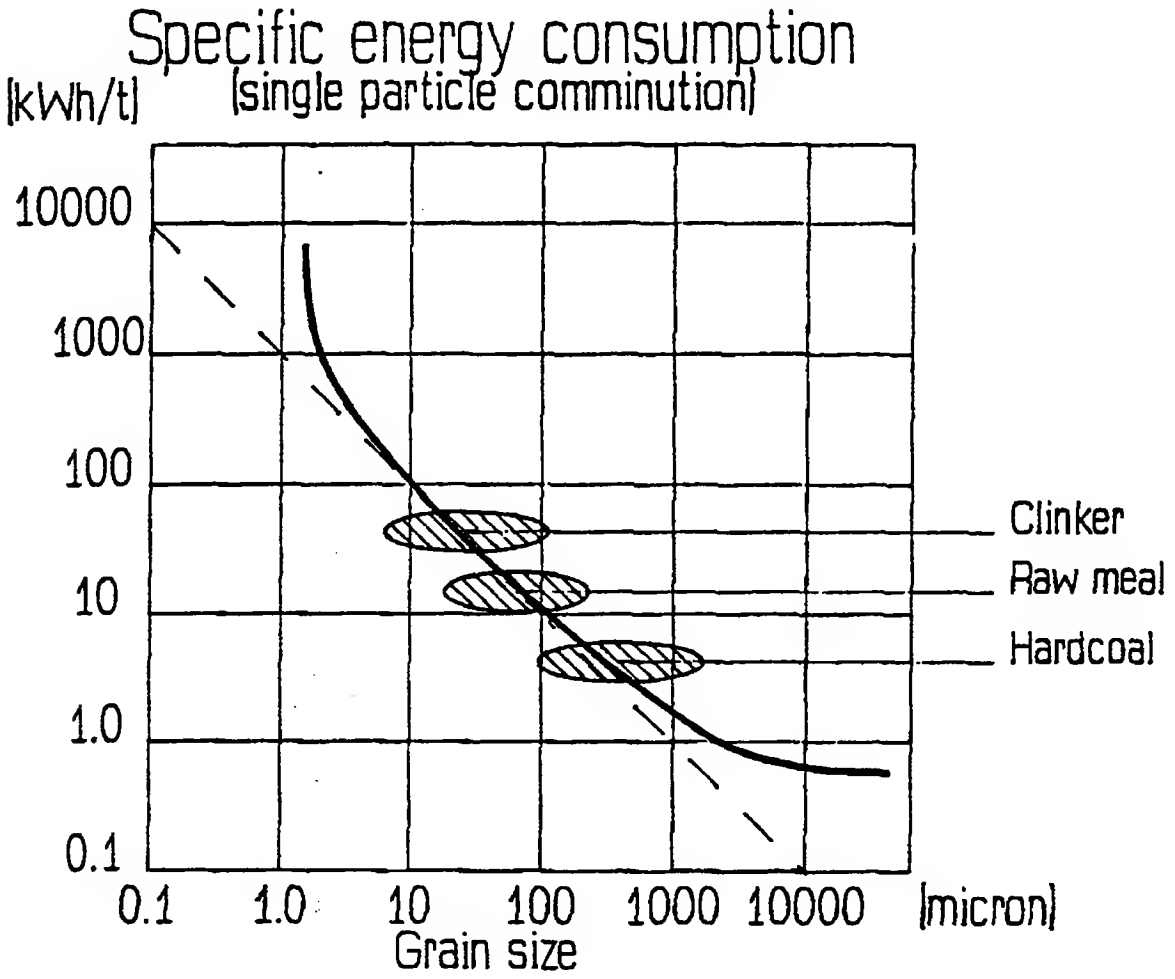

3. COMMINUTION METHODS

The most known comminution forces are:

1) Compression forces
2) Friction / Shear forces
3) Inter-particle contact forces
4) Impact forces

A crushing process is often characterized by the application of a single type of comminution force, whereas a grinding process is mostly characterized by the application of a combination of above comminution forces, e.g. compression - friction - shear (roller mill) or impact-friction (ball mill).

An important fact is that impact comminution processes tend to produce a greater amount of fine particles than a compression comminution process.

Note that at time there are no evidences, that the feed particle shape affects a grinding process. This is not the case for crushing processes.

Some comminution forces and machines are shown in fig. 1.

The selection of the adequate grinding machinery / method principally depends on three types of parameters; material, process and environmental parameters.

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Material parameters:

* Chemical composition (SiO_2)
* Mineralogical composition (Moisture)
* Physical properties (Strength, Abrasiveness)
* Possible time-dependent variations of above parameters

Process parameters:

* Comminution-, drying-, feeding- and wear behavior of input material
* Requirements on fineness, homogeneity of output material for further processing.

Environmental parameters:

* Life cycle costs (investment, operation, maintenance)
* Specific energy consumption (kWh/ton throughput)
* Noise level, space requirements

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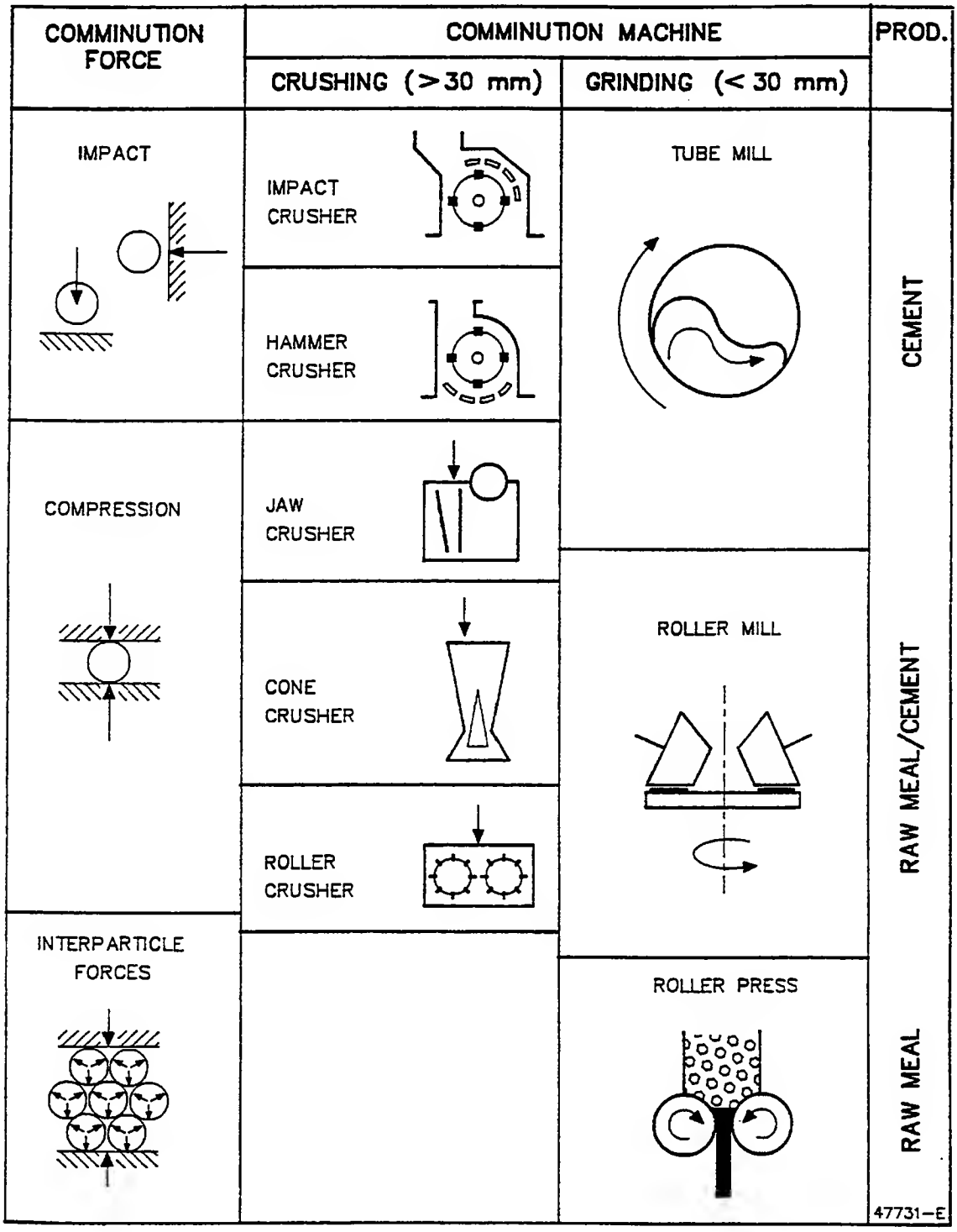
Figure 1



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4. COMMINUTION THEORIES

Comminution theories are trying to explain / describe the relationship between process energy input and change of material PSD. Only a small portion of the energy input in a comminution process is converted into actual comminution work. Main reasons for the poor energy utilization are:

- Mechanical losses (gears, transmission, etc.) 5 % - 10 %
- Material handling, drying and de-dusting 80 % - 95 %
- Noise emission
- Poor distribution of comminution forces to every single particle in particle bed

4.1 Energy

As mentioned earlier the energy requirements for a comminution process varies over a wide range.
To give an idea about how and why the energy consumption varies so much, some 'laws of comminution' are presented. These are all based on the relationship between the changes in particle size and energy consumption.

In 1967 RITTINGER presented the so called 'First law of comminution', which postulates that the new surface created by the comminution process is directly proportional to the energy input. This means that the energy input, E_R, is inversely proportional to the decrease in particle size.

$$W_R \sim \frac{1}{X_2} - \frac{1}{X_1}$$

In 1885 KICK presented the 'Second law of comminution', which in contrast to Rittingers 'law', postulates that the energy input is proportional to the particle volume reduction during the comminution process.

$$W_k \sim \ln\left(\frac{X_1}{X_2}\right)$$

In 1961 BOND presented the 'Third law of comminution' postulating that the energy input is proportional to the new crack length created by the comminution process. This gives that for particles of a similar shape the energy input is inversely proportional to the square root of the average change in PSD.

$$W_B = E_B\left[\frac{10}{\sqrt{X_{80,P}}} - \frac{10}{\sqrt{X_{80,P}}}\right]$$

E_B = BOND's workability index
X_{80} = 80 % passing [µm]
W_B = specific comminution energy [kWh/t]

The Bond workability index, E_B (or W_i) is derived from the Bond grindability test, which simulates, on a laboratory scale, closed circuit grinding. When using the Bond index, E_B, in above specific energy formula, it should be kept in mind that a certain index value, E_B, can only be used with the same material and with a size reduction range close to the one used when the grindability test was carried out.
Note that the above described 'laws' are no laws, but approximations and/or empirical findings.

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4.2 Energy Utilization

The energy utilization, E_u, is the quotient of the new created specific surface and the specific energy used hereof.

$$E_U = \frac{\Delta S}{W_S \cdot 3{,}6}[cm/J]$$

$\Delta S = SSB_{Prod} - SSB_{Feed}$ [cm^2/g]

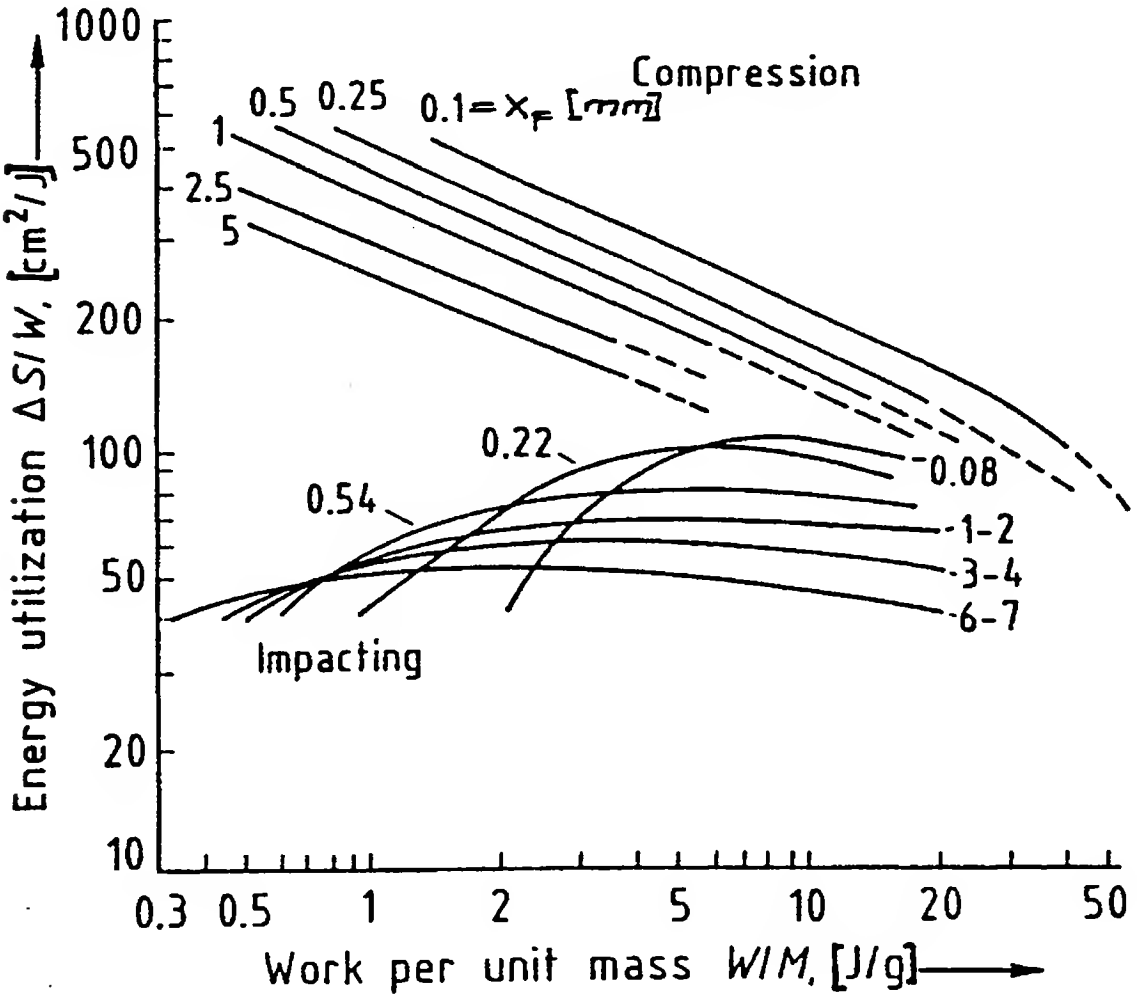
W_S = specific energy [kWh/t]

The energy utilisation value can only be used to compare comminution process, where material with the same chemical/mineralogical composition and feed size is used.

Note the specific surface value used in above formulas is the SSB (according to Blaine). Since there are al lot of different methods to obtain the specific surface value, the method used in each case must be specified. If the specific surface value is obtained through calculation, SSC, using the powder's PSD, the following conversion formula can be used:

$$SSB = \frac{SSC - 807}{12}; SSC = 807 + 1{,}2SSB$$

Figure 2 Energy Utilization, Limestone (Single Particle Comminution)



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4.3 Efficiency

The comminution efficiency, $_c$, is defined as the relation between 'the theoretical energy input' and 'the actually process input energy'.

$\eta_c = E_i / E_p$

where

E_i = theoretical energy input
E_p = actual energy input

The theoretical energy input is defined as 'the least energy input required to over-come a single particles strength and break it into smaller pieces' and the actual process energy input is defined as 'the mill shaft energy input'.

The comminution efficiency depends, apart from material properties, strongly on the material PSD, on type of comminution machine and on operational state. Note, that the theoretically calculated input energy is more representative for coarse crushing than for fine grinding, as shown below.

Based on investigations on single particle comminution, STARHAND indicated the following ranges of comminution efficiency for some industrial comminution machinery.

Jaw and Roller crusher	70 - 90 %
Impact crusher	30 - 40 %
Hammer crusher	15 - 25 %
Roller press	10 - 20 % F
Vertical roller mill	7 - 15 %
Tube mill	6 - 9 %

5. PARTICLE SIZE DISTRIBUTION

The Particle Size Distribution (PSD) is one of the most important parameters affecting a comminution process. This is why it is important to depict a PSD and characterize it by means of some significant values in the cement industry.

In the cement manufacturing industry the following ways of describing a PSD are commonly used.

For coarse materials the PSD is usually plotted in a LIN/LIN or LIN/LOG - diagram, as shown above. Significant values for such a PSD are the cumulative percentage of residue on a standard sieves or the sieve size at a given percentage of residue (R 10 %, R 5 %, R 25 mm, etc.).

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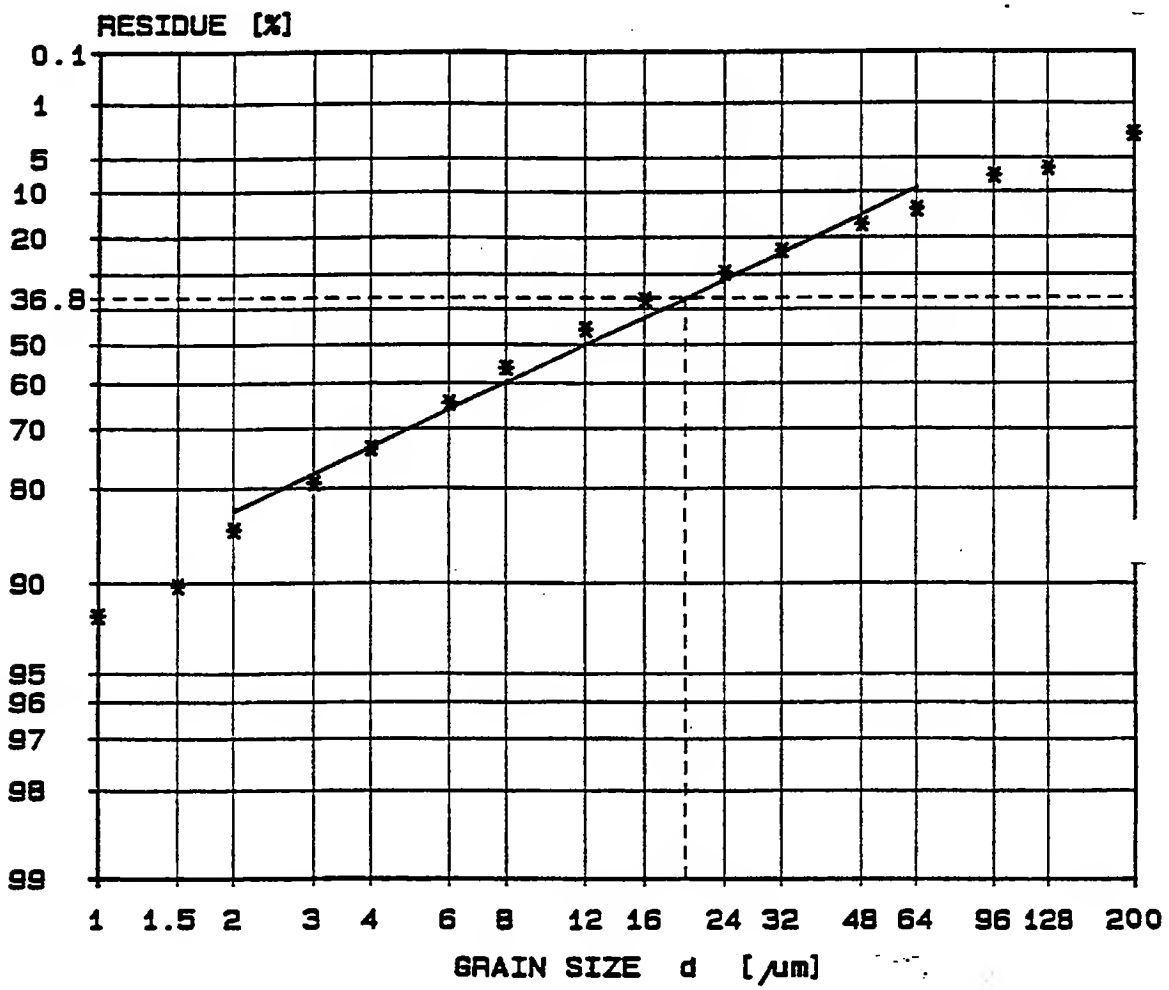

GRAIN SIZE DISTRIBUTION ACCORDING ROSIN-RAMMLER-SPERLING

SCREENING METHOD: LASER GRANULOMETRE 715 CILAS

SLOPE n : 0.74[-]
GRAIN SIZE d' : 19.6 [µm]

For very fine materials, like cement products, the PSD is plotted in a so called RRSB-grid 1), because herein a fine PSD tend to correlate with a straight line. This type of diagram has a double logarithmic ordinate (y-axis) and a logarithmic abscissa (x-axis). After linear regression of the PSD, the slope, n, of the straight line can be calculated. This slope, together with the particle size, d', at 63.2 % cumulative passing (36.8 % residue) are significant values for a PSD, stating its location (µm) and degree of distribution (wide-narrow).

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The RRSB - formula

$$\ln\left[\ln\left(\frac{100}{R_D}\right)\right] = n[\ln(d) - \ln(d')]$$

R_d = % of particles with diameter greater than d (residue)

particle size in mircons µm

d = particle size in microns, such taht 36, 8% of the particles have a size greater than d'

d' = slope of RR straight line

n =

6. COMMINUTION PROCESS OPTIMIZATION

Before trying to optimize a comminution process, or any other process, it is essential to record the actual working state, to determine the working state of comminution machine, some process values are compared against values from other processes. Since there is no detailed knowledge about energy consumption for comminution, values like energy consumption, energy efficiency and energy utilization have to be compared with and related to other values, obtained under similar conditions.

Generally optimizing means that an industrial comminution process, which is fed with a preset and fixed material PSD, is manipulated in such a way, that it uses a less total energy input to, simultaneously, produce a greater amount of particles finer that set size.

The common ways of optimizing a comminution process are:

* Closed loop grinding, with separator
* Narrower PSD, giving less fines / less energy consumption
* Change of comminution machinery (upgrading)
* No 'overgrinding'
* Avoiding stochastical grinding force application

7. LABORATORY TESTS

7.1 General

The comminution theories, dealt with earlier, are empirically found and based on the knowledge of basic material parameters such as PSD, specific surface, grindability, abrasiveness, moisture content, etc.

Therefore the laboratory tests described below constitutes, together with practical comminution experience, a vital 'base of knowledge' for the design and improvement of comminution machines and processes.

All the tests described herein are performed at the HMC's laboratories.

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7.2 Particle Size Analysis

Since the materials used in cement manufacturing are mostly compounds, with any kind of Particle Size Distribution, it is essential to be able to determine the actual feed PSD and the transformations in the PSD during the comminution process.

The PSD is determined by sieving the material on several sieves, with different mesh sizers, and recording the weight passing or retaining on each sieve. It is obvious that the smaller the sieve size mesh intervals the better the PSD is depicted.

Below table summarizes the most common sieving methods.

Sieving method	Sieve range [micron]	Sample [gram]	Test time [min]
Mechanical (dry sieving)	33 - 2000	50	60
Alpine, air-jet (dry)	33 - 2000	50	60
Supersonic (wet sieving)	2 - 40	2,0	180
Sedigraf (wet)	0,1 - 300	1,0	60
Laser, Cilas (wet)	1 - 200	1,0	20
Laser, Sympatec (dry or wet)	1 - 875	(wet) 1,0 (dry) 50	20 8

After determining the PSD, by any of above listed test methods, it is essential to plot the PSD in a diagram and to calculate some significant PSD-values. Usually the cumulative percentage passing or residing on a sieve is plotted against the particle diameter (mesh size). Depending on the material fineness a suitable diagram is chosen.

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7.3 Specific Surface

The Specific Surface according to Blaine (SSB) is at time the most common single parameter for characterization of a cement and its behavior. The SSB-value indicates the magnitude of a powder's total particle surface per weight unit (cm^2/g).

Note, that if two powders, with the same chemical composition, are characterized by the same SSB-value, this does not necessarily mean that they have the same PSD.

The SSB-value is obtained by measuring the powder's air permeability with the test equipment shown in appendix. The measuring itself is performed by recording the time, t, needed for a specified amount of air to pass through the test sample in the permeability cell. The recorded time is indirectly proportional to the powder's specific surface.

The specific surface of a powder can also be calculated from a known PSD. Since, the individual particles in a powder mostly have an irregular form the formula for calculation of the SSC-value is extended with a form factor, f.

The SSC-value is also strongly influenced by the material density. At HMC the SSC-values are calculated with below formula, under the assumptions that the particles are all spherical (f = 1) and that the powder has a standard density (= 3.1 g/cm^3).

$$SSC = \frac{6f}{\rho} \cdot \sum_{i=1}^{16} \frac{R_i - R_i + 1}{\frac{1}{2}(d_i + d_i + 1)}$$

f: form factor (=1)

ρ: specific density off cement (g/cm^3)

R_i % of particles with diameter (microns) greater than d_i (I = 1,2,3...,16)

d_i particle equivalent diameter (di = 0.1, 0.3, 1, 1, 5, 2, 3, 4,....., 196)

For cements about 60 % of the calculated specific surface originate from particles less than 3 µm, which shows how essential it is to standardize the calculation formula to be able to compare SSC-values.

The correlation between the SSB-value and the SSC-value is shown by below empirical formula, which has been derived from great sample of cements with different PSD.

SSC = 807 + 1.2 x SSB

SSB = (SSC - 807) / 1.2

Note that a SSB-value is not the same as Specific Surface Calculated (SSC) value. The SSC-value is calculated from the PSD under the presumption that the particles have a spherical form, and usually indicates a greater total specific surface.

7.4 Grindability Test

The grindability is defined as 'the resistance behavior against size reduction of a material (or particle) in a grinding process'. Quantitatively the grindability is expressed as 'energy input' (energy consumption) required for the size reduction of one mass unit of material. The energy input depends of course on the type of mill and on the type of material to be ground.

Several grindability tests have been developed to simulate, on a laboratory scale, the industrial grinding performance. The material to be investigated is tested under strict observation of the relevant test conditions. The values means of scale-up factors.

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7.4.1 "Holderbank" Grindability Test

The basic test method was introduced by MITTAG and further developed at "Holderbank", Technical center.

Test equipment

Test mill: ∅ 0.5 x 0.5 m
Mill speed: 37.8 min. $^{-1}$
Ball charge: 100 kg
Sample weight: 10 kg (< 12 mm)

Test procedure

In a first step, the material is ground with balls of ∅ 70 mm (40 %) and ∅ 40 mm (60 %) to a fineness of about 60 % residue on the 90 µm sieve. Then the balls are removed and cubes of 17 x 17 mm are used for the subsequent steps. After 200 - 300 revolutions the mill is stopped for sampling and a PSD-analysis as well as specific surface (Blaine)-analysis is performed.

Test evaluation

During the grinding periods the torque Md is measured at the mill shaft. The specific power consumption $E_{spec.}$ can be calculated according to the following formula:

$$E_{spec} = \frac{Md \times \Delta n}{59{,}6} [kWh/t]$$

Md = torque on mill shaft [Nm]
n = number of revolutions of the mill in one grinding step

The specific energy consumption of the test mill has to be converted with a scale-up factor to obtain the specific energy consumption at the mill shaft of an industrial mill.

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7.4.2 HARDGROVE - Grindability Test

The HARDGROVE test is based on the comminution law developed by RITTINGER.

Test equipment

* Grinding bowl with 8 balls (∅ 25.4 mm)
* Grinding ring driven by motor of 0.2 kW
* Load of grinding ring on grinding bowl approx. 29 kg
* Sample: 50 g (590 <x> 1190 µm)

Test procedure

50 g of the material with a particle size limitation of minimum 590 µm and maximum 1190 µm is prepared and fed to the grinding bowl. After 60 revolutions the ground sample has to be removed from the bowl and the weight of material passing the 74 µm sieve to be determined.

Test evaluation

The HARDGROVE index H can be calculated from the weight D of the tested sample passing the 74 µm sieve.

$H = 13 + 6.93 \times D$

The HARDGROVE index H can be converted into the work index E_H:

$EH = 435 / (H^0 \times 82)$ [kWh/t]

The specific power consumption $E_{spec.}$ can then be calculated:

$$E_{spec} = E_H \left(\frac{10}{\sqrt{X_2}} - \frac{10}{\sqrt{X_1}} \right) [kWh/t]$$

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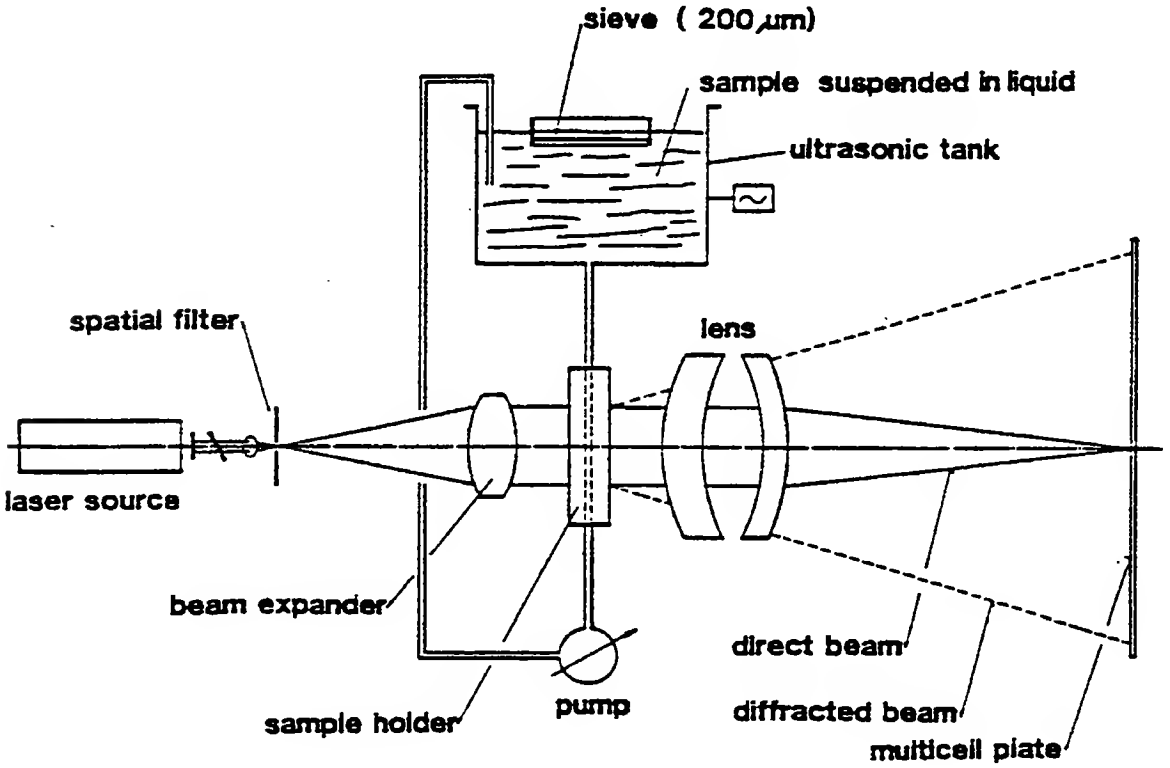
PARTICLE SIZE DISTRIBUTION TEST	
Test method:	CILAS (Laser)
Purpose of the test:	Determination of particle size distribution of cement and coal dust

Test equipment:



Test procedure:
- → Zero calibration of liquid
- → Sample preparation: 0.1 - 1.0 g (< 200 µm)
- → Test run
- → Automatic display of particle size distribution

Total testing time (16 particle fractions): 10 min.
Sieve range: 1 - 180 µm

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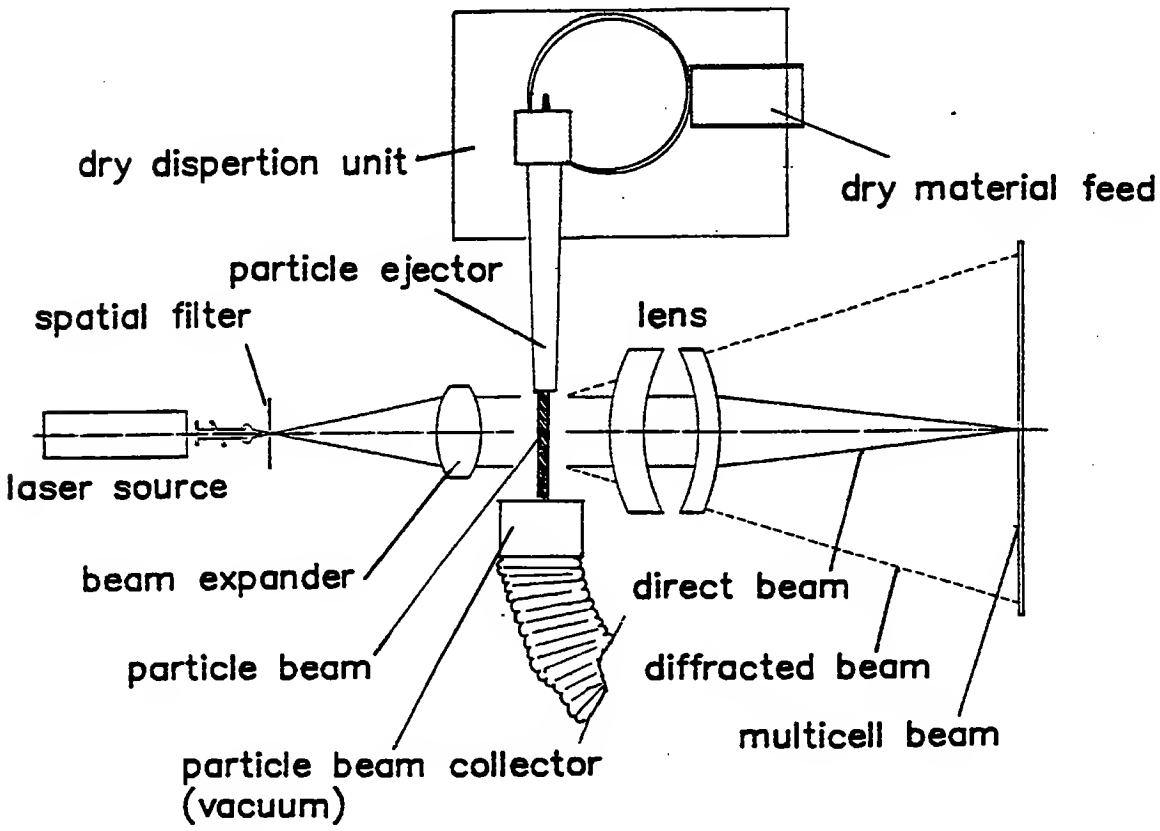
PARTICLE SIZE DISTRIBUTION TEST	
Test method:	Sympatec (laser, dry process)
Purpose of the test:	Determination of particle size distribution

Test equipment:



Test procedure:	→	Zero calibration	
	→	Sample preparation:	20 - 50 g (< 875 µm)
	→	Test run	
	→	Automatic display of particle size distribution	
		Total testing time (31 particle fractions):	5 min.
		Sieve range:	1 - 175 µm
			1 - 350 µm
			1 - 875 µm

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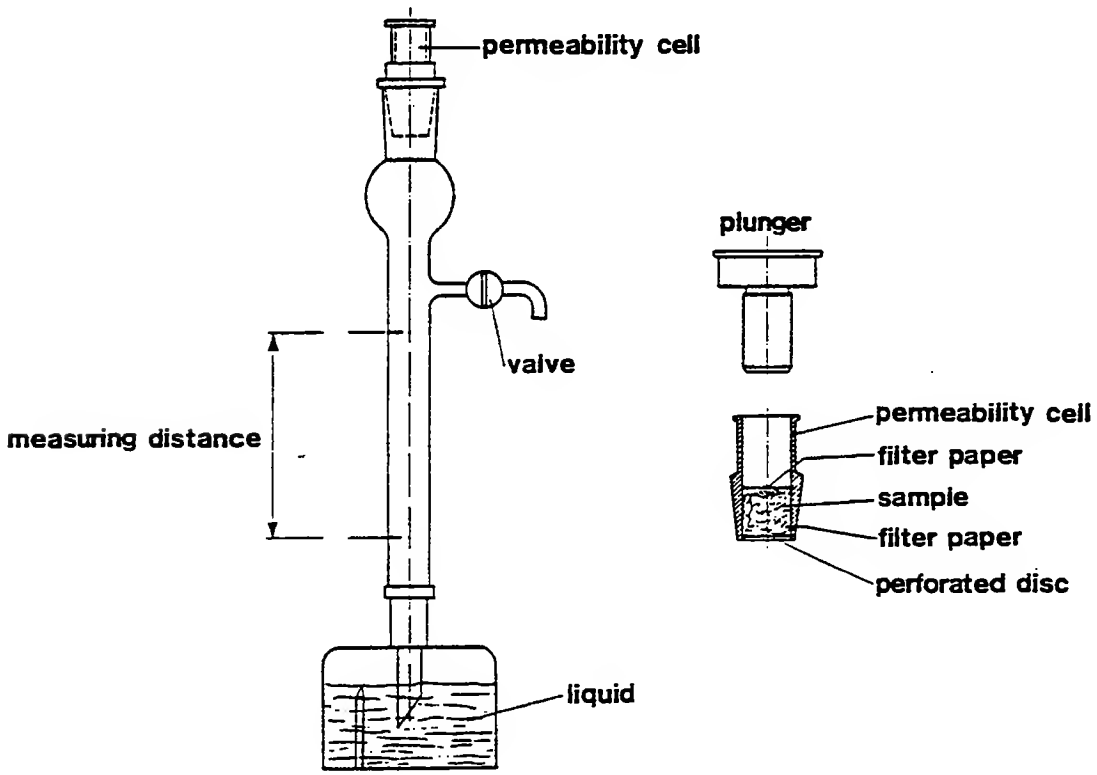
SPECIFIC SURFACE TEST	
Test method:	BLAINE
Purpose of the test:	Determination of specific surface of cement

Test equipment:



TEST PROCEDURE :

- **Sample preparation** : **2,7 g**
- **Three test runs**
- **Time measurement** : **t**
- **Determination of bulk weight** : **ϱ**

} **cm^2/g**

Total testing time : **5 - 10 min**

Measuring range : **1000 - 8000 cm^2/g**

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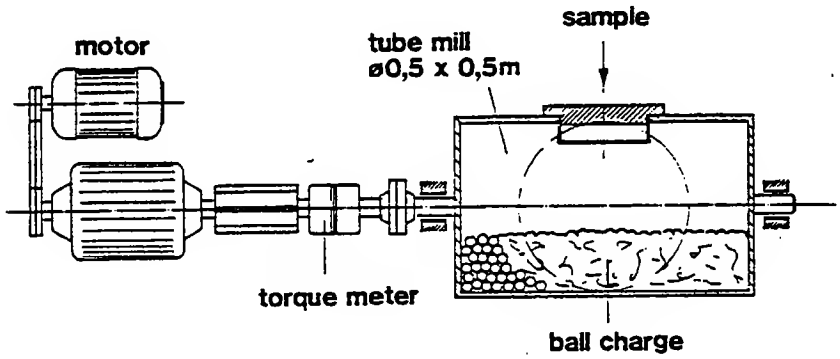
GRINDABILITY TEST	
Test method:	HOLDERBANK
Purpose of the test:	Determination of energy input for grinding of raw material and clinker

Test equipment:



TESTING PROCEDURE :

- → Sample preparation : 10 kg (< 12mm)
- → Test run : stepwise (100–200 revolutions
- → Sieve test : % Residue (90 µm)
- → Torque measurement
- → Number of revolutions

→ kWh/t

TEST EVALUATION :

- → Plotting of grindability curve
  → kWh/t in function of product fineness
- → Converting of test results (scale-up) to industrial mill
  → mill motor size of tube mill

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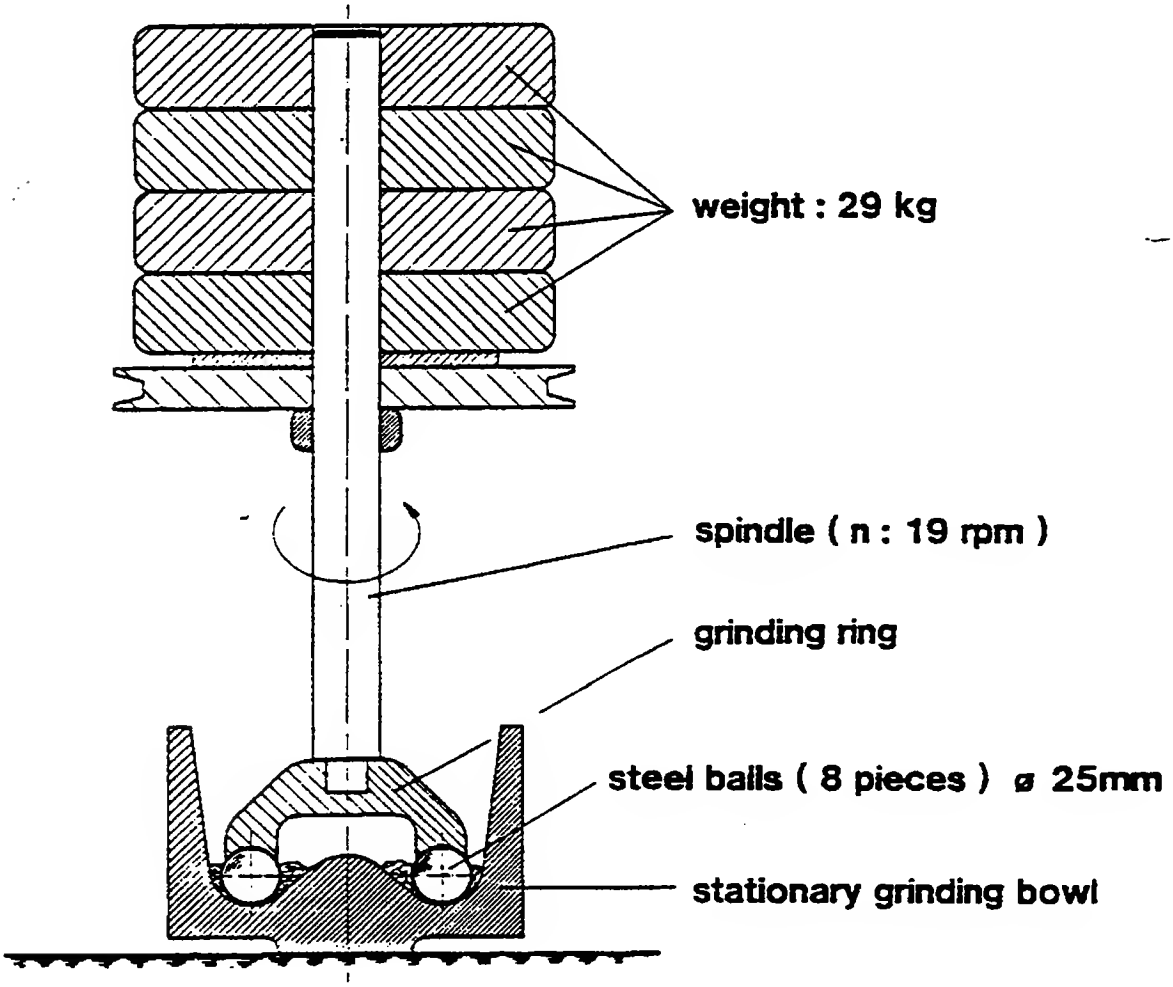
GRINDABILITY TEST	
Test method:	HARDGROVE
Purpose of the test:	Determination of energy input for grinding of coal

Test equipment:



Test procedure:	→	Sample preparation:	50 g (590 - 1190 µm)
	→	Test run:	60 revolutions
	→	Sieve test:	D = weight of particles < 90 µm
Test evaluation:	→	Calculation of Hardgrove Index H	
	→	Comparison with standard material	
	→	Conversion to energy input (kWh/t)	
		→ mill motor size of coal mill	

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Chapter 7

Tube Mills

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Dimensioning of Tube Mills

S. Ortega
PT 99/14486/E

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1. INTRODUCTION

The tube mill is a simple, approved machine, not difficult to operate well and still competitive when compared with more modern type of mills.

This paper deals with the design criteria used to dimension a tube mill. These criteria are guidelines. But there is no complete analytical theory for optimal mill design and, therefore, this topic still remains a matter of experience.

2. MAIN DESIGN CRITERIA

2.1 Length to diameter ratio

The relation between the mill shell length and the shell internal diameter is the length to diameter ratio λ (fig. 1).

The length to diameter ratio depends on various factors. The most important ones are:

- Hourly throughput,
- Type of material to be ground,
- Fineness of the finished product,
- Mill in open or closed circuit,
- Fresh feed size (one stage or two stage grinding),

On one side the hourly throughput depends on the mill diameter. On the other side the fineness at the mill outlet depends mainly on the time the material remains inside the mill, also called retention time. The main factor influencing the retention time is the mill length. Therefore, the ratio of length to diameter of a mill is an important factor for an optimum design of the mill. Table 1 shows guidelines for the length to diameter ratio λ:

Table 1: Length to diameter ratio λ for different mill systems

Type of mill system	Ratio λ	Remarks
Cement mills		
Open circuit mills	3.0 – 6.0	
Closed circuit mills	3.0 – 3.5	large mills: λ = 2.8 - 3.2
Closed circuit mills with pre grinding unit	2.8 – 3.5	low λ yields higher circulating load
Slurry mills		
Ratio λ similar as for cement mills. Max. length 12 - 14 [m]		
Raw mills		
Center discharge mill	2.1 – 2.7	The lower λ are applied for large raw mills
Two comp. Mill	2.0 – 2.5	
Single comp. mill	1.7 – 2.2	
Air swept mill	1.5 – 2.0	

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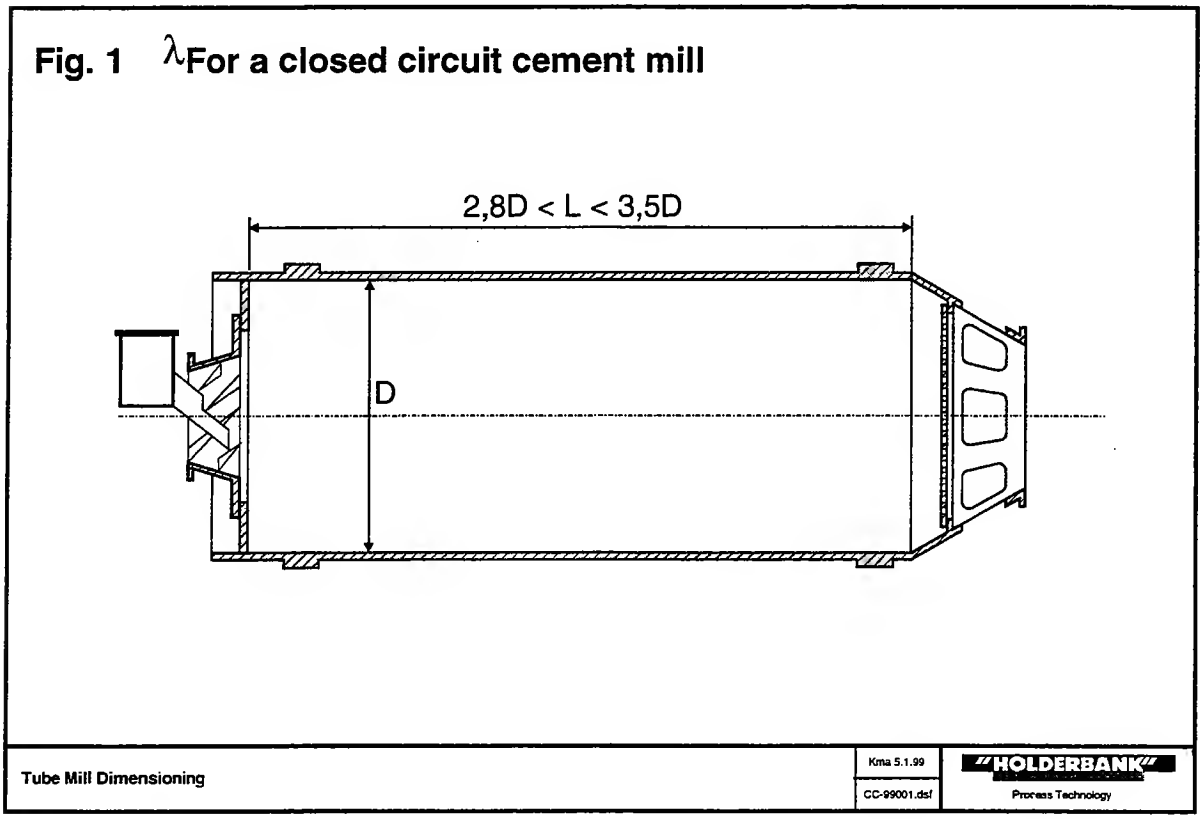
Fig. 1 λ For a closed circuit cement mill



2.2 Mill internal dimensions

The mill shell would only last a few thousand hours if not conveniently protected by shell liners. These internal elements determine the shell internal dimensions.

2.2.1 Mill useful length

The mill useful length is defined as the shell length L reduced by the cumulated width of the head liners and the diaphragms (intermediate and outlet).

Figure 2 shows the mill useful length (L_u) for a two-compartment mill.

L_{ID} is the total width of the intermediate diaphragm. For normal applications this value is in the order of 400 [mm].

2.2.2 Length of compartment

For mills with two or more compartments there are some guidelines for the length of each different compartment as shown in table 2

Figure 2 shows for a two chamber mill the length of the first (L_{u1}) and second (L_{u2}) compartments of a two chamber mill.

Table 2: Length of Grinding Compartments for Different Types of Mills (see Fig. 7)

Type of mill	[%] of total useful length		
	Comp. I	Comp. II	Comp. III
Two comp. mill	30 - 35	70 - 65	---
Three comp. mill	20	30	50
Center discharge mill	50	50	---

2.2.3 Mill internal diameter

The mill internal diameter is defined as the mill shell internal diameter (D) reduced by twice the liner average thickness ($\bar{e}$).

As already defined in chapter "tube mills" there are many different types of liners. Figure 3 shows typical liner thicknesses for first (C1) and second (C2) chamber as a function of the mill shell diameter.

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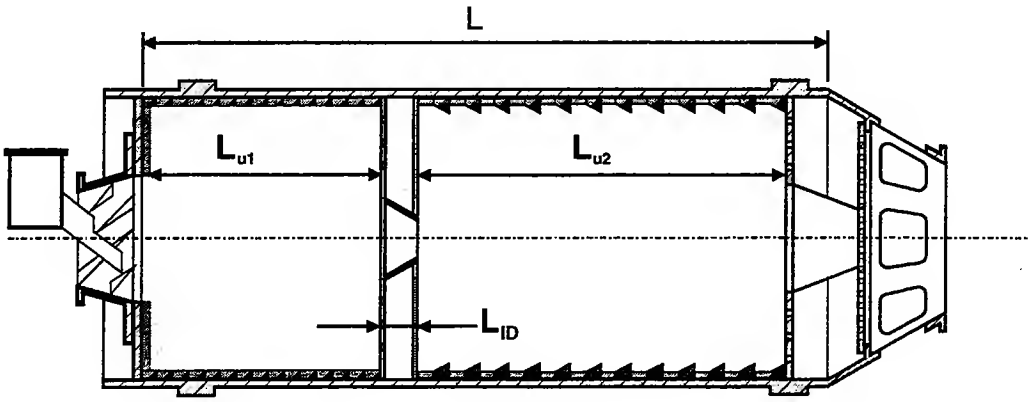
Figure 2 & 3

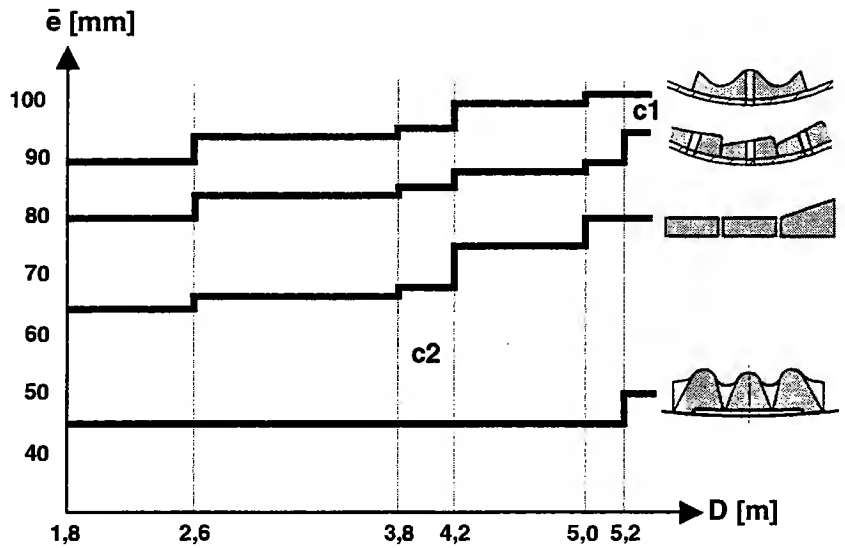
Fig. 2 Useful lenght for two chambers mill



$$L_u = L_{u1} + L_{u2}$$

$$L_{ID} \sim 400 \text{ [mm]}$$

Fig. 3 Liner thickness



$$Di = D - 2 \cdot \frac{\bar{e}}{1000} \text{ [mm]}$$

Tube Mill Dimensioning	Kma 5.1.99	"HOLDERBANK"
	CC-99002.dsf	Process Technology

2.3 Filling degree

The filling degree f is defined as the volume V_Q of the grinding media charge expressed as a percentage of the total useful mill volume V_M.

$$f = \frac{V_Q}{V_M} \times 100[\%] \qquad (1)$$

Table 3 shows guidelines for filling degree values:

Table 3: Filling degrees for different types of tube mills

Type of mill	Filling degree [%]		
	Comp. I	Comp. II	Comp. III
Single comp. mill	27-33	---	---
Two comp. mill	27-33	25-32	---
Three comp. mill	26-32	26-30	23-27
Air swept mill	≈ 26	---	---

Studies have proven that there is a maximum grinding efficiency for a filling degree of 26 – 28 [%]. Above this value the higher the filling degree the lower the grinding efficiency. Nevertheless, in countries with low electrical energy cost and high market demand, filling degrees of 40 – 45 [%] are used to maximise mill production.

The filling degree in a tube mill can be practically determined by measuring the free heights h above the grinding charge according to figure 4.

Due to liner design it is not possible to measure the free height accurately. Figure 5 shows an alternative method to determine the free height. Having measured d and h', h can be calculated and graph from figure 4 can be used with.

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Fig. 4 & 5

Fig.4 : Filling degree f in function of free height h above grinding media charge

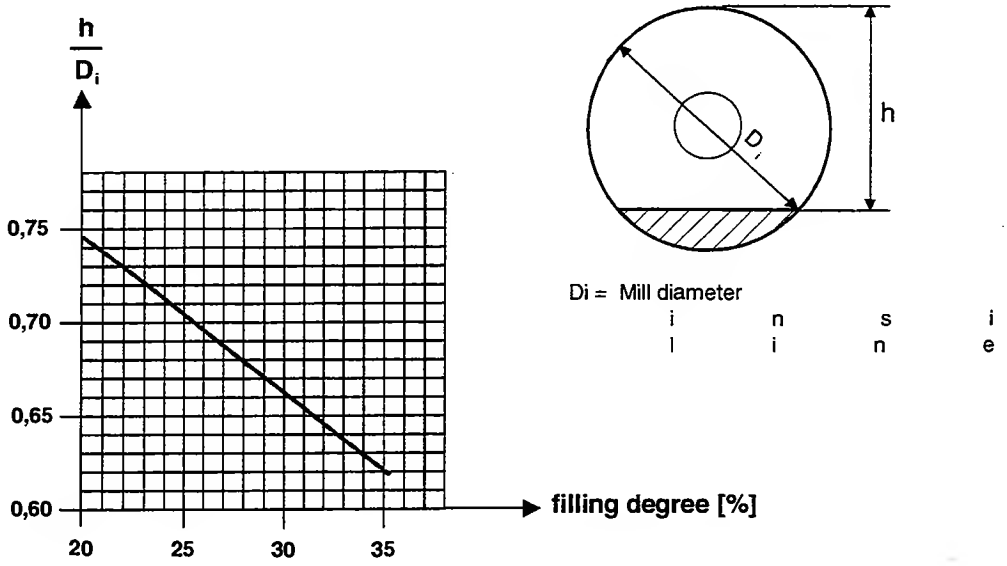
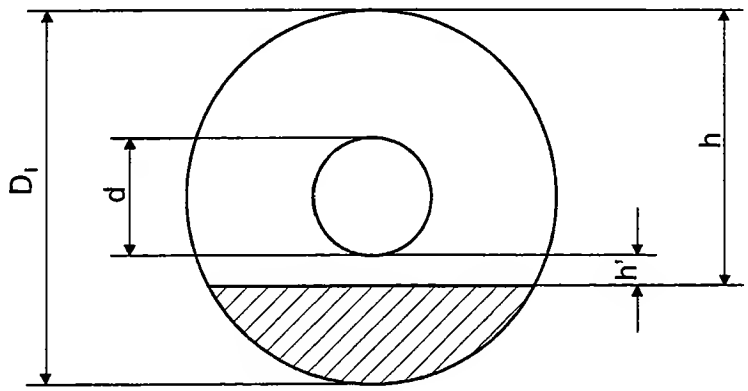


Fig.5 : Alternative way to determine the free height

$$h = \frac{Di + d}{2} + h'$$



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2.4 Weight of grinding media

The weight of the grinding media charge can be calculated as follows:

$$Q = \frac{\Pi}{4} \bullet D_i^2 \bullet L_u \bullet \frac{f}{100} \bullet \gamma_Q \quad [t] \qquad (2)$$

where:	L_u	=	internal length of the mill or compartment	[m]
	f	=	filling degree	[%]
	γ_Q	=	grinding media bulk weight	[t/m^3]
	D_i	=	internal mill diameter (inside lining)	[m]
	Q	=	grinding media charge	[t/m^3]

The bulk densities of the grinding media are given in the following table:

Table 4: Bulk weight of grinding media charge (Figure 6)

	Ball size ∅ [mm]	Bulk weight [t/m^3]
Steel balls	100 – 60	4.4
	50 – 30	4.6
	30 – 20	4.7
Cylpebs	30 – 20	4.8

Figure 6 shows the grinding media bulk weight distribution in a two chamber mill equipped with classifying liners in the second chamber.

For calculation 4.4 [t/m3] and 4.65 [t/m3] are used as grinding media bulk weight for first and second chamber respectively.

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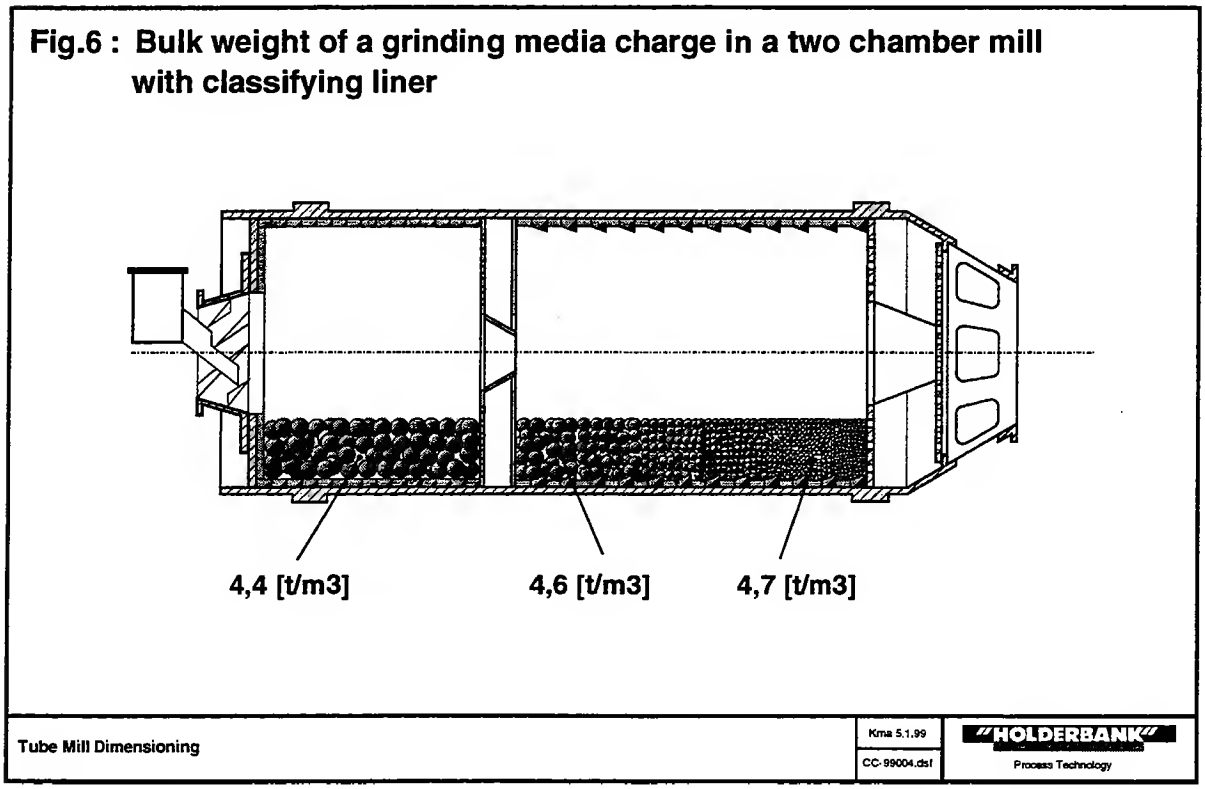
Fig. 6: Bulk weight of a grinding media charge in a two-chamber mill with classifying liner



2.5 Mill speed

The operating speed of the mill can be expressed as a percentage of the critical mill speed. Critical speed is attained when centrifugal force F_C compels the outer layer of grinding media to rotate with the mill lining.

The critical speed is obtained when the centrifugal force F_C is equal to the force of gravity F_G; i.e. (Fig. 7):

$$F_G - F_C = m \bullet g - m \bullet \frac{D_i}{2} \omega^2_{crit} = 0 \quad (3)$$

The critical angular speed **ϖcrit** will then be:

$$\omega_{crit} = \sqrt{\frac{2 \bullet g}{D_i}} \quad (4)$$

calculated as a function of the mill diameter D_i

$$n_{crit} = \frac{30}{\Pi} \bullet \sqrt{\frac{2 \bullet g}{D_i}} \quad (5)$$

$$n_{crit} = \frac{42.3}{\sqrt{D_i}} \quad [min^{-1}] \quad (6)$$

The operating mill speed **n** is then calculated as:

$$n = \frac{k}{100} \bullet n_{crit} \quad [min^{-1}] \quad (7)$$

n	=	operating mill speed	[min^{-1}]
n_{crit}	=	critical mill speed	[min^{-1}]
k	=	ratio n/n_{crit}	[%]
D_i	=	internal mill diameter	[m]

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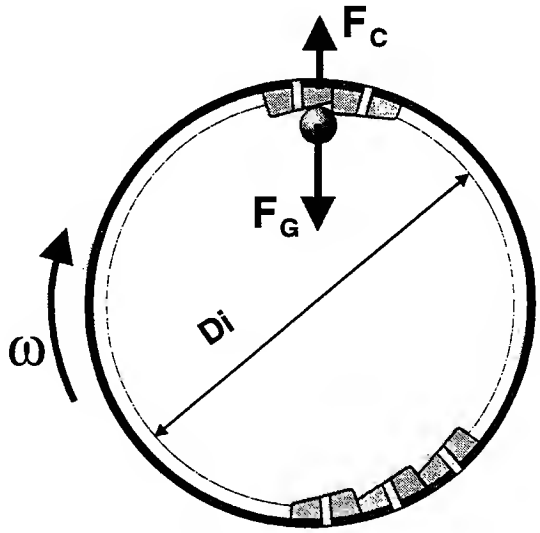
Fig. 7: Critical Mill speed

Fig.7 : Critical mill speed



FC =centrifugal force [N]

FG =force of gravity [N]

Di =mill diameter inside liner [m]

ω =angular speed [s-1]

g =gravity [ms-2]

$$n_{crit} = \frac{42{,}3}{\sqrt{D_i}} \quad [min^{-1}]$$

k = [%] of critical speed

$$n = k * n_{crit} \quad [min^{-1}]$$

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The critical speed chart in Figure 8 is based on the equations 6 and 7 and allows a quick determination of the critical speed for tube mills with various diameters.

Figure 9 shows the ball charge behavior under different combinations of filling degrees and percentage of critical speed.

Low filling degrees and low percentage of critical speed do not allow an efficient ball charge action on the material. High filling degrees and high percentage of critical speed lead to ball charge centrifugation and very little grinding efficiency.

Mills with high percentage of critical speed can be operated with low filling degrees but wear of internal element is very high.

The framed picture in Fig. 9 shows a typical first chamber ball charge behavior for 30 [%] of filling degree and 70 [%] of critical speed.

Grinding efficiency shows a somewhat indefinite peak in the range between 65 [%] and 75 [%] of critical speed. The driving power increases with mill speed. However, this linear relationship is only valid for the above mentioned range of speed Therefore, an increase in out-put can be expected by increasing speed.

Modern mills have a speed range from 70 – 75 [%] of critical mill speed.

The type of liner, the filling degree and the ball charge composition are to be adjusted to the mill speed for optimum grinding performance.

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Fig. 8 & 9

Fig.8 : Percentage of critical mill speed in function of mill diameter

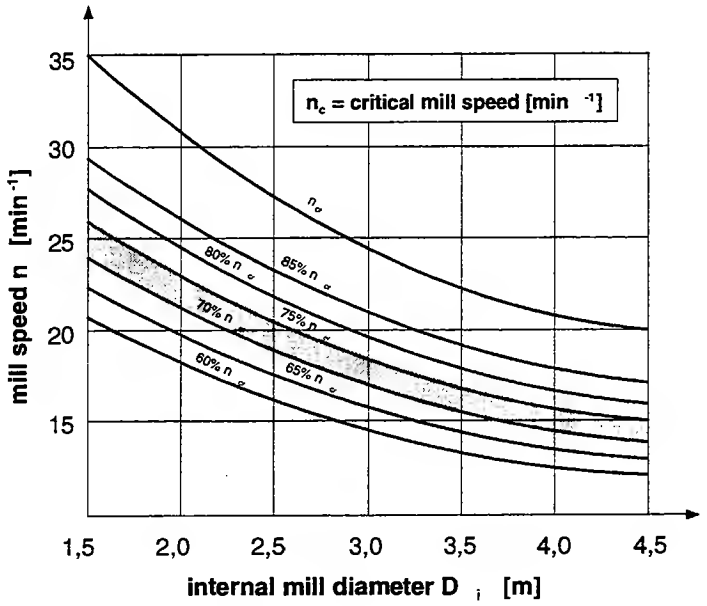
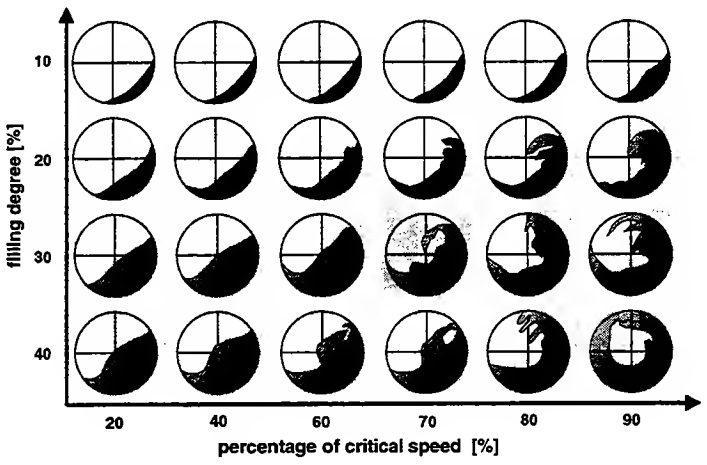


Fig.9 : Cascading effect of grinding media charge in function of filling degree and critical speed



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3. NET DRIVING POWER

The net driving power for a tube mill is the power necessary at the mill shell to maintain the centre of gravity of the ball charge load in a position of kinetic equilibrium.

According to MITTAG the distance **b** (Figure 10) can be expressed as a function of the mill diameter D_i, taking into account the following assumptions:

- Distance **b** is always in the same relationship with the diameter for mills with the same filling degree but different diameters.
- Distance **b** does not depend on the mill speed for the usual range of speed.

The following formula can then be derived:

$$b = x \bullet D_i$$

The torque **M** can be expressed as:

$$M = x \bullet D_i \bullet Q \ [Nm]$$

and the net driving power **P** as a function of the angular speed ω and the torque **M**

$$P = M \bullet \omega \qquad [kW]$$

With the angular speed ω being

$$\omega = \frac{\Pi \bullet n}{30} \qquad [s^{-1}]$$

the net driving power **P** can finally be written as

$$P = \frac{x \bullet P_i \bullet Q \bullet \Pi \bullet n}{30} \qquad [kW]$$

For practical calculations the formula can be simplified to

$$P = c \bullet Q \bullet D_i \bullet n \qquad [kW] \quad (8)$$

P	=	Net driving power	[kW]
Q	=	Grinding media charge	[t]
$\mathbf{D_i}$	=	Mill diameter inside liners	[m]
n	=	Mill speed	[min^{-1}]
c	=	Power consumption factor	[-]

$$c = \frac{x \bullet \Pi}{30}$$

Formula (8) allows the determination of the net driving power for a tube mill with an accuracy within 5 – 10 [%].

For multi-compartment mills the total net driving power can be calculated as the sum of driving powers for each individual compartment.

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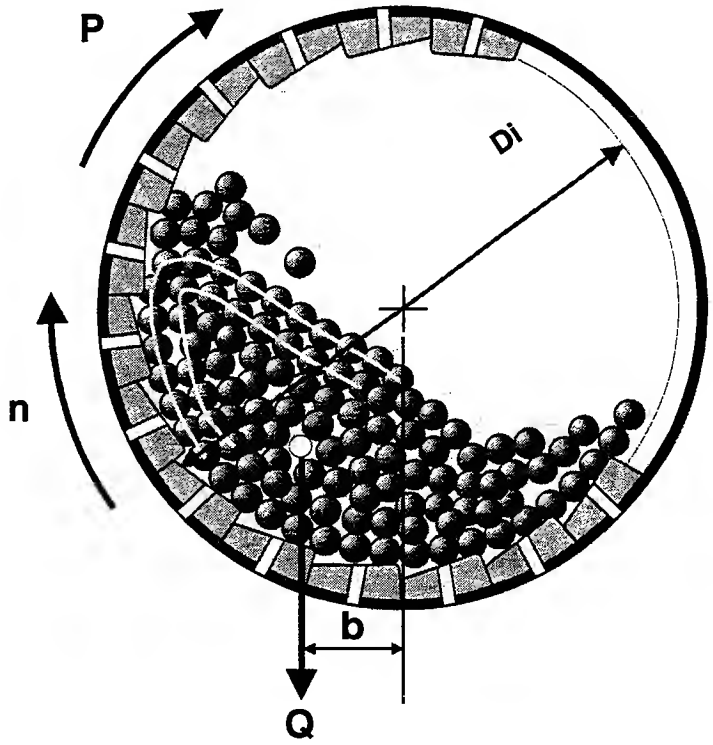
Fig. 10: Net driving power or mill shell power

Fig.10 : Net driving power or mill shell power



Di	=	diameter inside liners	[m]
Q	=	weight of grinding media charge	[t]
n	=	speed of the mill	[min-1]
P	=	net driving power	[kW]

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The power consumption factor **c** depends on filling degree and on grinding media size. The value of **x** and therefore the value of **c** cannot be calculated theoretically. But by measuring the total driving power of industrial mills in operation and by considering the known values of **Q**, $\mathbf{D_i}$ and **n**, the power consumption factor c can be determined.

Figure 11 shows this factor **c** versus the filling degree **f**.

The driving power is proportional to the:

- Weight of the grinding media charge
- Distance between the centre of gravity of the grinding media charge and the mill rotation centre, or lever.

As the filling degree increases, the weight of the grinding media increases but the lever decreases. It results that the driving power has a maximum value for a filling degree within 40 – 45 [%].

Formula (8) indicates that the net driving power varies as the grinding media charge, which in turn varies as the square of the diameter $\mathbf{D_i}$ for a given shell length; that driving power likewise varies as the diameter; and that it varies as the speed, which is an inverse function of the diameter, it follows that

$$P \approx D_i^2 \bullet D_i \frac{1}{\sqrt{D_i}} = D_i^{2.5}$$

This formula means that the net driving power of tube mills of the same length operating under the same relative conditions varies with 2.5 power of the internal diameter. Differences of length can be considered by the following formula

$$\frac{P_1}{P_2} = \frac{D_{i_1}^{2.5} \bullet L_{i_1}}{D_{i2}^{2.5} \bullet L_{i2}}$$

By assuming the same ratio of length to diameter for both mills the above formula can be written as:

$$\frac{P_1}{P_2} = \left[\frac{D_{i_1}}{D_{i_2}}\right]^{3.5} \qquad (9)$$

It can be seen (from the above) that the diameter of a tube mill has a great influence ($\sim D^{2.5}$) on the driving power, whereas the length of the mill has just linear influence (Figure 12).

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Fig. 11 & 12:

Fig. 11 : Factor c, depending on filling degree and ball size

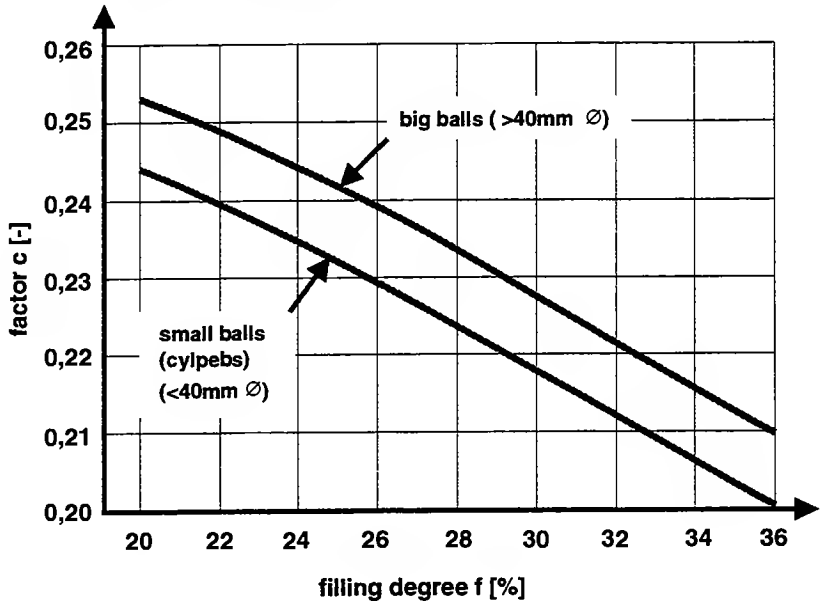
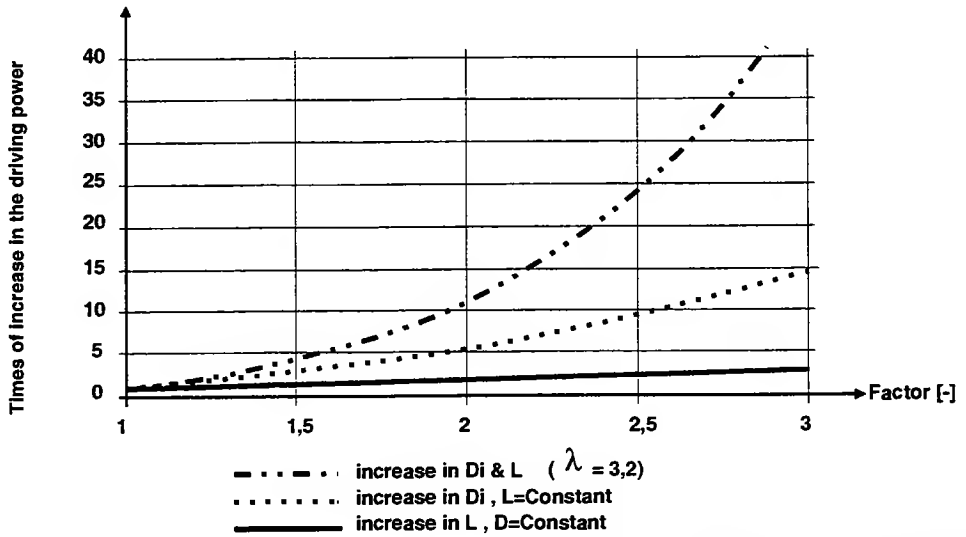


Fig. 12 : Increase of the net driving power of a tube mill with the Di and L Point (1,1) : Di=2, L=6.4



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4. POWER TRANSMISSION CHAIN

Mill power is always dimensioned at mill motor shaft.

The power available for grinding is referred at the mill shell, also called net power. The power consumption the cement plant is billed for is measured at the mill motor kilowatt - hour meter. All these defined powers are lined to each other through the power transmission chain.

Figure 13 shows two possible transmission chains. The one in full line is for a mill with a central drive. The one in doted line is for a mill with a gearth drive.

The efficiency figures shown are only indications and might vary with the type of mill drive and the mill power (see fig. 11 in the chapter "Tube Mills").

η_1 to η_4 are the efficiencies along the power transmission chain:

- η_1: Due to cable loss between the mill motor and the power measuring point. Bigger than 99 [%] except if wattmeter very far.
- η_2: Motor efficiency. (< 98 [%]).
- η_3: Gearbox efficiency (< 97 [%]).
- η_4: Pinion transmission efficiency. Bigger than 99 [%] if pinion in good state.

To avoid misunderstandings or in the worst case a wrong mill design it is of utmost importance to mention to which link of the chain the power is referred.

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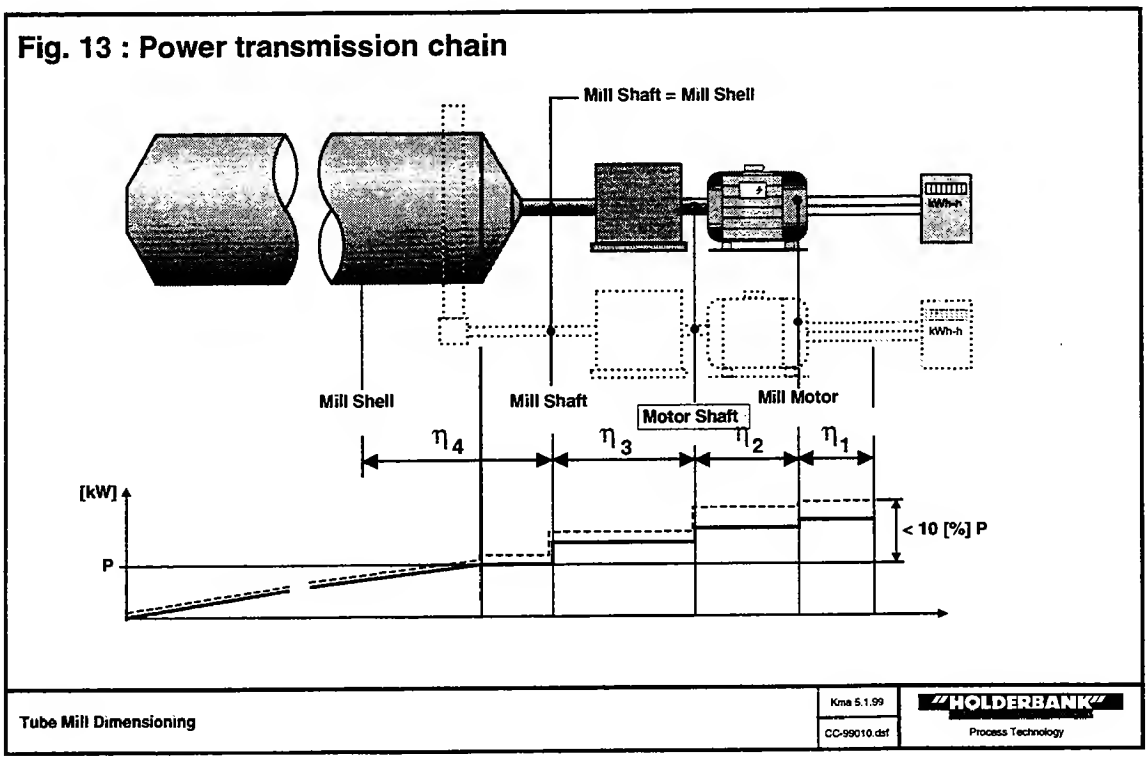
Fig. 13: Power transmission chain



5. PRACTICAL CALCULATIONS

For rough calculation of the main mill dimensions the mentioned formulas:

$$Q = \frac{\Pi}{4} \bullet D_i^2 \bullet L_u \bullet f \bullet \gamma_o \quad (2)$$

$$n_{crit} = \frac{42.3}{\sqrt{D_i}} \quad (6)$$

$$n = \frac{k}{100} \bullet n_{crit} \quad (7)$$

$$P = c \bullet Q \bullet D_i \bullet n \quad (8)$$

can be combined to one formula:

$$P = c \bullet Q \bullet D_i \bullet n = c \bullet \frac{\Pi}{4} \bullet D_i^2 \bullet L_u \bullet \frac{f}{100} \bullet \gamma_o \bullet \frac{42.3}{\sqrt{D_i}} \bullet \frac{k}{100}$$

$$P = 33.22 \bullet 10^{-4} \bullet c \bullet L_u \bullet f \bullet \gamma_o \bullet k \bullet D_i^{2.5}$$

Introducing the length to diameter ratio $\lambda = \frac{L_u}{D_i}$ yields the following equation:

$$P = 33.22 \bullet 10^{-4} \bullet c \bullet \lambda \bullet f \bullet \gamma_o \bullet k \bullet D_i^{3.5} [kW] \quad (10)$$

The internal mill diameter can therefore be calculated according to formula (10) :

$$D_i = \left[\frac{P}{33.22 \bullet 10^{-4} \bullet c \bullet \lambda \bullet f \bullet \gamma_o \bullet k} \right]^{1/3.5}$$

$$D_i = \left[\frac{P}{33.22 \bullet 10^{-4} \bullet c \bullet \lambda \bullet f \bullet \gamma_o \bullet k} \right]^{0.286} \text{ [m]} \quad (11)$$

P	=	mill net driving power	[kW]
c	=	power consumption factor	[-]
Q	=	weight of grinding media charge	[t]
L_u	=	useful mill length	[m]
D_i	=	internal mill diameter	[m]
k	=	percentage of critical speed	[%]
f	=	filling degree	[%]
γ_o	=	bulk weight of grinding media	[t/m^3]
λ	=	length to diameter ratio	[-]

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Tube Mills

M. Bürki
VA 95/4313/E

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1. INTRODUCTION

The mechanical elements of a tube mill can be subdivided into internal and external parts.

The external parts of a standard tube mill consist of mill shell, mill heads, mill bearings, feed and discharge equipment and include also the mill drive.

The internal parts have a direct function with regard to the grinding process and include principally the wear parts of a mill such as mill liners, diaphragms and grinding media.

This paper treats first the external parts and afterwards the internal parts of the tube mill. Special attention has been paid to the process technological aspects.

2. GRINDING WITH TUBE MILLS

The rotation of the mill tube causes the charge consisting of grinding media and feed material to be lifted due to centrifugal forces and friction between the media and the lining. The height to which the charge is lifted depends on a number of factors:

- liner design
- circumferential velocity of the mill
- shape, size and weight of the grinding media
- friction between the lining and the grinding media
- friction within the mill charge itself

The media performs a **cataracting motion** as can be seen in Figure 1a if the speed of rotation of the mill is sufficiently high, the loading percentage is appropriately chosen, the ball charge is relatively coarse and lifting liners are installed.

The feed material is ground mainly by impact in the zone "A", where almost the entire energy of the falling grinding media is concentrated. This form of comminuting action is especially effective in the primary size reduction of relatively coarse feed material supplied to the mill (chamber 1 or coarse grinding compartment).

Under similar conditions, but with a finer ball charge and without lifting liners the grinding media will perform a cascading motion as can be seen in Figure 1b.

In cascading the motion of the grinding media in their downward stream is characterized by flowing and rolling rather than falling. This motion of the grinding media causes friction forces. For this reason, cascading is not very suitable for the comminution of coarse feed material, but is on the other hand very effective for fine grinding (chamber 2 or fine grinding compartment).

Through the design of liners, the composition of the grinding media charge and the loading percentage of the mill it is possible to modify the motion of the grinding media so as to adapt it to the operating conditions in any given case.

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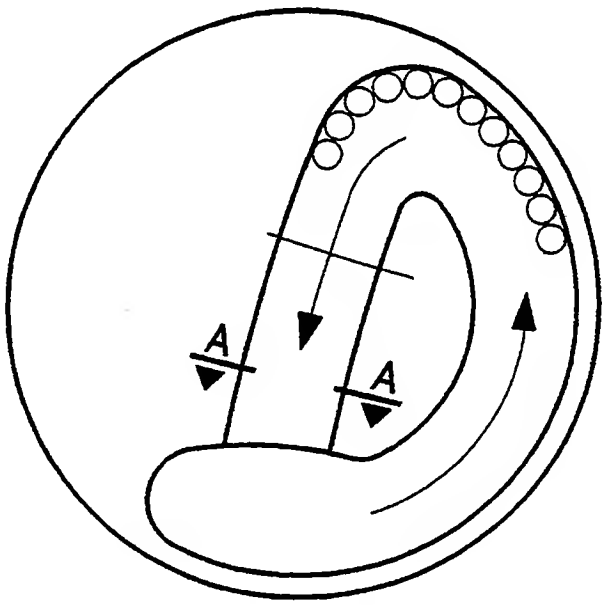
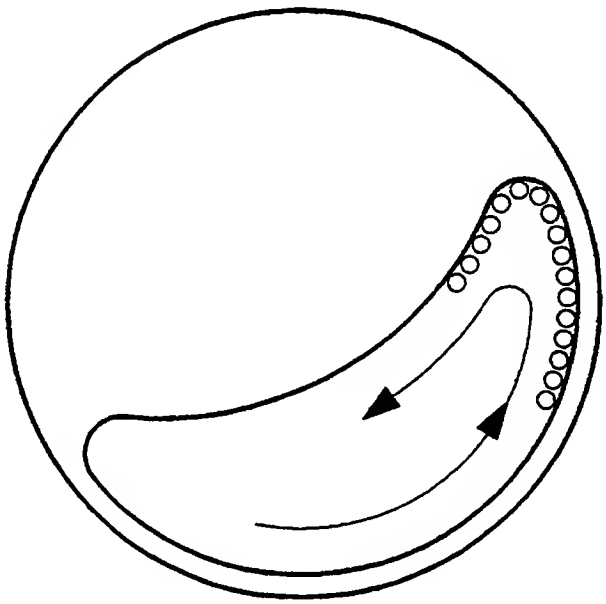
Figure 1 Grinding with Tube Mills

a) Cataracing of grinding media (Chamber 1)



b) Cascading of grinding media (Chamber 2)



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3. MILL TYPES

Since there is a various number of ball mill setups a subdivision according to the following characteristics can be made (Fig. 2):

- type of material which is being ground
- number of chambers (drying or grinding)
- method of material transport at mill outlet
- point of discharge
- grinding process

The different characteristics can be commented as follows:

3.1 Type of material being ground

The majority of today's installed tube mills are used for grinding cement. Besides cement a variety of other materials are being ground in tube mills such as raw meal, coal etc.

3.2 Number of chambers

A modern cement mill can consist of two or one chamber. Raw material mills are often equipped with one drying compartment to evaporate the water contained in the raw material and one or two grinding chambers. The drying compartment is provided with lifters to obtain a dispersion of the material to be dried. Ball mills with only one chamber are mostly applied in grinding circuits where the material enters the mill preground or raw mills for easy grindable raw material.

3.3 Material transport

There are basically two ways of transporting the material after leaving the mill.

The so called airswept mills carry the material pneumatically to the separator and are not equipped with a bucket elevator. This type of mill is only chosen if a high airflow is needed for the drying of very wet raw materials (or coals). The mechanical material transport by bucket elevators is more economical.

3.4 Discharge point of material

The discharge point of the material can be located at the mill end (end discharge mill) or in the centre (centre-discharge mill). With the exception of wet grinding mills where no airflow takes place the material and the air leaves the mill either at the mill end or at its centre.

3.5 Grinding process

Two grinding processes are applied in existing raw grinding plants, wet grinding (slurry mills) and dry grinding.

Since vertical roller mills are more economical than ball mills and also have the ability to dry extremely wet materials the ball mills were not considered in the latest evaluations of raw grinding plants. However ball mills are still common in wet grinding plants.

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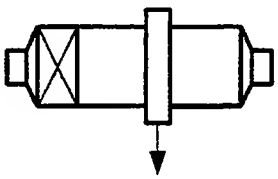
Figure 2 Overview Tube Mills

Characteristics	Classification			
Material	Cement	Raw meal	Coal	Others
Number of chambers	1 chamber: 1	2 chambers: 1 2	3 chambers: 1 2 3	
Material Transport	Mechanical		Pneumatical	
Discharge point of material				
Grinding process	Dry		Wet (Slurry)	

4. ELEMENTS OF TUBE MILLS

The following table and the according drawing (Fig. 3) show the classification (division) of the mechanical elements and their location on the example of a two compartment tube mill.

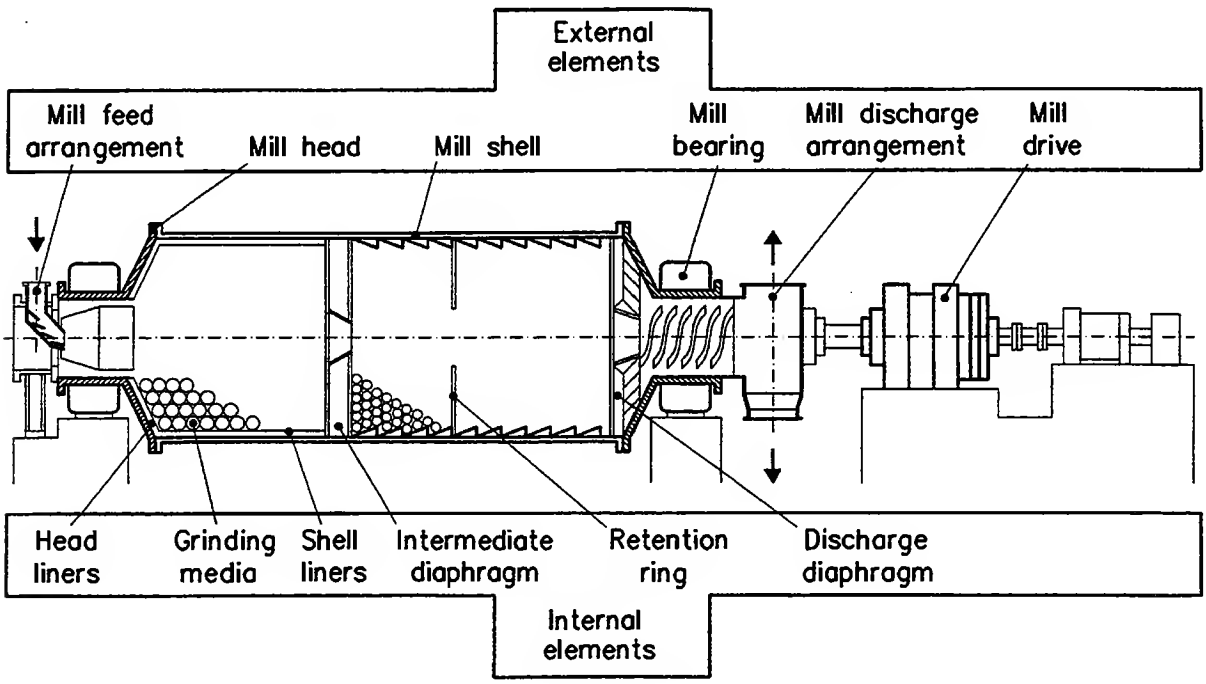
The external elements are introduced first in chapter 5. The internal elements follow in chapter 6.

feed arrangement	head liners
discharge arrangement	shell liners
mill shell	intermediate
mill heads	diaphragm
mill bearings	discharge
mill drive	diaphragm
	retention rings
	grinding media

The dimensions and design of the mill shell, the mill heads and the mill bearings are basically dictated by the required grinding capacity whereas the design of the feed and discharge devices are selected according to process considerations such as wet grinding, dry grinding or drying-grinding.

The internal elements are the so called wear elements. They have a direct influence on the grinding process.

Figure 3 Elements of Tube Mills Overview



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5. EXTERNAL ELEMENTS

5.1 Feed arrangements

The feed devices have to fulfill the following functions:

- enable the material to flow into the mill continuously and without building blockages
- prevent the back-spillage of material
- permit the intake of cold ventilation air in case of cement mills
- permit the intake of hot gas in case of raw grinding mills

There are different types of feed arrangements which meet above listed requirements. A selection is shown in Figure 4 and explained below.

5.1.1 Spout feeder

This type of feeder is applied on slurry mills and on older small cement mills. The limited cross section permits only a small intake of venting air in case of dry grinding.

5.1.2 Drum feeder

FLS provides their tube mills for wet and dry grinding with a drum feeder. The disadvantage of this feed device is its high pressure loss which appears in the dry grinding process where the mill should be ventilated.

5.1.3 Step type feeder

This type of feeder is used where high venting air or high amount of drying gas is required. The big opening gives a low pressure drop. The air can also find its way into the mill by passing in between the steps.

5.1.4 Feed chute of airswept mills

The mill has its name from the way of transporting the raw material. The ground material is swept out by the mill exhaust gases. This requires a high gas flow rate within the mill. Therefore its inlet device is designed to meet this requirement. The raw material is fed into the gas stream entering the mill. To prevent the intake of false air the feed inlet has to be sealed by pendulum flaps or by rotary valves.

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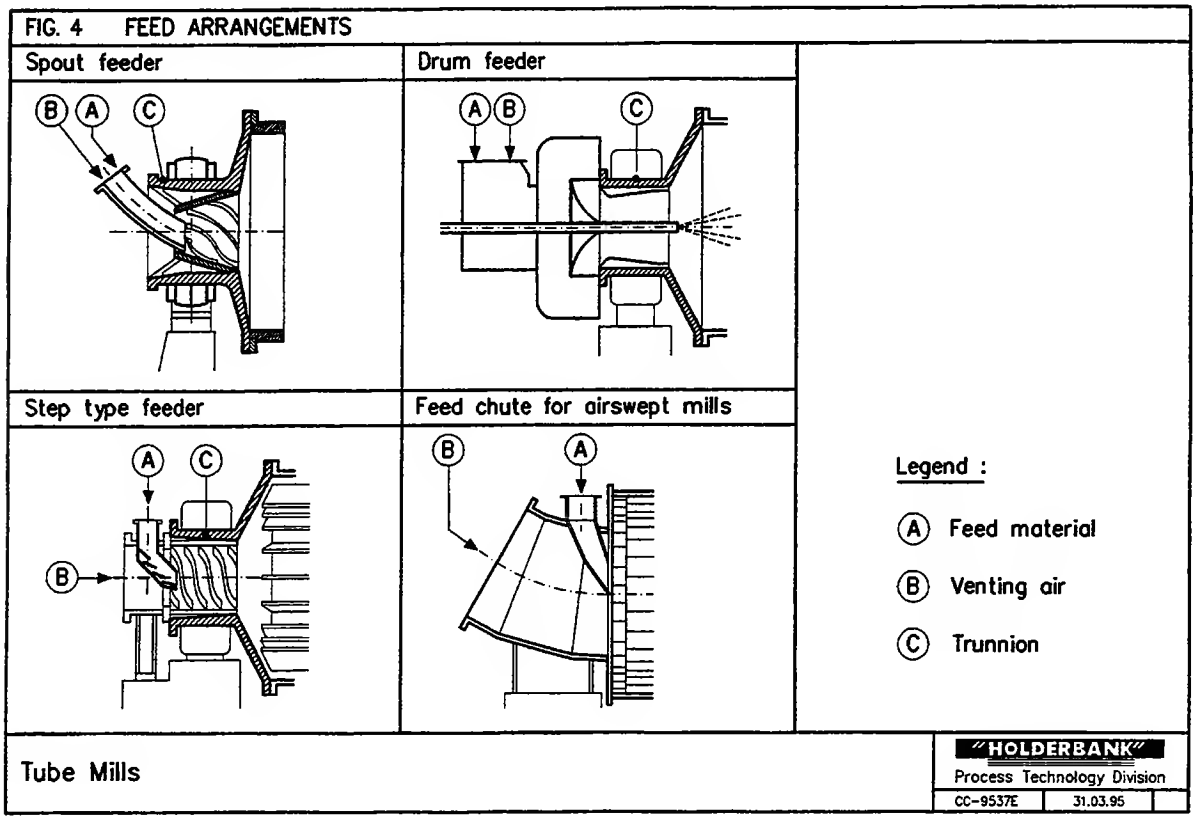
Figure 4 Feed Arrangements



5.2 Discharge arrangements

5.2.1 Centre discharge

The centre discharge is located between the coarse and the fine grinding compartment of a tube mill (Fig. 5). The mill is fed from both sides through its hollow trunnions. The material leaving the coarse and fine grinding compartment is led together through a elevator to a separator.

The separator fines are the final product. The coarses are transported back to the fine, or to both grinding compartments.

At the discharge itselve the material leaves the mill through outlet holes in the mill shell. The material is then collected by a casing which is located around the mill shell.

The ventilating air also leaves the mill through the outlet holes.

5.2.2 End discharge

The material leaves the mill through the hollow trunnion at the mill discharge side.

The outlet casing includes a sealing which is located between the rotating and static parts. A steady under-pressure prevents the escape of dust.

Inside this outlet casing the first separation of venting air and ground material takes place.

5.2.3 Discharge of airswept mills

The ground material leaving an airswept mill is transported only by gas. Therefore the discharge arrangement must be in a way that the gasflow is not hindered and causes no excessive pressure losses and no accumulation of material takes place.

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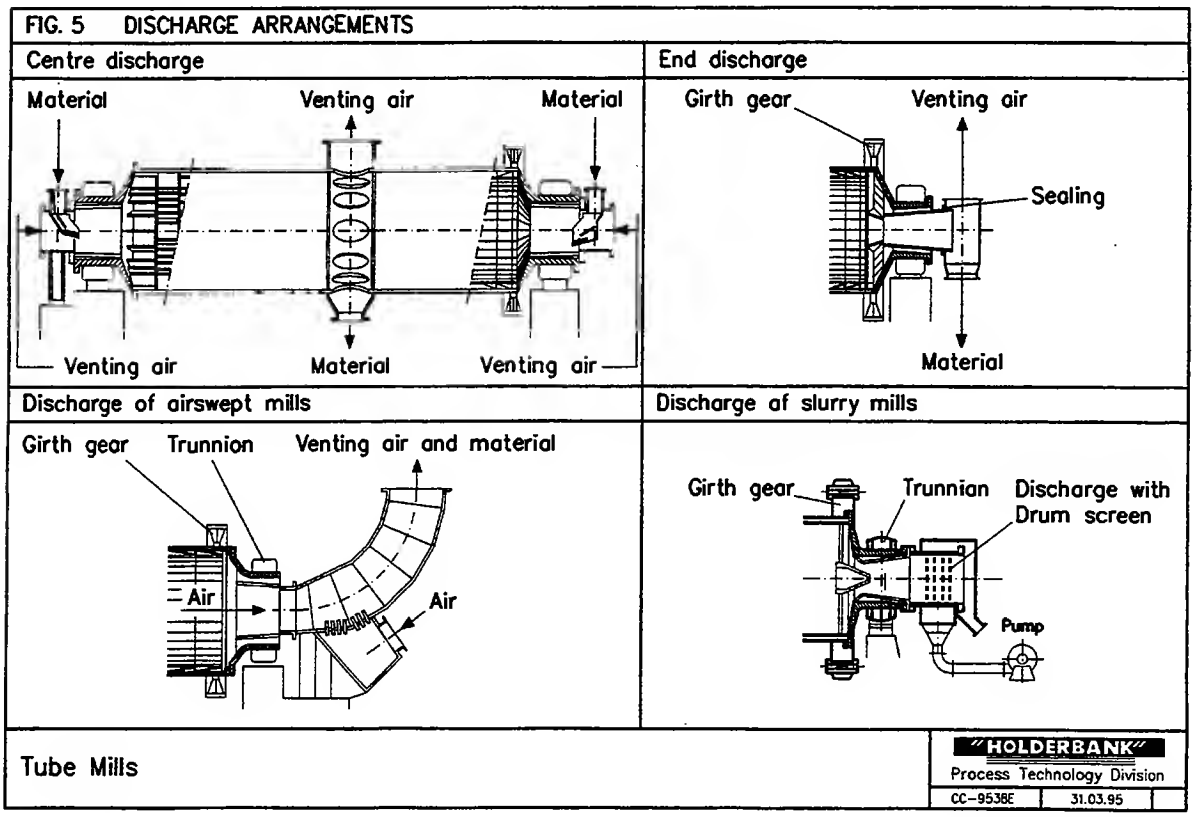
5.2.4 Discharge of slurry mills

The most slurry mill discharge arrangements are equipped with a drum screen. Slurry mills are installed in open and cloased circuits. The rejects (nips) are led back to the mill inlet in case of a cloased circuit grinding plant.

Figure 5 Discharge Arrangements



5.3 Mill shell

The mill shell consists of several sections of welded plates.

The thickness of the shell is determined by the weight of the material, the grinding media and the rotating parts and the design of the mill. The shell thickness of long mills is graded, i.e. the thickness increases from both ends towards the mill centre.

The stress distribution within the mill shell can roughly be estimated by using a simplified load distribution model, where the total rotating weight presents the sum of the material, the grinding media and the rotating parts.

The stresses are maximum at the centre of the shell and minimum at the ends.

For the listed shell material qualities the maximum admissible bending stresses for the different types of mills can be found in Figure 6.

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5.4 Mill heads

Tube mills supported with trunnion bearings are provided with mill heads. The heads are bolted or welded to the mill shell.

Small mills are often equipped with one piece mill heads (integral heads) where the trunnion and the mill head are cast as one piece. They are bolted to the flange of the mill shell as shown in Figure 7-1.

Trunnions can also be bolted to the mill heads as indicated in Figure 7-2. The mill head in this example is directly welded onto the mill shell.

The mill heads of large mills are mostly divided concentrically into an outer and an inner part (Figure 7-3).
The inner conical part of the head can either be cast together with the trunnion or it is jointed to each other by welding.
The outer part is bolted to the inner part and welded to the mill shell.

In order to entirely relieve the mill heads of large mills of the elevated bearing reaction forces, there is a certain trend to use slide shoe bearings in conjunction with a riding ring mounted to the mill shell. The mill head consists in this case of a welded construction of mild steel plates (Figure 7-4).

The mill heads can be cast from modular cast iron. However a cast steel with a low sulphur content is often preferred to permit a proper welding of the mill heads to the mill shell.

Figure 6 Mill Shell

Mill shell Material	– Boiler plates HI – St 37-3 (DIN 17155) with low sulphur content – ASTM A 283 grade C with carbon content < 0,2% (USA)

Type of mill	Max. admissible bending stress (N/mm^2)	
	With trunnion bearings	With slide shoe bearings
End discharge		
Mill ø < 4,2m	16 – 18	
Mill ø < 4,2m	12 – 14	9 – 10
Centre discharge	6 – 8	

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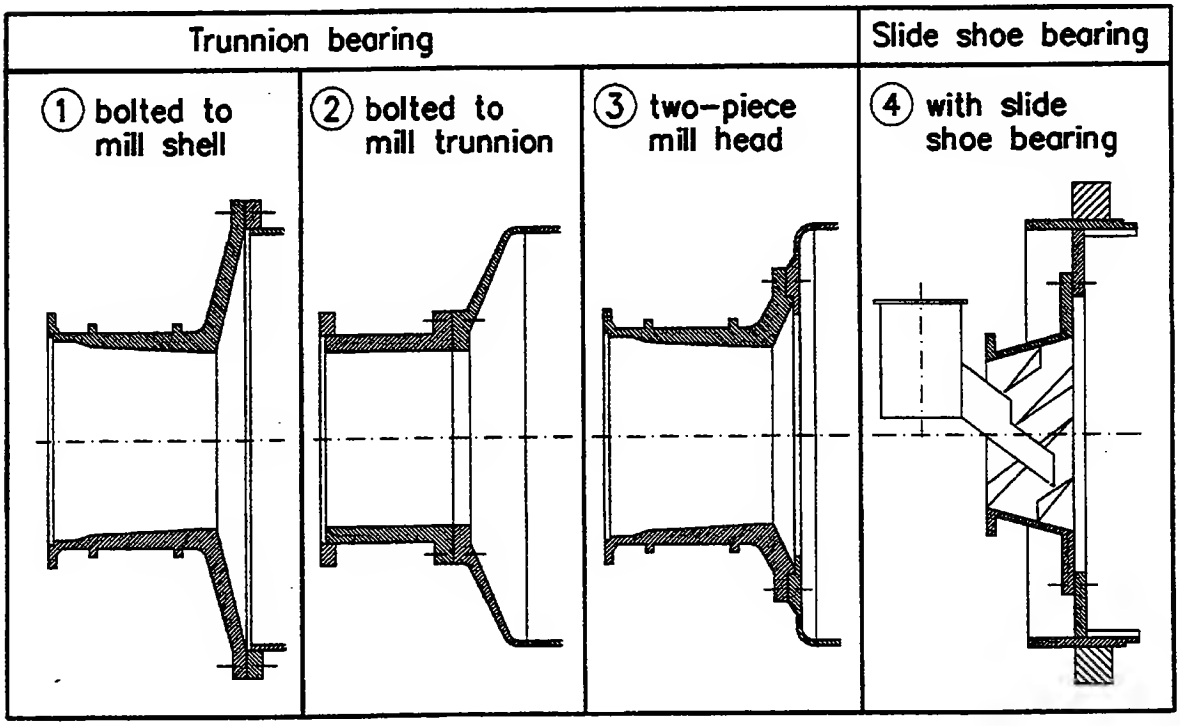
Figure 7 Mill Heads



5.5 Mill bearings

5.5.1 Trunnion bearing

The hollow trunnions of tube mills are supported by trunnion bearings (Fig. 8). The bearing comprises a welded or cast casing with the bearing liner, the supporting insert mounted on spherical surfaces, the white metal lined bearing liner and the lubricating and cooling equipment.

Trunnion bearings are hydrodynamically lubricated and support approximately 30 - 50 % of the mill head trunnion. During the starting and shut-down period high pressure pumps float the bearings (hydrostatic lubrication).

This is because there is a critical zone between the mill end and the trunnion where severe stresses occur. The fabrication of the mill ends requires special precaution to ensure sound component quality with an extended life.

5.5.2 Slide shoe bearing

The slide ring (or riding ring) rests on a set of self adjusting sliding shoes as shown in Figure 8. Hydrostatic lubrication is provided for the start-up phase and hydrodynamic lubrication during operation.

The main advantages of slide shoe bearings as compared with trunnion bearings are:

- no limitation in size and capacity of mill
- no delicate large size mill head castings are necessary
- simple design of wear plates
- simplification of feed and discharge devices
- large feed and discharge openings allowing the passage of considerable amounts of gas

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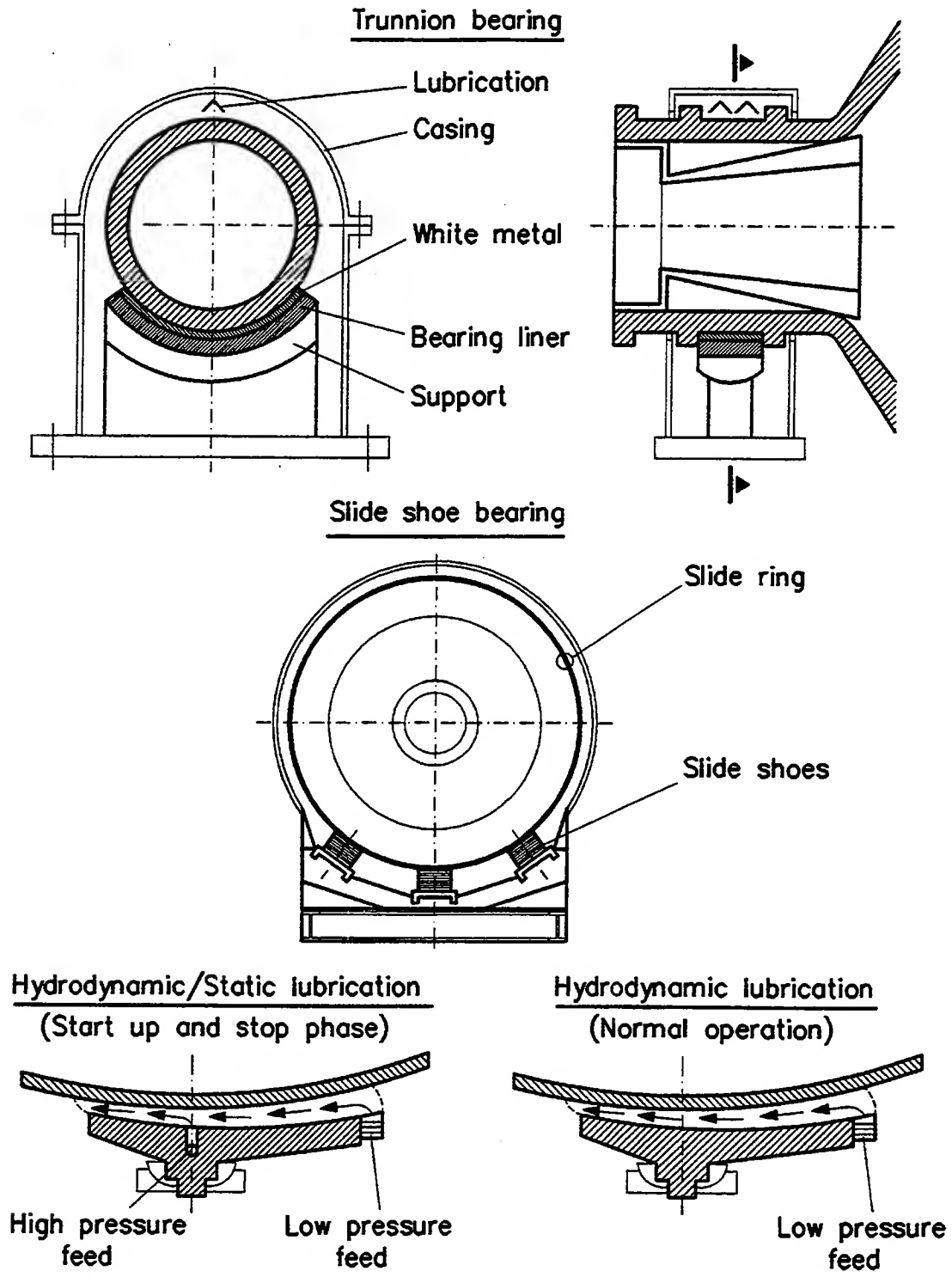
- no limitation in gas temperature influencing expansion of trunnion and trunnion bearing

Figure 8 Mill Bearings



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5.6 Mill drives

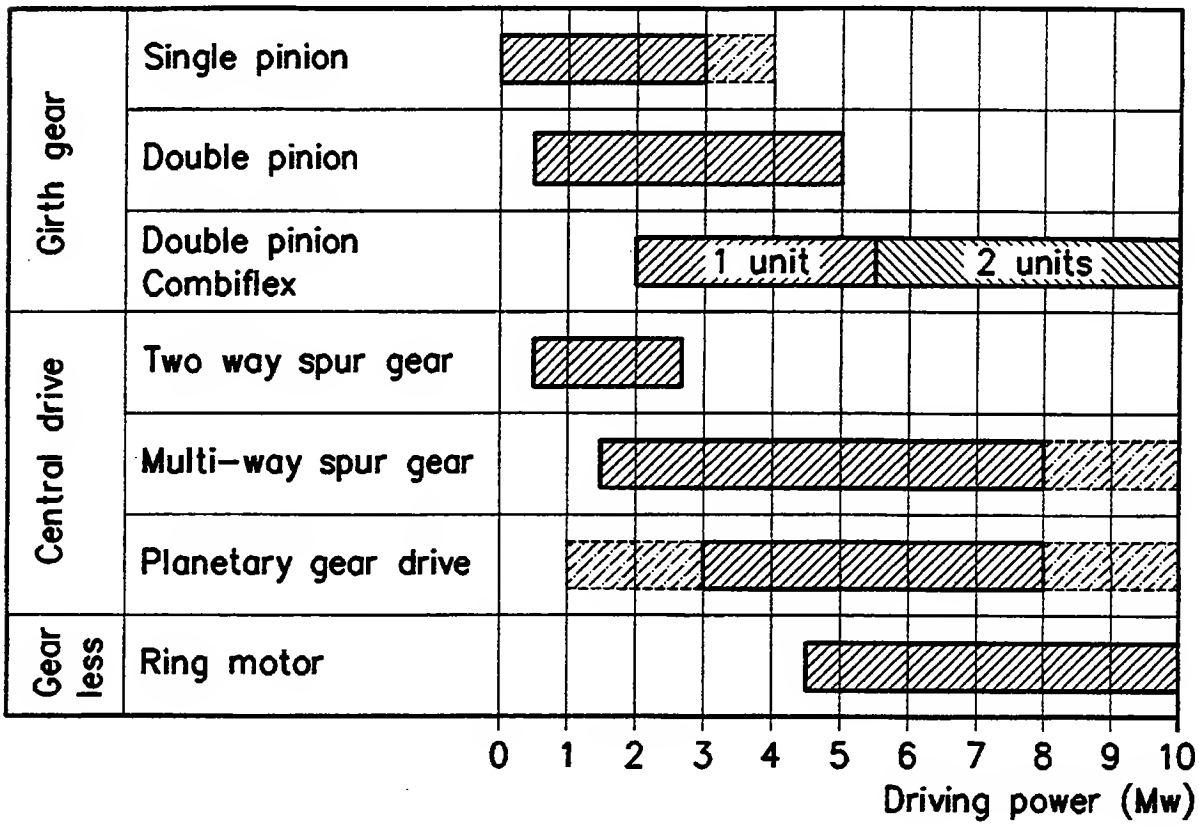
5.6.1 General

The mill drives can basically be subdivided into three groups:

- girth gear / pinion drives
- central drives
- gearless drives

Each type can be applied for a certain range of power as indicated in Figure 9.

Figure 9 Power Ranges of Mill Drives



5.6.2 Girth gear / Pinion drives

The girth gear (or spur rim) is bolted to the mill body and is driven either by one or two pinions as shown in Figure 10 a . The reduction gear box arranged between pinion and mill motor is equipped with an auxiliary drive. The auxiliary drive allows slow rotation of the mill (0.2 - 0.3 min^{-1}) for inspection or to turn the mill to a certain position.

Girth gear drives are often used for mills which are designed for high drying capacities which due to the large required outlet cross section do not permit the use of central drives.

The girth gear is enclosed in an oil-tight casing and is spray or splash lubricated.

Girth gears are made from Cr-Mo heat treatable cast steel hardened to max. 900 N/mm^2 tensile strength. The pinions are generally made from Cr-Ni-Mo heat treatable steel hardened to max. 1150 n/mm^2 tensile strength.

US suppliers often prefer a low speed synchronous motor instead of a reduction gear.

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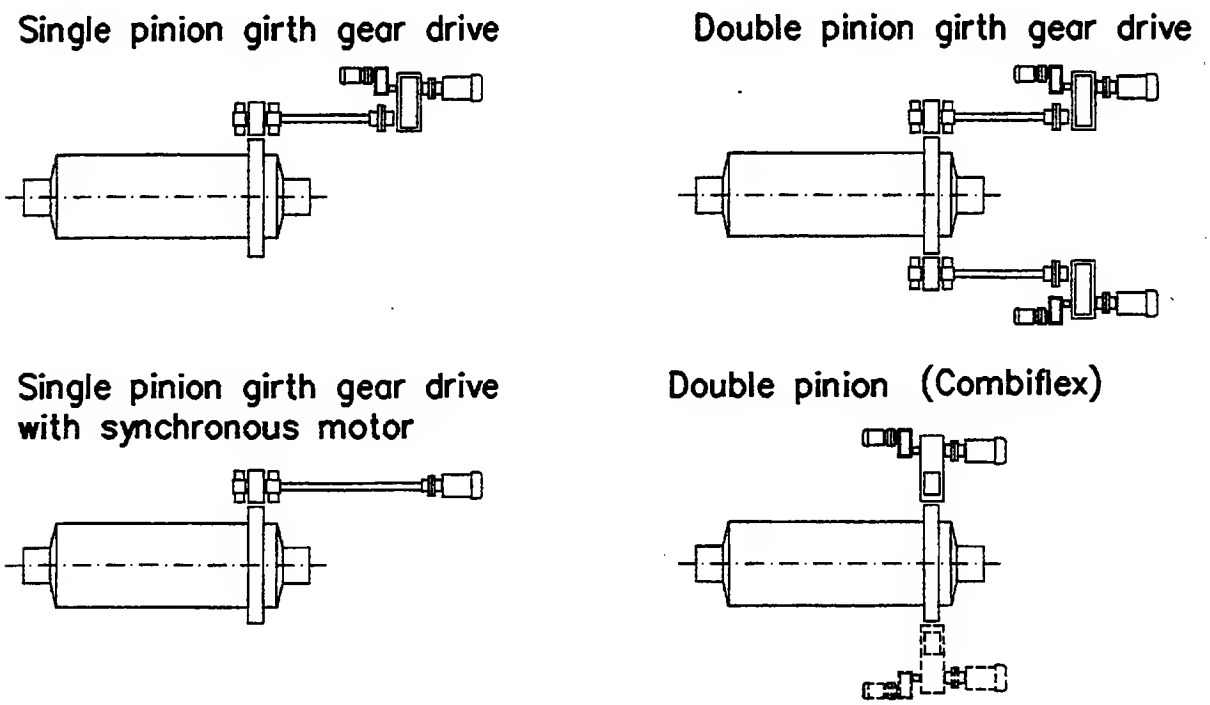
Double pinion drives can be used where a driving power of up to 5000 kW needs to be transmitted. The Combiflex presents a good solution for driving powers above 5000 kW.

One Combiflex gear consists already of two pinions. If two units are installed, four pinions transmit the forces to the girth gear. This design allows to install driving powers above 10'000 kW. A further characteristic of the Combiflex is that the girth gear and the slide shoe ring are located next to each other and covered by the same casing.

Figure 10a Types of Mill Drives



5.6.3 Central drives

The gear unit of a central drive can either be coupled by a hollow shaft, by a torsion shaft or directly to the mill. The gear is operating in an oil-bath in a completely enclosed casing.

Smaller existing mills with input powers up to 3'000 kW are equipped with a two way spur gear with either one or two motors, Figure 10b.

For larger driving power either a multi-way spur gear or a planetary gear could be used.

Multi-way spur gears (e.g. FLS "Symetro" gear or Flender "Polyflex" gear) are manufactured in sizes up to 10'000 kW.

Single motor planetary gears are built up to 8'500 kW. For higher power ratings up to 10'000 kW dual planetary gears are available. The planetary gears are generally more expensive than multi-way spur gears, offer however a very high gear efficiency and are of a very compact design.

The planetary gear drive where the gearbox is flanged to the mill (MAAG) has no direct contact with the mill foundation, i.e. mill movements or any lowering of its foundation have no effect on the gearbox teeth. This planetary gearbox requires only a little support for the torque arm.

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5.6.4 Gearless drives

Large tube mills can be driven by low frequency electrical motors; the so-called ring motors or wrap-around motors. The ring motor is directly arranged around the mill shell; i.e. there is no speed reducer required.

For the required low mill speed of 13 - 15 min^{-1} a speed adjustable AC drive with a frequency of 5-6 Hz is used.

The motor can be supported by an extended mill head or it can be arranged on the mill shell according to Figure 10c.

The ring motor can be accelerated from standstill up to operation speed. For a limited time the mill can be turned with a very low speed to allow a positioning of the manholes in the mill shell.

The main advantages of the ring motor as compared with conventional mill drives are:

- adjustable mill speed
- no wearing part on gearless mill drive
- no costly lubrication required
- low mechanical maintenance costs
- no limitations for larger tube mills
- more flexible layout of mill room

The disadvantages include mainly:

- low efficiency (91 - 92 %)
- higher investment cost (for small units)
- extra load (15 %) on mill bearing due to weight of motor
- sensitive for electrical failures

Figure 10b Types of Mill Drives (Central Drives)

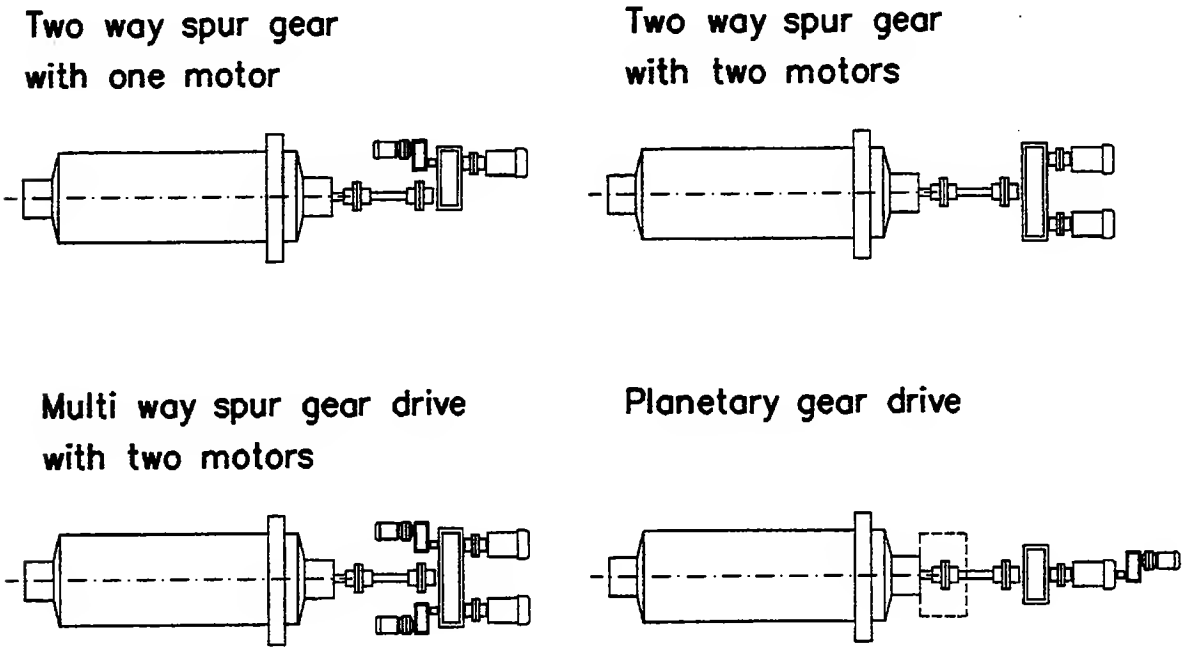
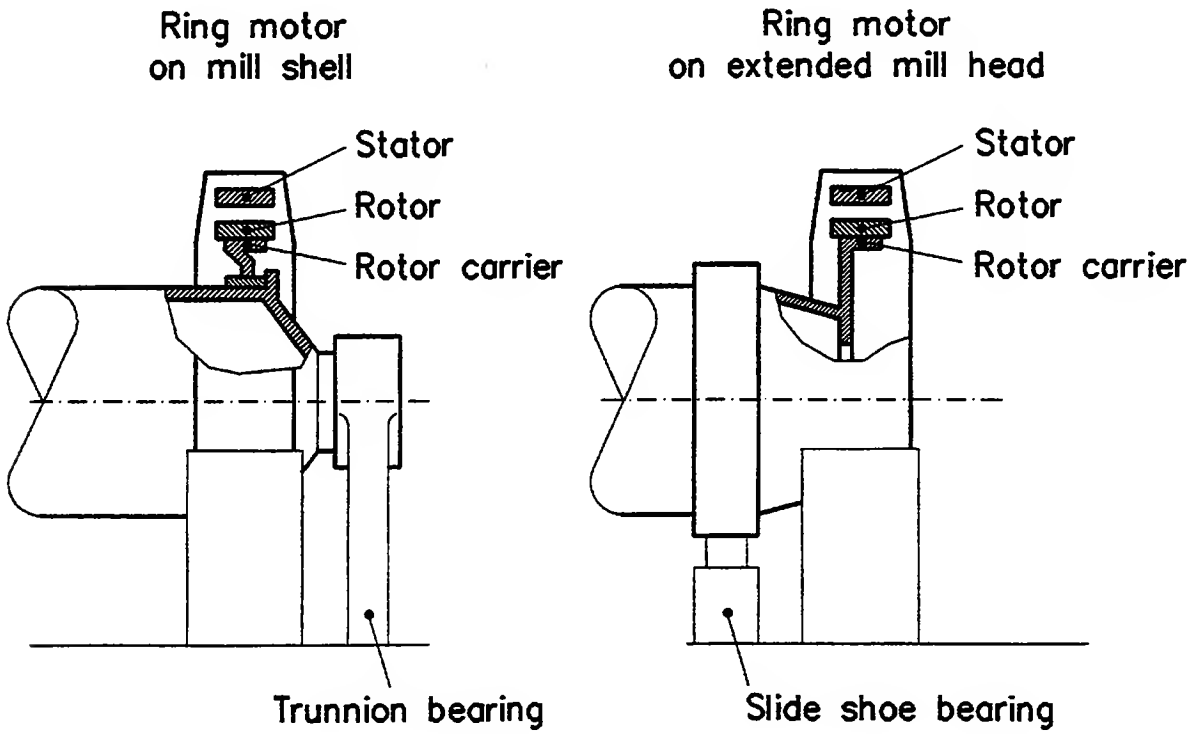


Figure 10c Types of Mill Drives (Gearless Drives)



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5.6.5 Selection criteria for mill drives

For the proper selection of the type of mill drive following facts have to be considered:

- investment costs
- operation costs (e.g. gear efficiency)
- maintenance costs (e.g. lubrication)
- process requirements (e.g. mill discharge design)
- availability and reliability
- space requirement

The most important features of the various types of mill drives are summarized in Figure 11.

The latest tube mill installations show a trend toward the application of either double pinion Combiflex drives or central drives with a planetary gear.

A comparison of these two mill drives shows that the planeatary gear drive has a good drive efficency but also high investment costs whereas the efficency of the Combiflex drive is low at lower investment costs.

Both mill drives permit the installation of rather high driving powers.

Figure 11 Selection Criteria for Mill Drives

FIG. 11 SELECTION CRITERIA FOR MILL DRIVES

Legend :
L Low
M Medium
H High
1) Mech. and electr. efficiency
2) Drive inclusive tube mill
3) Electrical energy and lubricatian
4) Man power and spare parts

Type of mill drives		Range of power (MW)	Drive efficiency (%) 500–2500 kW	Drive efficiency (%) > 2500 kW	Investment costs 2)	Operation costs 3)	Maintenance costs 4)	Space requirement	Remarks
Girth gear	Single pinion	0,1 – 4	93	93,5	L	H	H	M	Small mills only
	Double pinion	1 – 5		92,5	L	H	H	M	Medium size mills with high drying capacity
	Double pinion (Combiflex)	2 – 10	92	92,5	L–M	H	L	L	Large mills with high drying capacity
Central drive	Two–way spur	1 – 3			M	M	M	M	Small mills only
	Multi–way spur	1,5 – 10	93	93,5	M	M	M	H	Not for mills with high drying capacity
	Planetary gear drive	3 – 10	94	94,5	H	L	L	L	
Gear less	Ring motor	< 4,5		92	H	H	L–M	L	Variable mill speed possible

Tube Mills

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6. INTERNAL ELEMENTS

6.1 Overview

6.1.1 Liners

The internal parts of a tube mill are protected by means of various types of wear resistance liners as can be seen in Figure 12. They have to assist the grinding function in the respective grinding compartment.

Liners need to have resistivity against impact forces (deformation, breakage), friction and corrosion.

Dry grinding mills are equipped with steel liners whereas wet grinding mills can alternatively be equipped with rubber liners.

6.1.2 Diaphragm

Diaphragms are installed to separate the grinding compartments. They have to be selected in such a way as to obtain the required retention time of the material in the former grinding compartment as well as to allow a sufficient flow of venting air or drying gases.

6.1.3 Grinding media

The sizes of the grinding media as well as the material quality are determined according to:

- material feed size
- product fineness
- mill diameter
- mill length
- grindability
- mineralogical structure of the material

6.1.4 Retention rings

Cement mills in open circuits are often equipped with retention rings. The installed rings increase the rotating surface. More friction within the ball charge should result.

However an increase in grinding capacity has never been demonstrated.

Initiated further trials should show the effect of retention rings in one compartment slurry mills. At present time results are not available.

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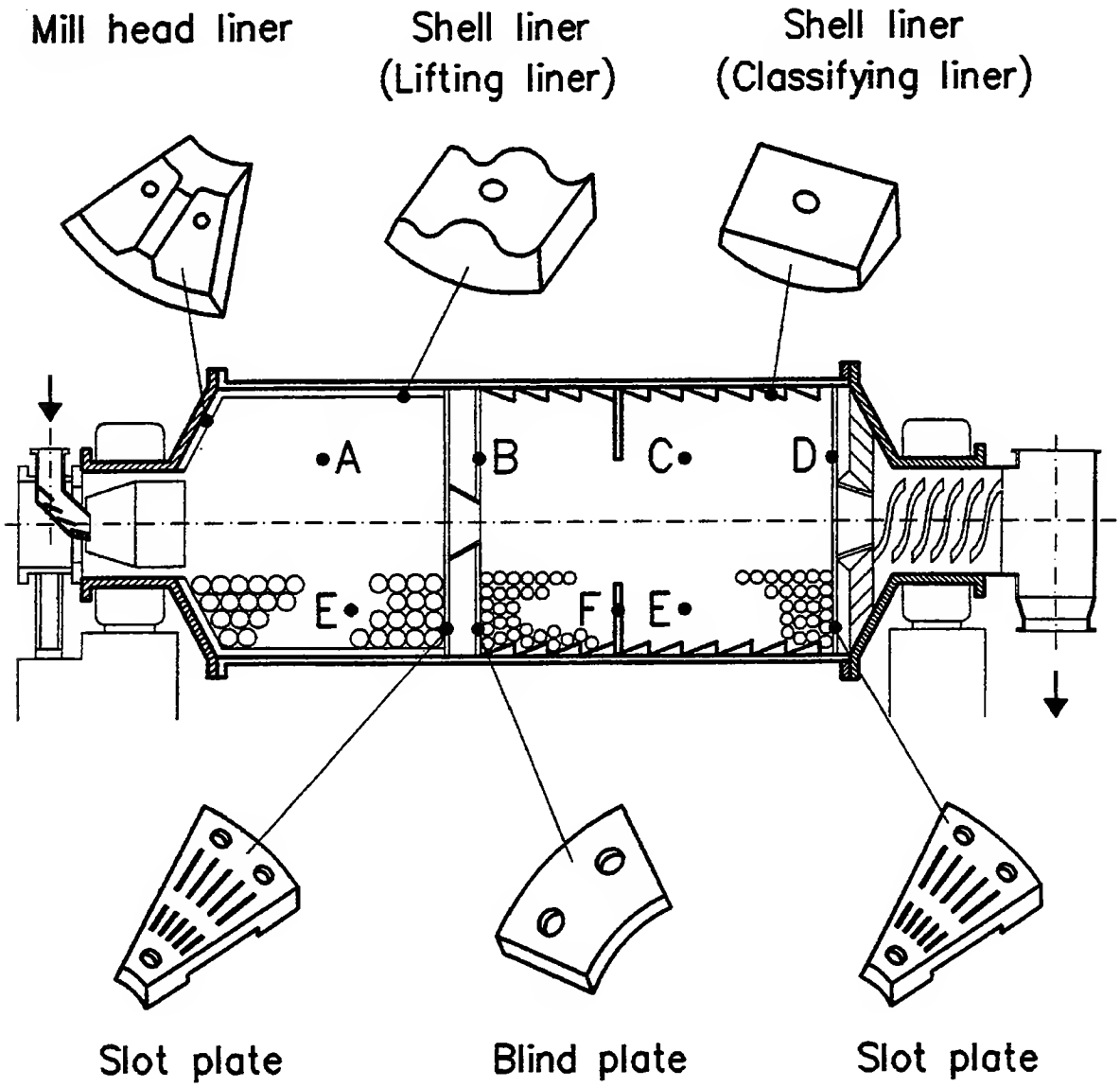
Figure 12 Overview Internal Elements



A Coarse grinding compartment
B Intermediate diaphragm
C Fine grinding compartment
D Discharge diaphragm
E Grinding media
F Retention ring

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6.2 Head liners

Tube mill heads are built conical (with trunnion bearings) or even (with slide shoe bearings) as can be seen in Figure 13. Both types of mill head liners are exposed to high friction wear due to the relative movement between the liners and the grinding media. The liners of a conical mill head are additionally exposed to impact forces caused by the falling grinding media.

The wear ratio is not even across the liner rings and finds its maximum in the middle section of the liners.

Some mill head liners are designed with a radial rib in order to lessen the relative movement between the liners and the grinding media.

6.3 Shell liners

The following table shows the appearance forms of wear within the tube mills and the consequent requirement to the liner material.

Table

	Chamber 1 Coarse grinding	Chamber2 Fine grinding
Wear appearance within the tube mill		
Impact wear	High	Low
Friction wear	Low	High
Requirements to Liner Material		
Hardness	Low	High
Notch impact strength	High	Low

The selected material should further fulfill the following requirements:

- Homogenous hardening and homogenous structure composition throughout
- Low wear rate, no deformation and no breakage

Today's common material qualities, their composition and hardness and their field of application can be found in Figure 14.

High chromium alloy cast liners are the mostly used liner type due to their relatively good wear behavour.

The hardness of the shell liners for the coarse grinding compartment is in general somewhat lower than the hardness of the respective grinding balls.

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Figure 13 Head Liners

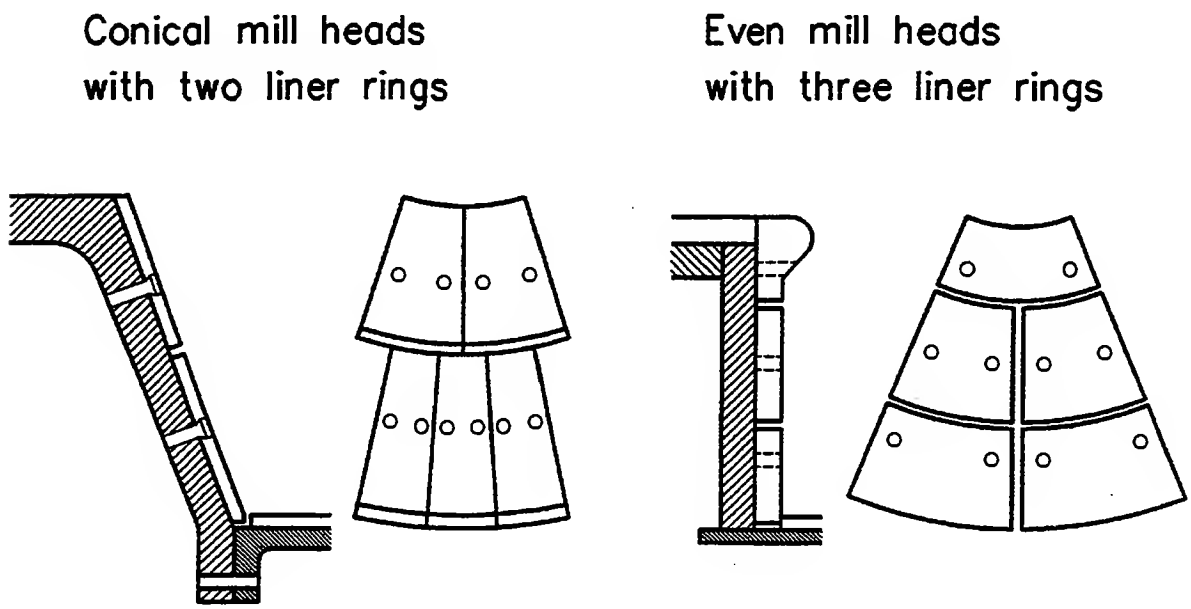

Figure 14 Material Quality and Application of Shell Liners

Material	Cem.comp.	Hardness (HRC)	Remarks
Manganese steel	12–14% Mn	40	Large mills deformation of liners Recommended for small mills (Inicial surface hardness 20–25 HRC)
Low chromium alloy cast liners	2–3% Cr	40–42	Less deformation than manganese liners Suitable for big and small mills
High chromium alloy cast liners	12–15% Cr	50–55	Most used material Suitable for big and small mills (Notch impact strength 4–10 J/cm^2)

The exact measurements of the liners depend on the mill diameter. There is a norm (DIN 24111) describing the liners' length and width. Figure 15.

There are still many tube mills in the United States where the norm has not been considered yet.

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It is the overall target of each liner design to reach:

- **the lowest spec. energy consumption**
- **the highest production capacity of the tube mill**

with the lowest possible specific costs for liners

6.3.1 Lifting liners

Lifting liners are applied in the coarse grinding compartment and in older mills, also in the fine grinding compartment. In the coarse grinding compartment the liners have to lift and release the grinding media in such a way that the impact will be strong enough to break the larger particles. The grinding media should not be overlifted, since some of them would fall on the bare liner and thus ball breakage and increased wear of liner and grinding media occurs.

In Figure 16 some of the most known types of lifting liners are shown.

Figure 15 DIN 24111

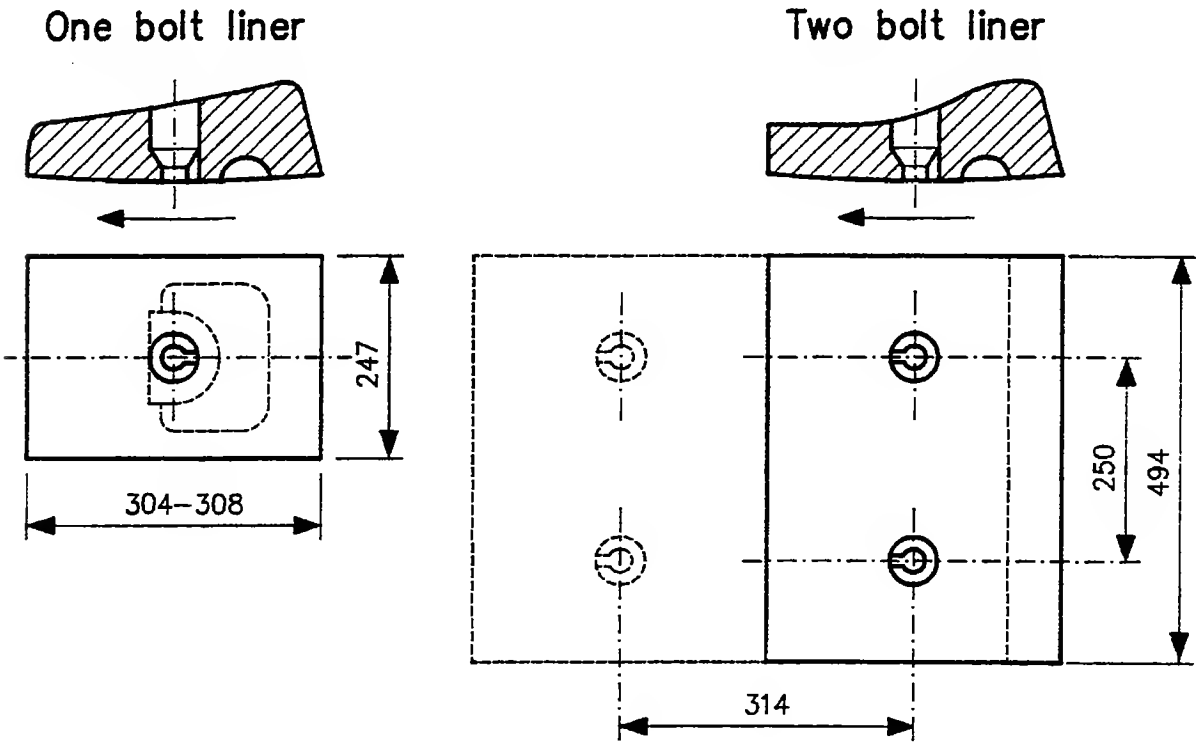

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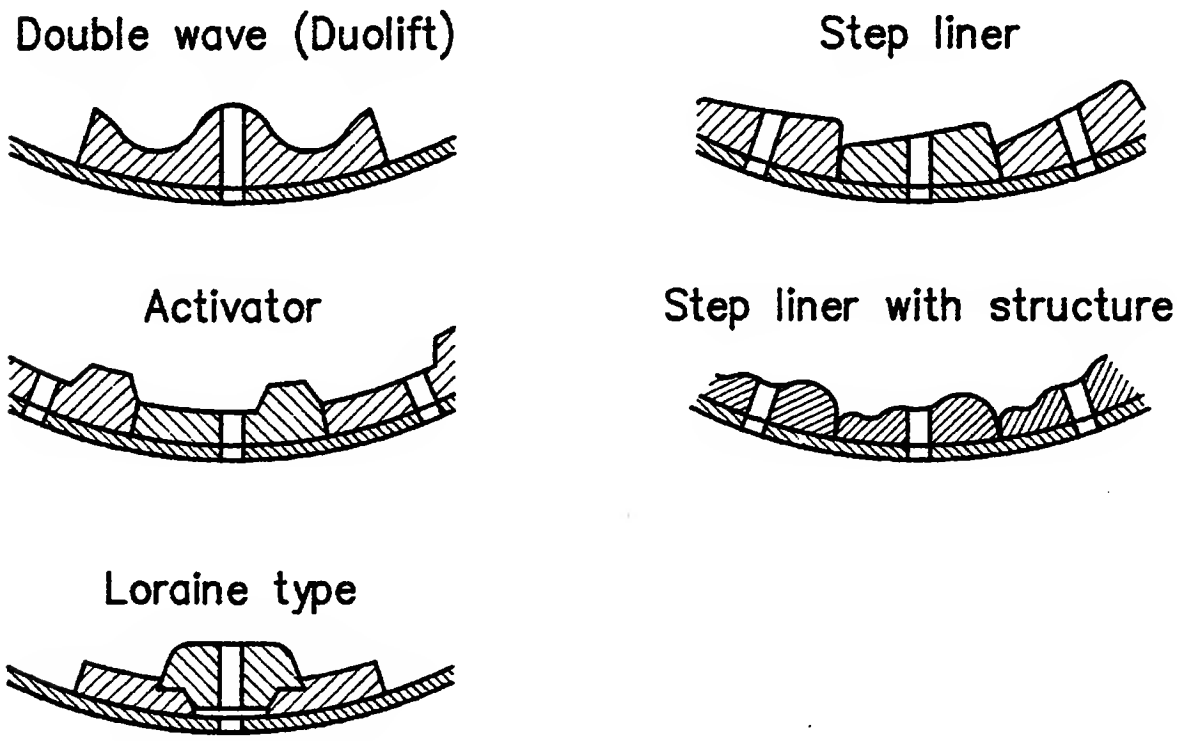
Figure 16 Lifting Liners



6.3.2 Classifying lining

The fineness of the material increases towards the discharge of the fine grinding compartment due to the ongoing grinding process.

At the inlet of the fine grinding compartment the 'coarse' material requires larger balls for an efficient size reduction. Less friction but more impact forces are needed.

Toward the outlet of the fine grinding compartment smaller balls are required for the production of surface (friction).

Therefore, fine grinding compartments are often equipped with classifying liners. These liners automatically segregate the grinding media; i.e. the larger balls to the inlet end of the compartment with a progressive decrease in the ball size towards the outlet of the mill. The basic principle is shown in Figure 17.

The quantity and shape of classifying liners depend on the size of the mill.

Often a combination of different liners can be found within the fine grinding compartment e.g. at the inlet lifting and classifying liners and at the discharge where less impact forces are needed only classifying liners.

6.3.3 Lifting and classifying liners (combined)

Airswept mills consist of only one compartment. Since some of the feed material is rather coarse the grinding media consists also of bigger balls up to 80 mm. A normal classifying lining does not match to this ball size.

The lifting and classifying lining lifts the grinding media sufficient in order to grind also coarse material by impact forces. Due to the classifying function they also prevent the reverse classification of the ball charge.

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Airswept mills are equipped with lifting and classifying lining over the whole compartment length or only in the first third. The remaining two thirds of the compartment are then lined with regular classifying liners.

Figure 17 Classifying Lining

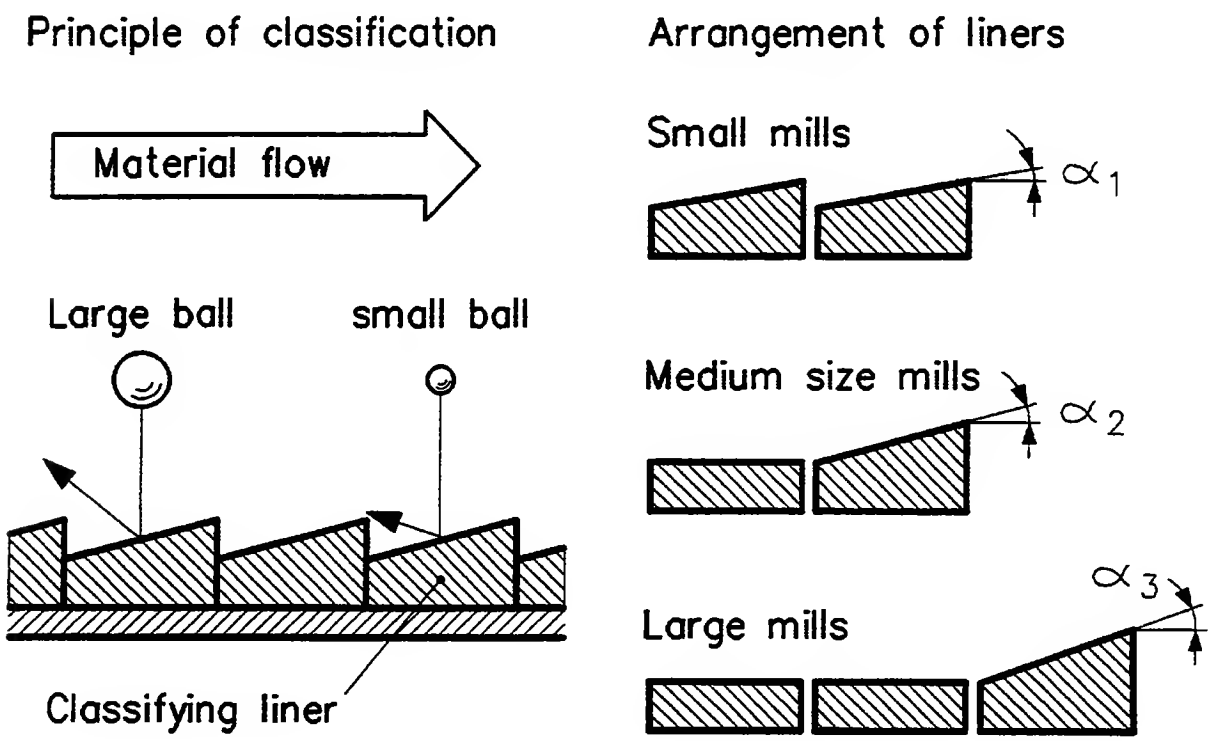

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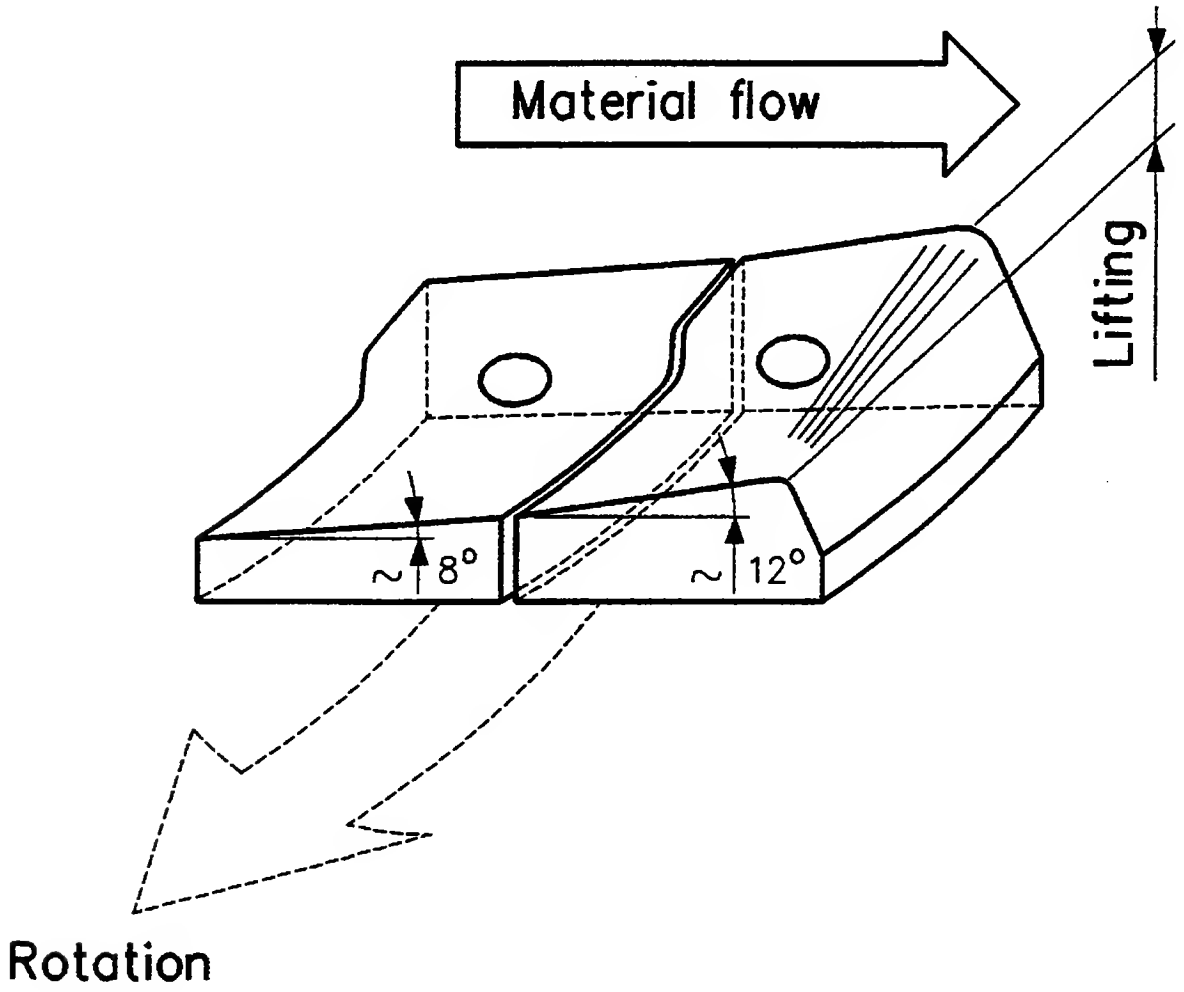
Figure 18 Lifting and Classifying Liners



6.3.4 Other liner types

6.3.4.1 Dragpeb

FLS supplies compartment 2 of its tube mills with either classifying lining or the so-called Dragpeb lining. The philosophy of this type of liners (Fig 19) is the maximum utilization of the mill diameter.

The mill capacity changes exponential with the mill diameter $D^{2.5}$. Therefore, the Dragpeb liners are very thin (~ 40 mm high) to have the largest possible internal diameter.

The Dragpeb liners do not classify the ball charge. For this reason the advantage of a classified ball charge as described in chapter 6.3.2 can not be applied.

6.3.4.2 Grooved liners

The philosophy of this liner type is to maximize the contact zone between the outer layer of the ball charge and the liners.

Figure 19 shows that the material between the liners and the balls can not escape due to the grooved liner shape.

There are also grooved liners available which have a classifying effect. The expected better efficiency has never been proven in direct comparisons to other mill liners.

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6.3.5 Fastening and installation of liners

The mill shell lining can be divided according to the method of fastening into three groups:

- bolted mill lining
- bolt saving mill lining
- boltless mill lining

The boltless or bolt saving lining has compared to the conventional bolted liner the following advantages:

- longer lifetime of liner due to elimination of fasting hole where wear generally starts
- elimination of mill shut-downs due to bolt breakages
- no leakage in mill shell due to untight fastening

Bolted mill lining

The bolted mill liners are fastened to the mill shell either by one or by two bolts depending on the plate size.

Bolt saving mill lining

To decrease the number of bolts also a bolt saving fastening according to Figure 20 can be applied. The bolted liners are located in an angle of 90 ° to each other.

Boltless lining

The boltless lining is self-supported. First the lower half of the mill is lined and the endliners braced with jacks; the mill is then rotated to facilitate lining of the other half of the mill. The installation should always be supervised by the supplier.

Figure 19 Other Liner Types

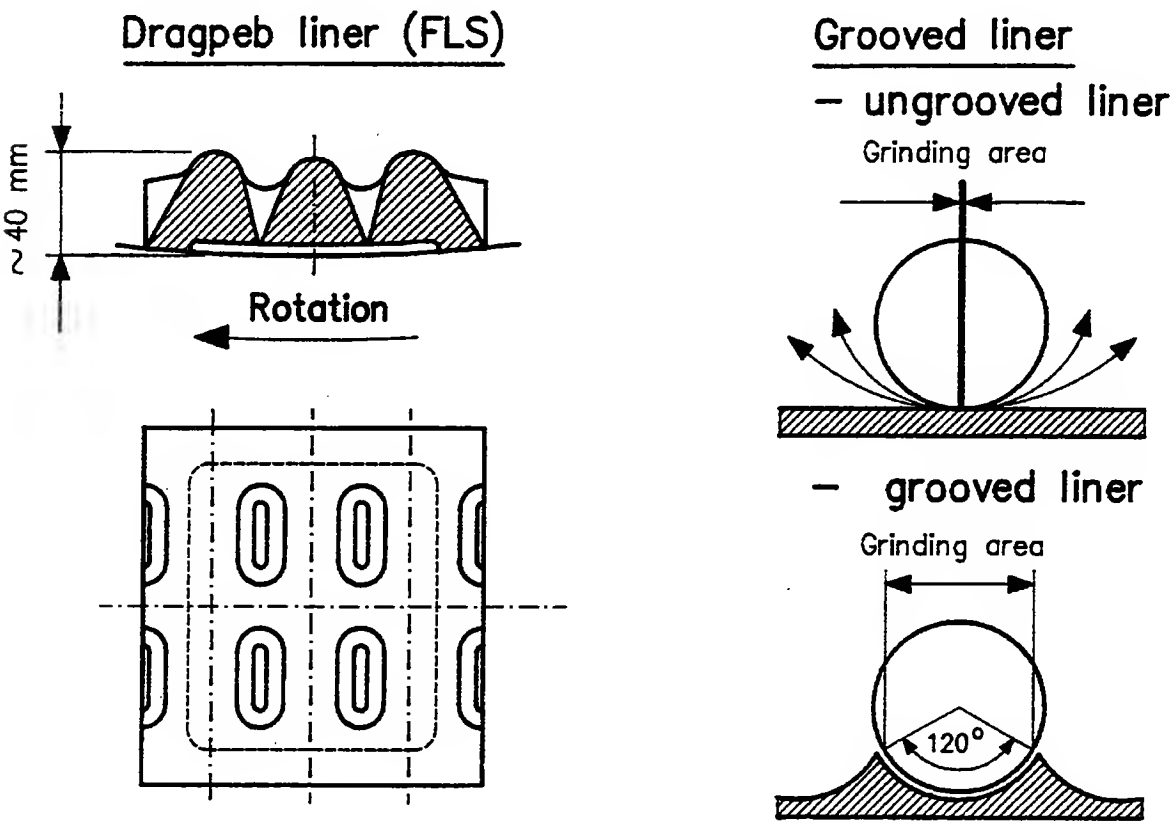

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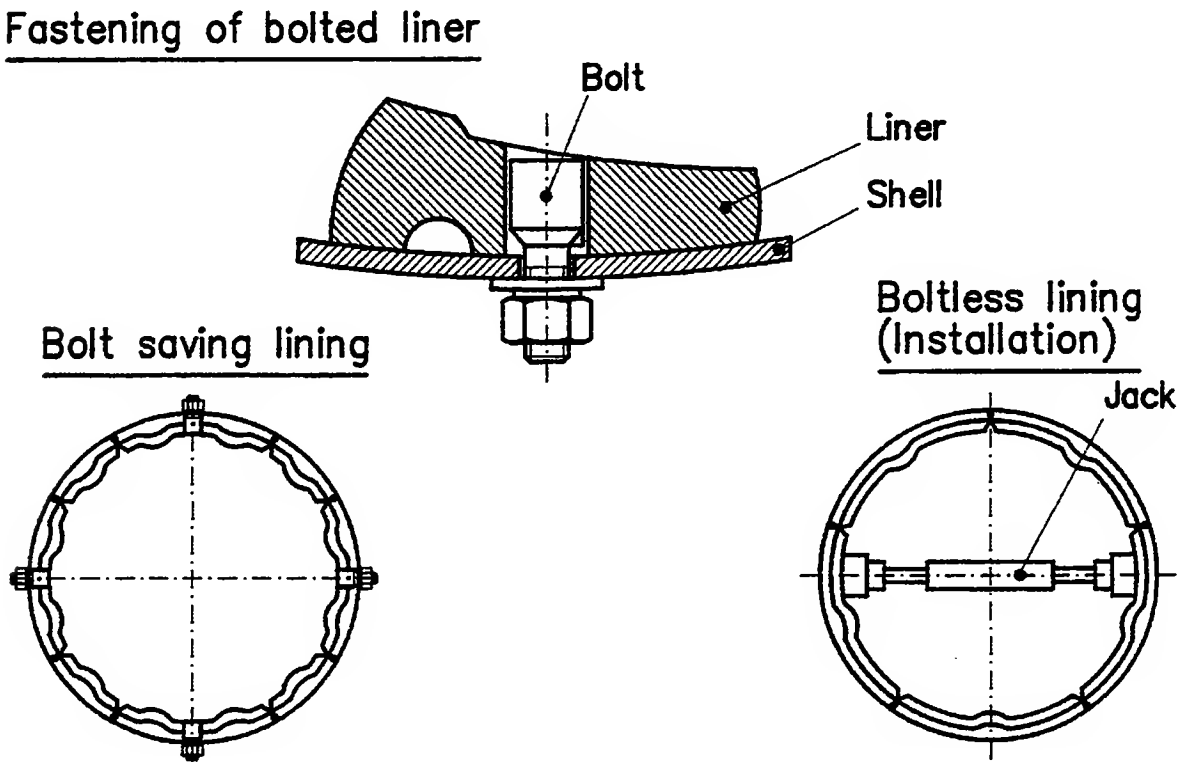
Figure 20 Fastening and Installation of Liners



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6.3.5.1 Life times of shell liners

High chromium alloyed cast steel is mostly used for the fabrication of the shell liners. The life time of this typical wear part depends on the type of material being ground.

Some ranges of the expected life times for high chromium liners can be found in Figure 21.

Some of the additives such as pozzolana or slag can cause high wear. They often are not only very abrasive but also consist of high moisture which increases the wear rates due to corrosion.

6.3.6 Rubber liners

Rubber lining is applied in wet grinding mills.

A comparison to steel liners shows the advantages and disadvantages of such a lining:

- Advantages:
  - low noise
  - less liner weight
  - easier installation
  - good abrasion resistance
  - more economical in combination with small balls due to less liner wear costs at smaller ball sizes as can be seen in Figure 22
- Disadvantage:
  - limited ball size (excessive increase of wear rate with bigger balls)
  - bigger liners -> less volume -> less grinding capacity
  - maximum operation temperature of 80 ° C (no dry grinding possible).

Figure 21 LifeTime Of Shell Liners

Material	Hours of life time	
	Coarse grinding compartment	Fine grinding compartment
Raw meal (12–14% R 90 µm 2–4% free quartz)	30'000–40'000	60'000–80'000
Portland cement (3'000 cm^2/g)	25'000–30'000	50'000–60'000
Portland cement with minor additives	23'000–27'000	46'000–54'000
Composite cement	15'000–17'000	30'000–34'000

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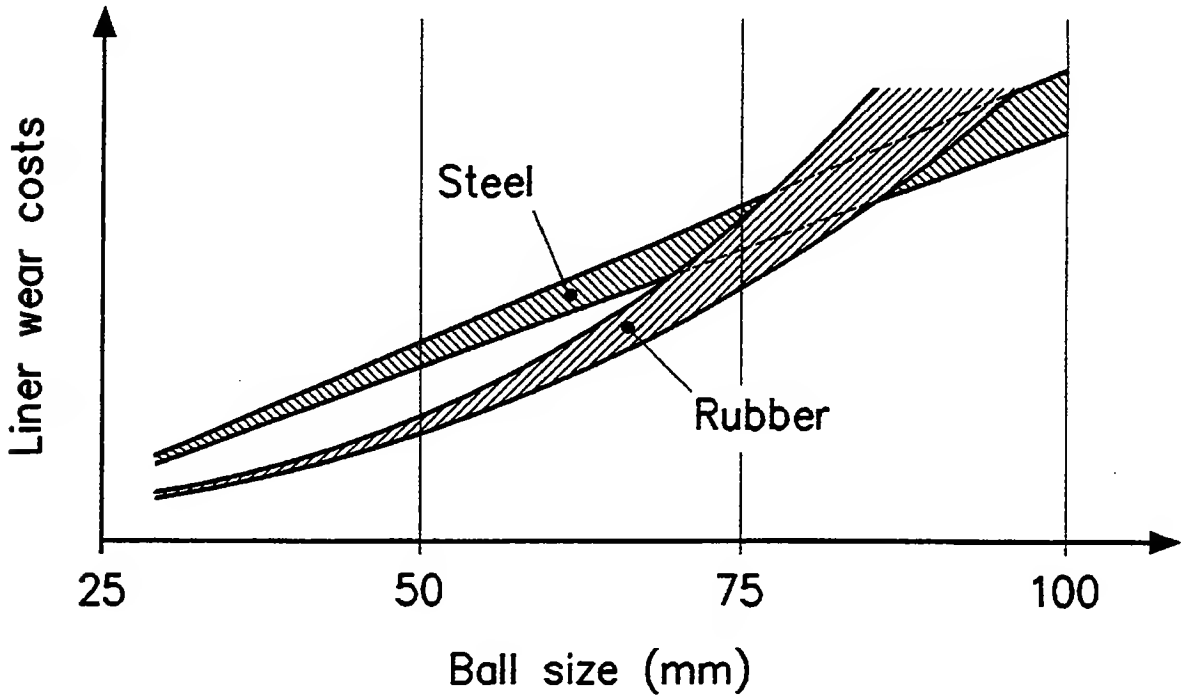
Figure 22 Rubber Liners



6.4 Intermediate diaphragm

They consist of the following elements (double diaphragm):

- slotted plates
- centre screen
- supporting structure
- blind plates
- lifters

Diaphragms can be located at following places:

- between a drying and grinding compartment
- between a coarse and fine grinding compartment
- at the mill discharge of either end or centre discharge mills

6.4.1 Function

The intermediate diaphragm subdivides the mill in a coarse grinding and fine grinding compartment. In order to maintain the grinding efficiency along the length of the mill a graduation of the grinding media is required.

The material ground in the first compartment passes through the slots of the intermediate diaphragm and enters the next grinding compartment.

Adjustable diaphragms permit to control the material flow. They keep as much material in the compartment as required for the highest grinding efficiency. Figure 23 shows the working principle of an adjustable diaphragm.

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Up to the diaphragm the fed material should be ground to a size smaller than the slot width. Oversized material is retained in the coarse grinding compartment until it is sufficiently ground.

The most important design criterias for intermediate diaphragms are the slot width and the total slot area (open area).

6.4.2 Single diaphragm

Single diaphragm consist of :

- slotted plates
- centre screen

They are installed as separation between coarse grinding and fine grinding compartments of smaller mills (Fig. 24).

The centre piece is built either open with a screen or closed.

The single diaphragm is also applied as discharge diaphragm for centre discharge mills as shown in Figure 24.

Figure 23 Function of a Intermediate Diaphragm

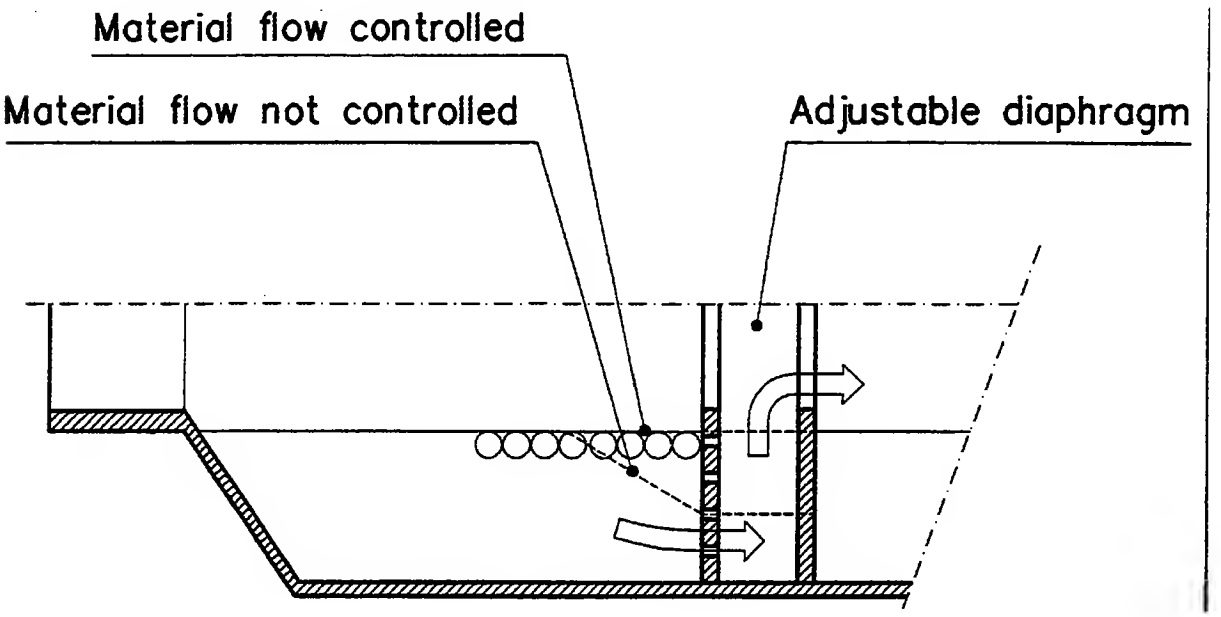

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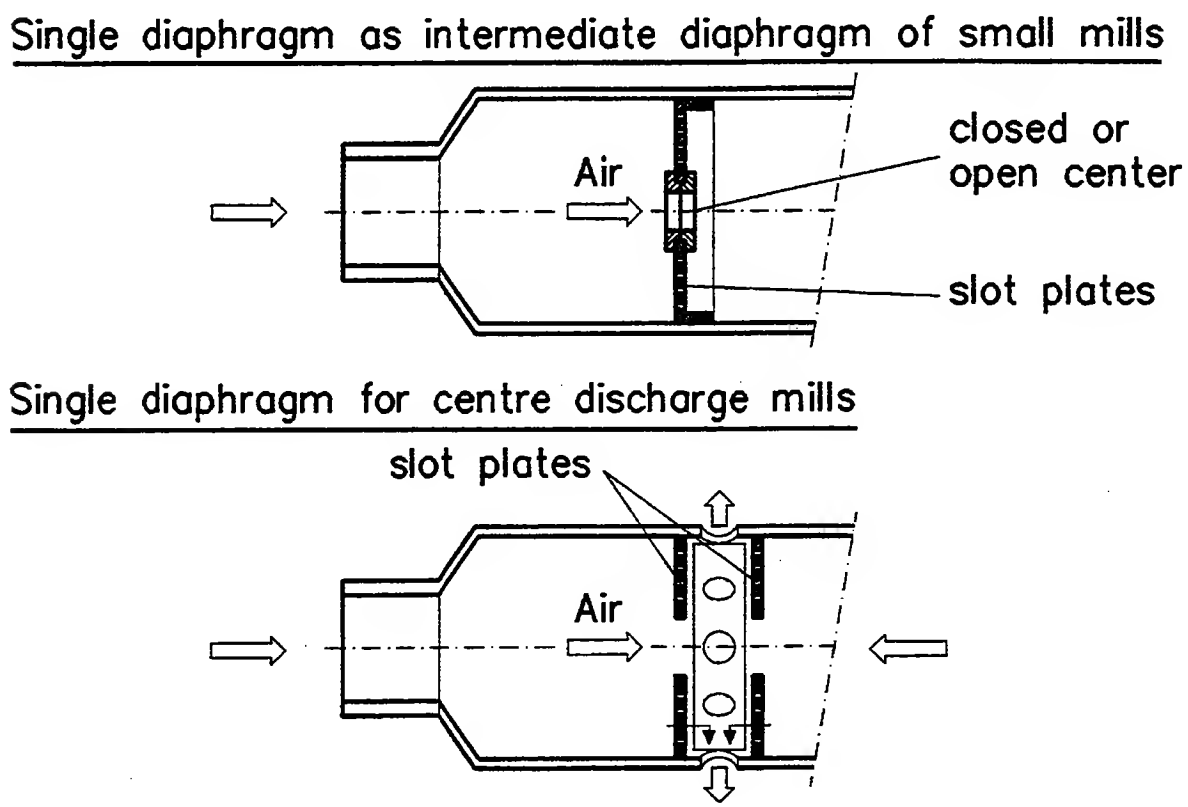
Figure 24 Single Diaphragm



6.4.3 Double diaphragm

6.4.3.1 Double diaphragm with lifters

Double diaphragms consist of slot plates on the inlet side and of blind plates on the outlet side. The diaphragm is furthermore equipped with lifters to transport the material (Fig. 25).

The lifters are either straight or curved. The centre part of the diaphragm is open to allow a sufficient passage of venting or drying air through the mill.

6.4.3.2 Double diaphragm system Combidan (FLS)

The Combidan diaphragm from FLS consists at the inlet side instead of slot plates of a screen which covers the whole mill cross section.

This relatively thin screen is protected against the impact of the large grinding balls by solid, highly wear-resistant protection plates. They are spaced to allow preground material but not grinding media to pass into the short separation compartment between the solid liners and the screen.

6.4.4 Adjustable diaphragm

The grinding efficiency of a mill is greatly affected by a proper retention time of the material in the first chamber.

There is a various number of material flow control systems available. All systems are generally based on the same working principle which controls in one way or the other the material flow from the first to the second chamber.

Slegten and Voest Alpine have instead of fix installed lifting plates adjustable scoops and Pfeiffer controls the material flow by changing the open area with an adjustable tube.

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Some of the main systems are shown in Figure 26.

Figure 25 Double Diaphragm

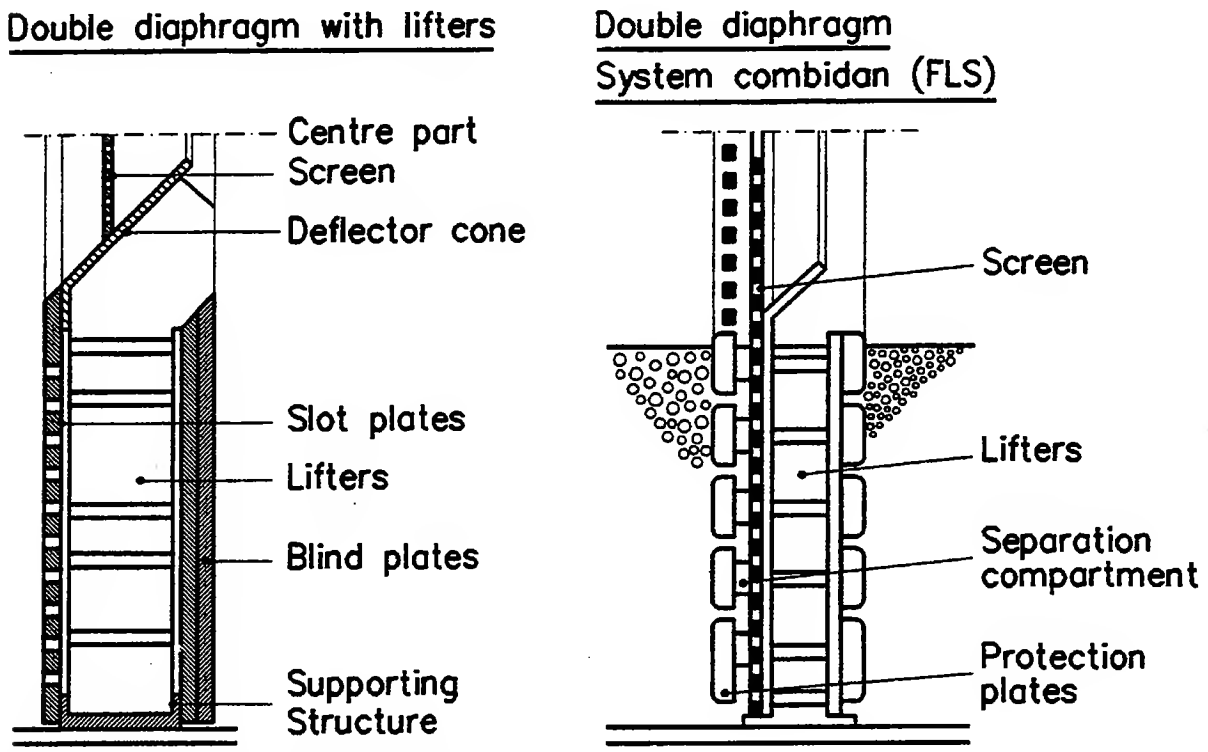
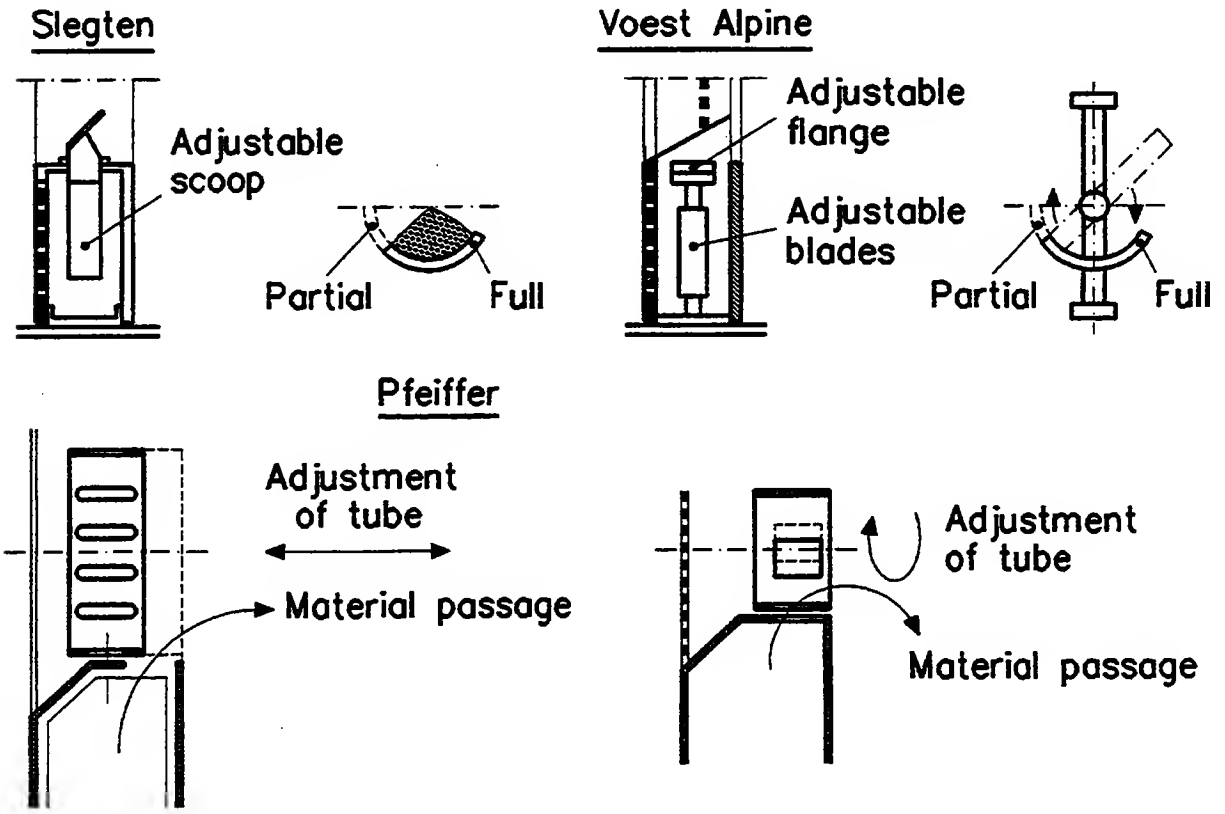


Figure 26 Adjustable Diaphragm



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6.4.5 Open diaphragms (Drying chamber diaphragm)

Open diaphragms (Fig. 27) are installed as separation between drying and coarse grinding compartment. Therefore they are often called drying chamber diaphragms.

The liner slots have to be large enough to allow the transfer of the material and the drying gas through the diaphragm.

The diaphragm is on the drying compartment side open and has to resist the forces coming from the ball charge of the grinding compartment.

In operation with hot gases the temperature on the diaphragm can go up to 400 °C. The diaphragm needs to be shock resistant also at high temperatures.

6.4.6 Slots (Intermediate diaphragm)

On the inlet side the intermediate diaphragms are provided with slot plates made of wear resistant steel. The slot width depends on:

the material to be ground	compartment length
ball size	mill venting air rate

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Some typical examples are shown in Figure 28.

The slot design must be in such a way that they fulfill the following conditions:

- constant slot width also with continuous wear
- low restriction to air and material flow
- resistance to slot breakage
- no clogging of slots by particles or due to impacts from the grinding balls

The slots are arranged radially or tangentially to the mill axis whereas radial slots are reducing the possibility of clogging and tangential slots yield a better material flow through the diaphragm.

6.4.7 Wear rate and life time of diaphragms

The slot plates are either made from special hardened wear resistant rolled steel, or they are made from high chromium castings. Some main features of both types of material are listed below.

Material	Cr content	Surface hardness
High chromium alloyed casting	up to 27 %	60 HRC
Rolled steel	low	50 - 60 HRC

The wear rate resp. life time of a diaphragm depend on the type of material being ground. Some guide figures are given in the table below.

Material	Life time [h]
Raw material (15 % R 90 µm; 17 kWh/t)	10'000 - 15'000
Clinker (2800 cm2/g; 30 kWh/t)	8'000 - 10'000

The slot plates are from the life time point of view the weakest point of the mill.

Figure 27 Open Diaphragm (Drying Chamber Diaphragm)

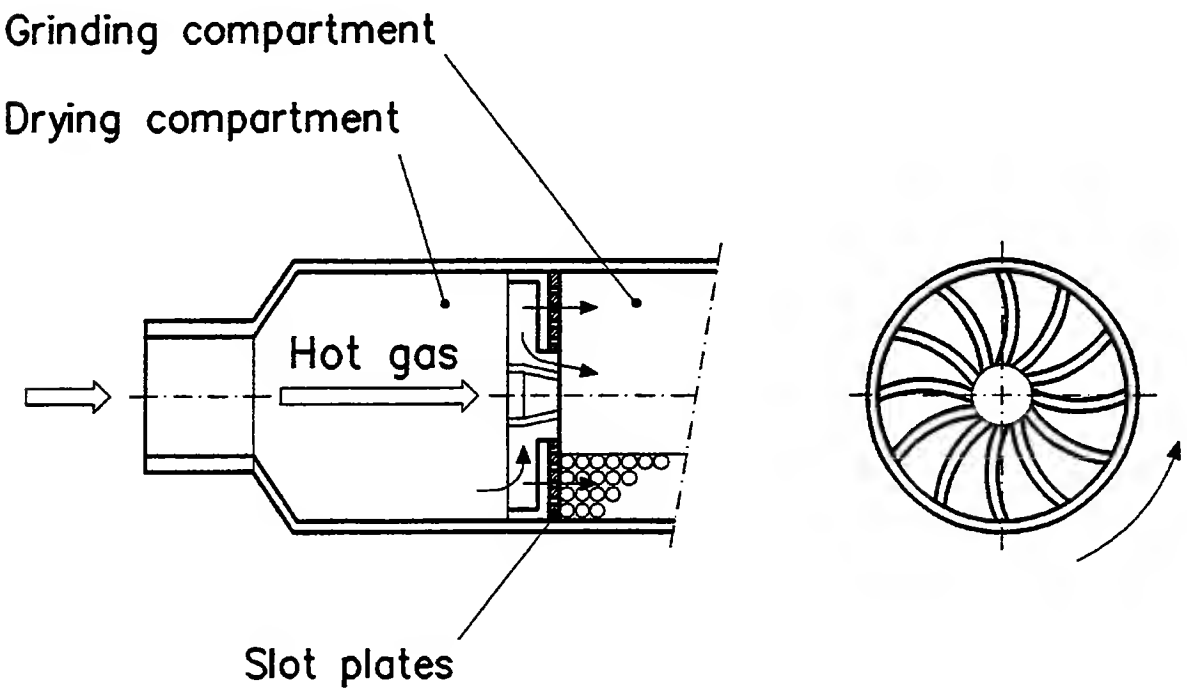

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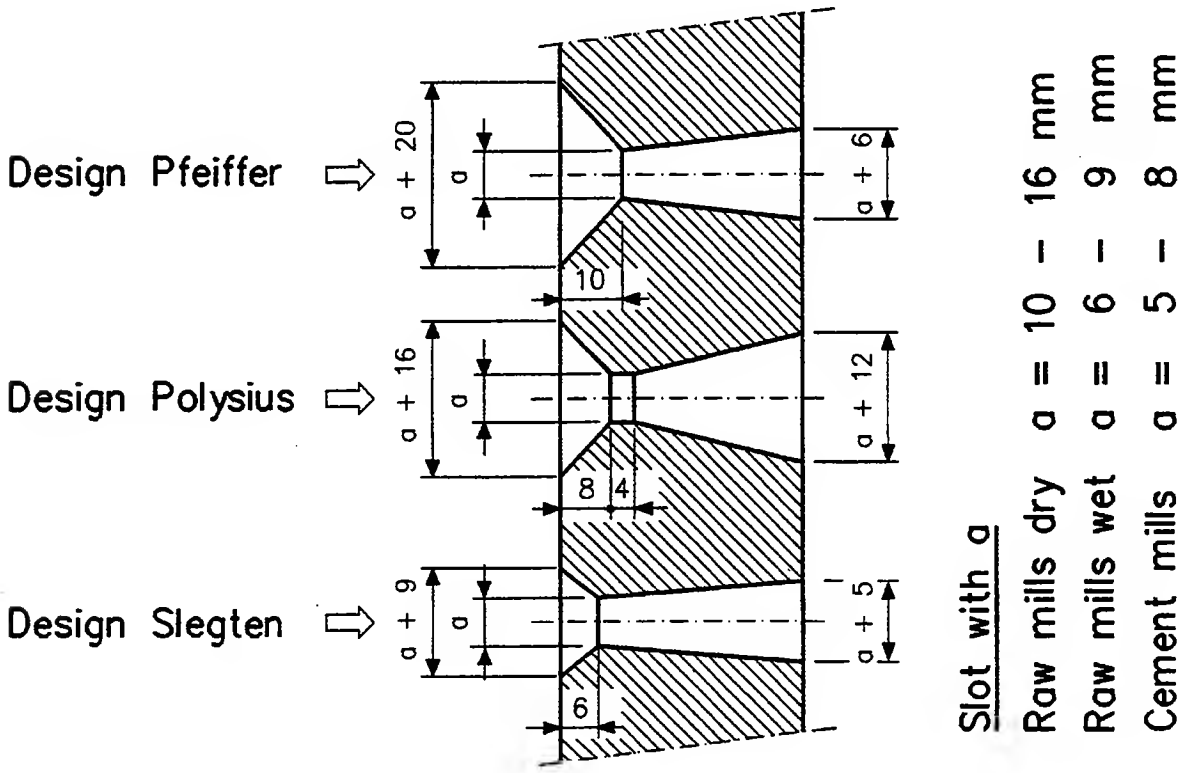
Figure 28 Slot Design for Diaphragm



6.5 Discharge diaphragm

Discharge diaphragms are installed at the outlet of an end discharge mill or are placed in the centre of a centre discharge mill.

The main components of a discharge diaphragm are shown in Figure 29. They are equipped with lifters and cones similar to the double diaphragm.

6.5.1 Diaphragm for centre discharge mills

The discharge diaphragm of a centre discharge mill consists of two single diaphragm which are installed at the outlet of the coarse and fine grinding compartment.

The difference between these two single diaphragms are their slot widths. Normaly the slots at the outlet of the fine grinding compartment are about two milimeters wider than the slots at the outlet of the coarse grinding compartment.

6.5.2 Combidan discharge diaphragm

The Combidan mill outlet (FLS) differs from the other known discharge designs (Fig. 30).

A very small outlet compartment is added to the fine grinding chamber. A dam ring equipped with liners protects the outlet screen from wear caused by the ball charge.

Figure 29 Discharge Diaphragm

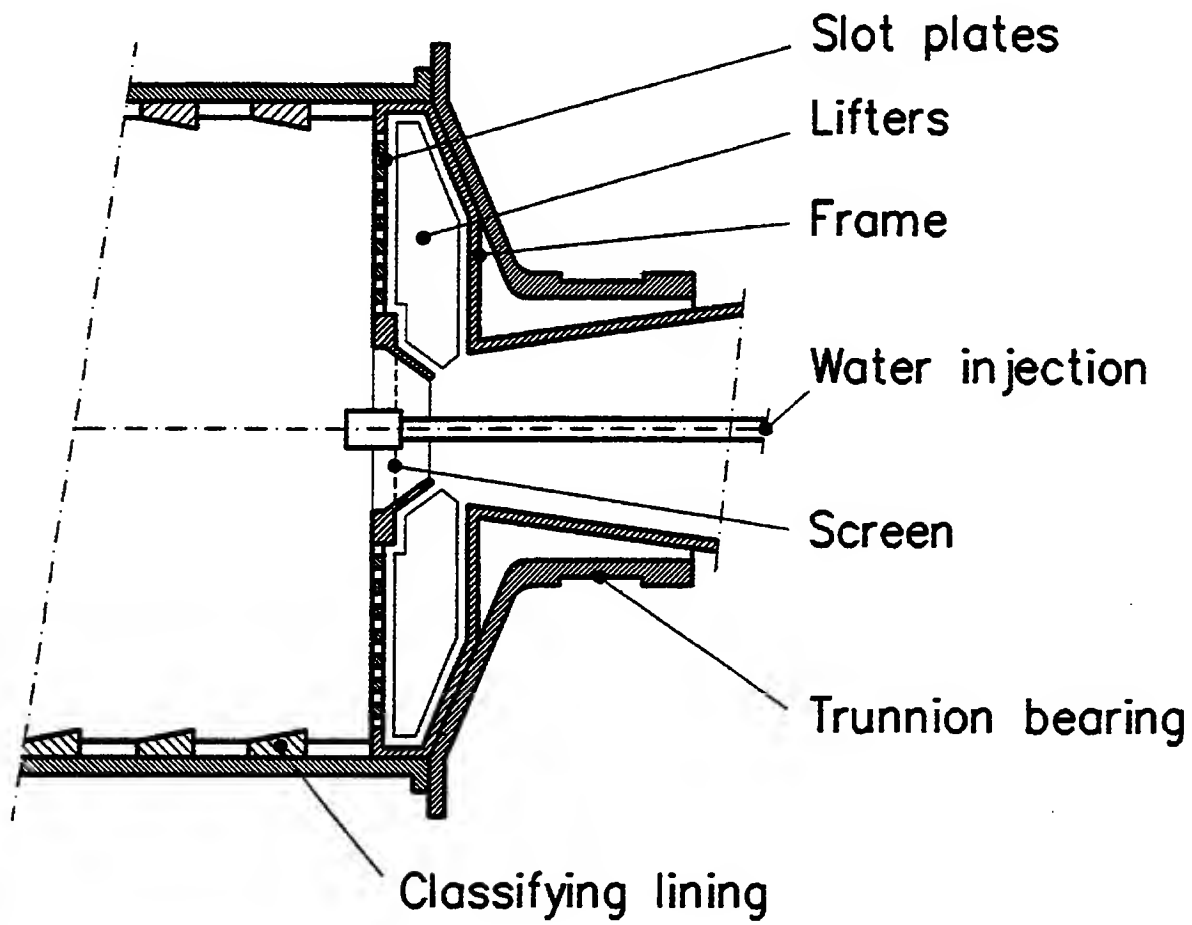


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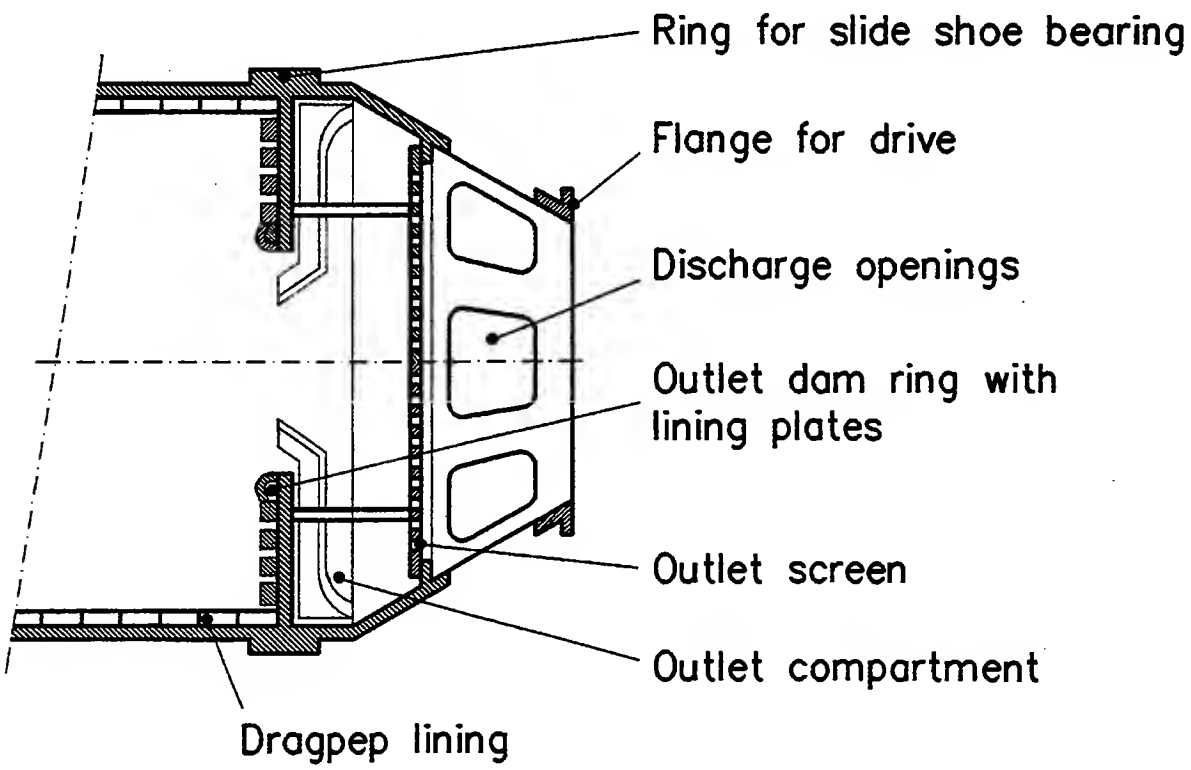
Figure 30 Combidan Discharge Diaphragm



6.5.3 Adjustable discharge diaphragm for slurry mills

This discharge diaphragm for slurry mills consists of slot plates, structure and an adjustable cone (Fig. 31).

Depending on the position of the cone the material level inside the diaphragm and grinding chamber stays from low to high, i.e. the material level in the chamber of slurry mills can be adjusted.

The correct amount of slurry in the grinding compartment has the following advantages:

- low wear rate on liners and grinding media
- good use of the grinding compartment

6.5.4 Slots (discharge diaphragm)

The slots for the discharge diaphragm are normally wider than those for the intermediate diaphragm but also depend on the material to be ground and the size of the smallest balls in the grinding chamber.

Following guide values for slot width can be given:

- Raw mills (dry) 12 - 18 mm
- Slurry mills (wet) 8 - 11 mm
- Cement mills 7 - 10 mm

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6.5.5 Wear rate and life time of discharge diaphragm

The wear at the discharge diaphragm is caused by friction forces between the grinding media and the diaphragm or the material and the diaphragm.

Since there is no heavy impact on the plates the wear rate is lower than the wear rate on the intermediate diaphragm.

Figure 31 Adjustable Discharge Diaphragm for Wet Grinding

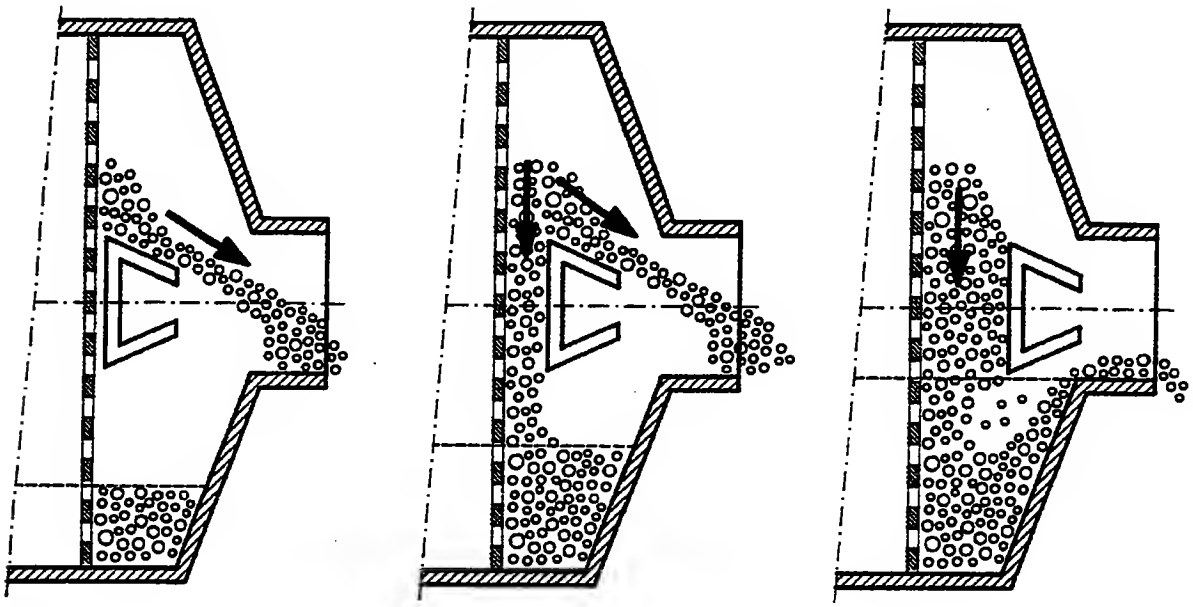
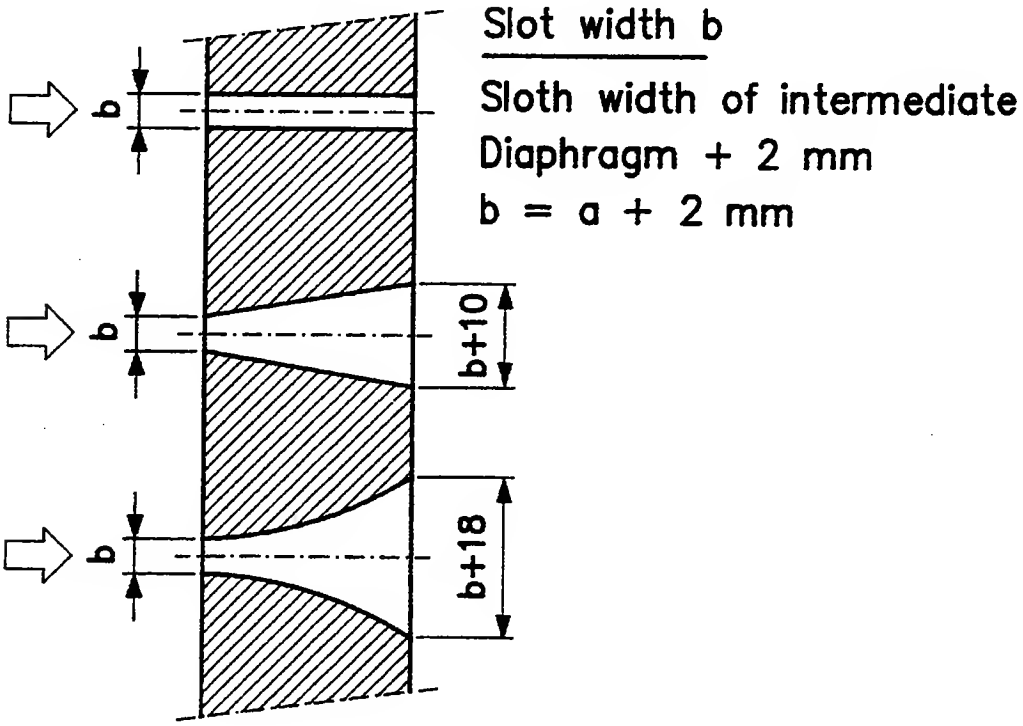


Figure 32 Slot Design For Discharge Diaphragm



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6.6 Grinding media

6.6.1 General

The ball charge of a mill consists of grinding balls of several sizes and of different material qualities.

For coarse grinding normally balls between 50 and 100 mm and for fine grinding balls between 15 and 50 mm are used. The maximum and minimum ball size and the composition depends on various factors, such as:

- max feed size of material to be ground
- product fineness
- mill diameter, mill length
- grindability and mineralogical structure of the feed material
- mill system (open/closed circuit, number of compartments, circulating load)

Depending on the grinding process resp. grinding step the balls have to resist:

- impact forces by the balls themselves (specially in coarse grinding with big balls)
- wear caused by friction forces between the balls and the liners
- wear caused by abrasive materials
- corrosion

To maintain a constant and optimum mill output, the selection of the adequate ball sizes as well as of the respective material quality has to be done very carefully.

6.6.2 Material quality

The mill can be filled either with cheap low quality balls with high wear rates or with expensive high quality balls with lower wear rates. It has to be considered, however, that low quality balls also call for more frequent recharging of the mill in order to replace the worn grinding media, thus resulting in a lower utilization rate of the mill and higher maintenance costs.

The material qualities as shown in Figure 33 are applied for the grinding media.

Due to their good wear resistancy a clear trend toward the use of high chromium cast iron quality balls is observable.

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Figure 33 Material Quality of Grinding Media

Material quality	Chemical composition		Feature
Forged grinding media	0,5 – 1,0 % C 0,8 – 1,2 % Mn 0,8 – 12,0 % Cr		– Available everywhere at favourable prices – Applied for not too abrasive materials – Hardness 25–65 HRC (surface only)
White cost iron	2,8 – 3,3 % C 0,1 – 0,4 % Mn 0,1 – 0,4 % Cr 0,4 – 0,8 % Si		– Specially used for wet grinding – Large quality variations – Only for mills < ⌀ 3,5 m – Only for fine grinding – Hardness : 45–50 HRC
High chromium cast iron	Raw grinding dry	2,0 – 3,0 % C ~ 12,0 % Cr	– Very wear resistant – Hardness : 48–68 HRC (throughout)
	Raw grinding wet	20,0 – 30,0 % Cr	
	Cement grinding	2,0 – 3,0 % C 12,0 – 17,0 % Cr	

6.6.3 Wear rate

The wear of the grinding media is caused by:

- impact
- friction forces
- abrasiveness of the material
- corrosion

Grinding media (e.g. balls and cylpebs) are either forged or are cast from wear resistant high chromium steel. Depending on the quality of the grinding media wear rates of 15 - 1200 g per ton of material can be expected.

Considering the material qualities as mentioned in the last chapter the specific wear rates as listed in Figure 34 could be expected. Following assumptions were made:

- Raw material ~ 17 kWh/t, 10 - 15 % R 90 µm
- Slurry ~ 10 kWh/t, 10 - 15 % R 90 µm
- Cement ~ 30 kWh/t, 3000 cm²/g

Balls in white cast iron quality can not be used in coarse grinding compartments due to the danger of ball breakage.

The wear rates given for cement consider only portland cement. For composite cements where parts of the clinker are replaced by additives such as slag, pozzolana, etc. the wear rates can be much higher, e.g. ~200 g/t grinding a pozzolana cement with high chromium balls.

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Figure 34 Specific Wear Rates of Grinding Media

Material quality	Unit	Specific wear rate					
		Raw material		Slurry		Cement ①	
		Small balls	Large balls	Small balls	Large balls	Small balls	Large balls
Forged balls	g/kWh	2 – 10	10 – 30	15 – 50	20 – 80	2 – 15	10 – 40
	g/t	30 – 170	170 – 500	150 – 500	200 – 800	60 – 450	300 – 1200
White cost iron balls	g/kWh	2 – 6	–	20 – 60	–	5 – 15	–
	g/t	35 – 100	–	200 – 600	–	150 – 450	–
High chromium balls	g/kWh	1 – 3	2 – 5	10 – 20	15 – 30	0,5 – 1,5	1 – 2
	g/t	15 – 50	30 – 80	100 – 200	150 – 300	15 – 50	30 – 70

① Portland Cement

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Operation of Tube Mills

Hp. Fisch
VA 89/5588/E

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1. INTRODUCTION

Grinding systems with tube mills have been operated throughout the world for many decades. Many practical ways to operate these mills in an easy way have been developed and many improvements introduced, including the application of new technologies.

But still, the operation of tube mills involves a large number of items and routines which importance often is not sufficiently considered by the responsibles. Many times a great deal of facts are neglected and even fundamental procedures are unconcernedly carried out, leading to constant drawbacks.

Any grinding system should be operated in such a way as to obtain the necessary product quality and to achieve the maximum mill output. As soon as an efficiency drop is observed then the reasons should be investigated. There are many different grinding systems and **each system underlies its own and specific conditions.** It is obvious that no standard instruction can be given for all possibilities and cases. This paper serves as general guide line for better mill operation and the procedure of practical mill assessment.

In a cement plant, the largest portion of electrical energy costs is related to cement grinding. Any improvement in grinding efficiency results in considerable savings. Subsequent guide lines refer therefore in the first place to cement grinding plants, where the procedure for raw mills and coal mills is similar and can be derived from this. Raw and coal mills include additionally the drying aspects which have to be assessed.

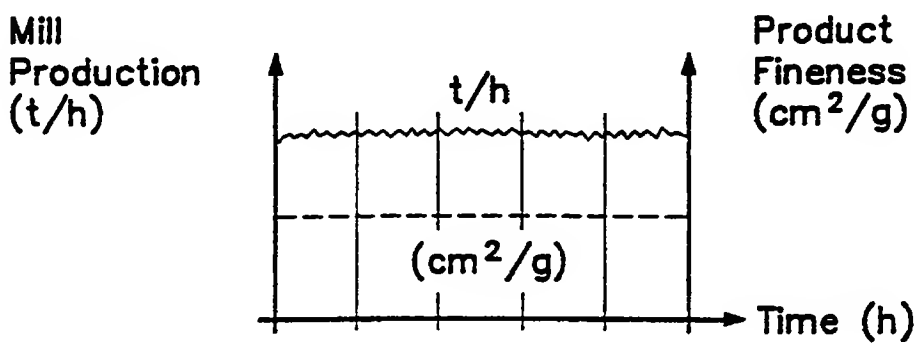
2. NORMAL OPERATION

Normal operation can be defined as the day to day operation of the mill system with a continuous mill output and a steady cement quality (fig. 1). Any common disturbance can easily be equilibrated with normal means of mill control, either manually or automatically with a closed loops control system.

Prerequisites for normal operation are

- well maintained equipment of the mill system with properly working control elements
- steady and normal feed material properties
- minimal interruptions in operation by external factors (lack of feed material, electricity failure, other shut downs, etc.)
- well trained mill operators and ev. a suitable, automatic mill control circuit

Figure 1 Normal Operation



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Small variations of operating parameters within the system can be corrected by adjustment of the available control elements. The most common control elements are:

Figure 2 Adjustment Possibilities

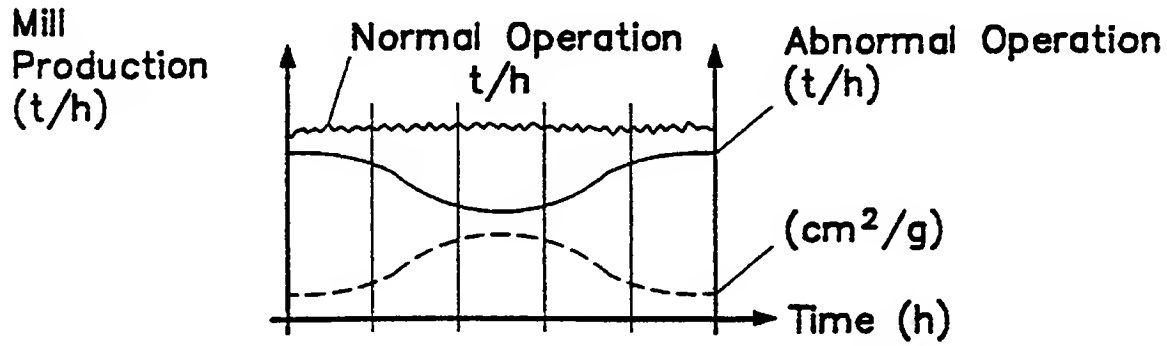
Control Element	Adjustment of	Effect on
Weighfeeder	Speed	Feed rate (= mill output) Proportions of components Circulating load
Separator	Rotor Speed Dampers Rotor Blades Fan Damper	Product fineness Granulometry Spec. surface Circulating load
Static Separator	Position of vanes	Granulometry of filter dust
Water Injection	Regulating valve	Mill discharge temp., Coating
Mill Venting Fan	Position of damper	Mill cooling, Dust granulometry

Conditions and parameters for normal operation have to be recorded for reference purpose.

3. ABNORMAL OPERATION

As abnormal operation can be defined all operating conditions beyond the normal range. In this case, either the required mill output rate and/or the required quality cannot be reached relative to the recorded normal operating conditions (fig. 3).

Figure 3 Abnormal Operation



Abnormal operating conditions can be caused by many different disturbances upsetting the continuous running. Disturbances might result from

- changes in feed material properties
- Equipment problems (mechanical or process technological)
- Deficiencies of control elements
- Sensing element deficiencies (signal transmission errors etc.)
- External effects

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As soon as deviations from the normal operation are encountered, the possible reasons have to be assessed in order to undertake necessary measures for correction. Often an abnormal operation can be counteracted by corrections through corresponding actions of the mill operator or the maintenance crew. If normal operation can not be reestablished by such procedures, the execution of a performance test of the mill system is indicated.

Fig. 4 gives a general check-list to find the possible reasons and, to a certain extent, the possibility of taking first actions. In some cases it is not adequate to take individual actions because the problems are more complex. Only a performance test of the entire grinding system with a subsequent evaluation will reveal the overall problems.

Figure 4 Check-list 'Abnormal Operation'

INDICATION	POSSIBLE REASON	ACTION
Product:		
too coarse	Mill overfilled Separator adjustment Mill feed very coarse Filter dust too coarse	Low mill feed rate Proper adjustment Improve precrushing Adjust static separator / Venting rate
too fine	Mill underloaded Separator adjustment Mill feed very fine Filter dust very fine	Increase mill feed rate Proper adjustment Further investigation required Adjust static separator / Venting rate
wrong chemical composition	Wrong feed proportions Wrong feed components	Adjust proper proportions Change component quality
Mill: *)		
sound dull	Mill overfilled	Lower mill feed rate
sound loud	Mill underloaded	Increase mill feed rate
Cement Temperature:		
too high	Clinker temp. too high Low mill / separator venting Lack of water injection Cement cooler failure	Check clinker cooling Adjust venting rate Adjust flow rate Mechanical replacement Repair
Mill Output:		
too low	Unsuitable ball charge composition or badly worn balls Diaphragm unsuitable or clogged Unsuitable or worn liners Change of feed grindability Coating of balls Adjustment of separator Operators error	Change ball charge Replace or clean diaphragm Replace liners Further investigations required Improve cooling / use grinding aids Adjustment Instruction / training
Mill Exhaust Pressure:		
too high	Inlet / outlet clogged Diaphragm clogged	Clean Clean diaphragm
too low	Diaphragm broken Major false air penetration	Replace broken plate Seal sources

*) same applies for Elevator Amps and reject rates from separator

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4. PERFORMANCE TEST

4.1 Purpose

Performance tests should be carried out on following occasions:

* After start-up of a new mill to verify the guarantee figures
* Poor performance of a grinding system; low mill output or poor product quality
* High operation or maintenance cost
* Implementation of new processes and techniques, e.g. roller press, new separator, new control circuit
* Planned capacity increase, e.g. in connection with increased clinker production, reduction of grinding hours (utilization of low tariff hours)

The results of performance tests could be expressed in terms of efficiencies which allow a **comparison** with other grinding systems and **reference values**. The results provide also the basic information for required changes or modifications within the mill system.

A complete mill performance test comprises basically the following work:

* Determination of mill output and power consumption
* Sampling within grinding system
* Sampling inside mill
* Gas flow measurements (volumes, pressures, temperatures)
* Recording of operational data
* Visual inspection of equipment
* Evaluation of test

The results provide also the basic information for required changes or modifications within the mill system:

- **Grinding efficiency** - improvement of mill internals (ballcharges, liners, diaphragms, compartment lengths, etc.), inlet and outlet arrangement
- **Separator efficiency** - improvement of separator system (separating air flow, dedusting, mechanical adjustments, etc.)
- **Cooling resp. drying efficiency** - adjustment of air / gas flows, adaptations of dedusting systems
- **Weak point analysis** - find points which limit the system performance (sizing and type of equipment, operating ranges)

The experience shows that this last point gains more and more importance with the implementation of new techniques, where often many existing items form a bottle-neck in the mill production.

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4.2 Measurements

4.2.1 Mill Output

The determination of mill output G [t/h] is one of the most significant factors for the performance of a mill system.

Methods for output measurement:

- **Indications from weighfeeders or belt scales**
  The most accurate values are the readings of totalizers T (accumulated [t]) and time measurement t [h] during the test period resulting in [t/h]. Instant readings may be used for reference purpose.
  The weighing facilities have in any case to be calibrated prior to start of tests.
- **Table feeders**
  The feed to the mill is bypassed for a short period, weighed and then returned to the mill. The test has to be done several times. No adjustments of the feed rate should be made during the test.
- **Silo measurement**
  The tonnage between two silo filling levels in relation to the recorded operating time gives and indication of the mill output. The variable material density can be based on experience values.

4.2.2 Energy Consumption

It is in most cases sufficient to determine the energy consumption of:

* Mill
* ev. Roller Press
* Auxiliaries
* Total Mill System

During the test period the energy readings from totalizers E (accumulated [kWh]) as well as the time t [h] is recorded. The division of the values is then an average absorbed power P [kW]. The relation to the measured mill output G [t/h] indicates the spec. energy consumption q [kWh/t].

$q = P\,[kW] / G\,[t/h] = [kWh/t]$

or

$q = E\,[kWh] / T\,[t] = [kWh/t]$

Cross checks for the power input to the mill drives can be made under utilization of the mill formula, calculating the mill power on basis of the mill dimensions and ball charges.

4.2.3 Temperatures, Air-flow and Pressures

Product:

Significant are mainly the temperatures of clinker, of mill discharge product and of the cement before the silos. These values are utilized to perform a heat balance and to judge process technological phenomena.

Air:

Of prime interest is the performance of the mill ventilation and, in an increasing way, also the separator. Fig. 5 shows the typical measuring points:

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Figure 5 Air Measurement Points

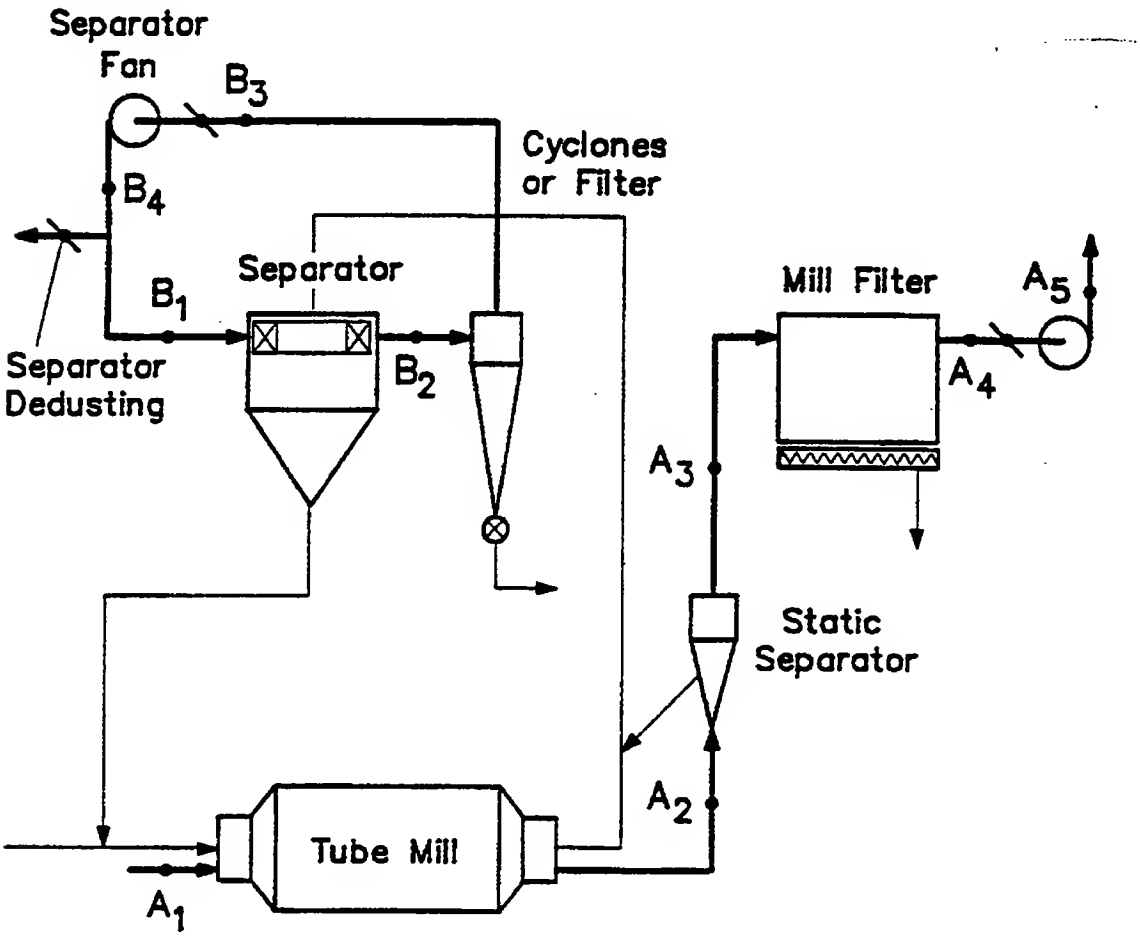

Figure 6 Type of Air Measurements

Mill Ventilation		
Points	**Pressure drop**	**Measurement**
A1 Ambient	Δp mill	T, pa
A2 Mill exit	Δp static sep.	T, p
A3 Static sep. exit	Δp filter	T, p
A4 Filter exit	Δp fan	T, p, $\dot{V}$
A5 Fan exit		T, p

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Separator Venting		
B1 Ambient / Inlet	Δp separator	T, pa
B2 Separator exit	Δp cyclones	T, p
B3 Cyclone exit	Δp fan	T, p, $\dot{V}$
B4 Fan exit		T, p

Symbols:
T - temperature
p - static pressure
$\dot{V}$ - air flow rate

4.2.4 Operational Data

During the performance test also the operational data have to be recorded. This can be done on an hourly basis:

Possible operational data are:

- **Mill**
  Absorbed power, mill feed rates;
  Water injection rates, grinding aids (- ev. checking via bypass measurements), mill sound levels
- **Separator**
  Rotor speed, damper and vane positions (number and position of auxiliary blades), absorbed power of each drive (including fan)
- **Static separator**
  Position of vanes
- **Mill ventilation**
  Absorbed fan power, position of dampers, pressure indications, filter data
- **Conveyors**
  Absorbed power of bucket elevator, pneumatic transport data

4.3 Sampling

To assess the performance of the grinding system, it is important to know the exact granulometry of the individual material streams within the grinding circuit. Prerequisites for a reliable sampling during the performance test are

* Stable operation of the mill system
  (begin sampling earliest 6 - 8 h after start of cold mill)
* 6 - 12 h duration of performance test
* Taking samples from the system every 1/2 to 1 h

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4.3.1 General Sampling Routines

Prior to the performance test, the exact sampling points have to be determined. The sampling points have to be selected in such a way as to make sure that the sample is representative for the entire material flow in that location. That means, it has to be avoided that

- part of the sample (the fines) is swept away by dedusting air.
- material from another source is partly collected with the sampler.

Suitable samplers must be provided for the corresponding location to assure an acceptable representativity of the sample.

It is often not possible to find a suitable sampling location which fulfill the requirements.

4.3.2 Sampling within the Mill System

Possible sampling points within a cement grinding system are shown in fig. 7 and the required quantities and sieve analysis in fig. 8.

Figure 7 Sampling Points with Mill System

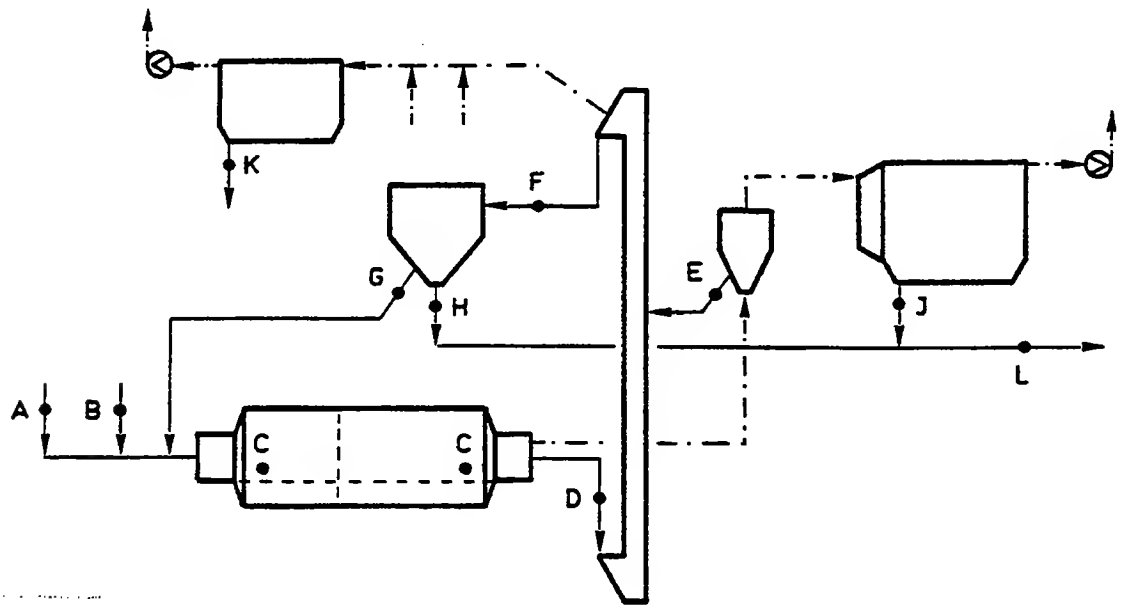

Figure 8 Sample Quantities and Analysis

Sampling point	Weight	Required sieve analysis
A/B Mill feed	10 kg each	e.g. 30, 25, 16, 8, 4, 2, 1 mm
C Samples within mill	See longitudinal sieving	
D Mill discharge E Separator tailings	0,5 kg	e.g. 30, 60, 90, 150, 200 µm Blaine
F Separator feed G Separator tailings H Separator fines		e.g. 4, 8, 15, 20, 30, 60, 90, 150, 200 µm Blaine
J Filter dust K Filter dust		e.g. 30, 60, 90, 150, 200 µm Blaine
L Final product		e.g. 4, 8, 15, 20, 30, 60, 90, 150, 200 µm Blaine

Individual samples from each sampling point are collected during the period of performance test and a composite sample formed per point. These samples have to be homogenized and splitted to the required sample of 0.5 kg for the sieve analysis and Blaine measurement.

Samples from the mill feed components have to be taken once only. Depending on the additives used, also the moisture contents of these additives have to be determined.

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4.3.3 Longitudinal Sampling

The grinding efficiency of a tube mill is influenced by a series of factors, e.g.

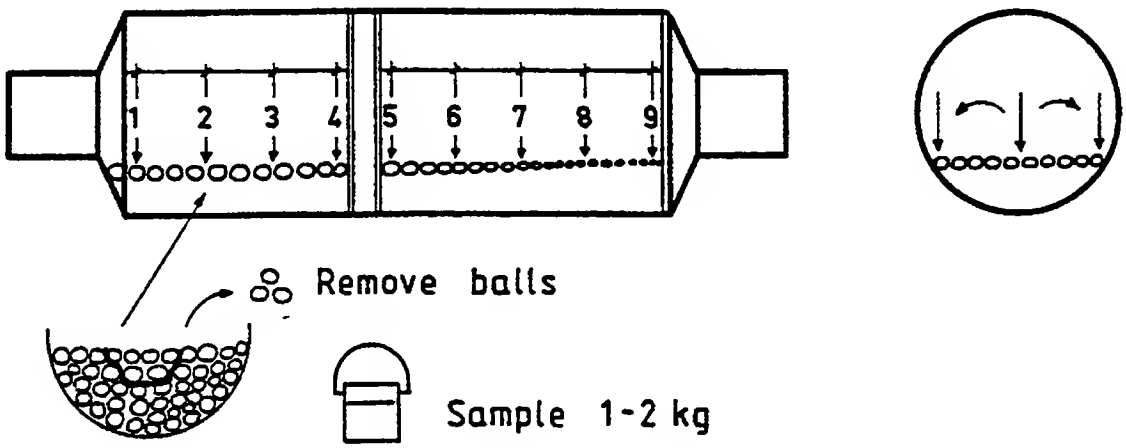
- Mill type (and mill system)
- Feed material properties
- Ball charge composition
- Filling degrees, mill speed, shape of liners
- Compartment lengths (in conjunction with cement type and circulating load)
- Diaphragms, type and mechanical conditions
- Mill ventilation (air speed inside mill)
- Temperature and moisture content of feed material

The best indication of the grinding efficiency and how the mill gets along with all these factors gives a longitudinal sampling test that means the sampling within the mill. For this purpose following procedure should be observed:

- Stable operation of the mill system for some 10 - 12 h
- The performance data prior to this sampling have to be recorded as described before
- Preparation of required sampling equipment such as metal cans (properly marked), light sources, measuring tapes, shovels, etc.
- The mill and all auxiliary equipment have to be stopped (emergency stop) to be sure that the conditions inside the mill remain unchanged. The shut down of the mill ventilation is important to avoid that the fines from the mill discharge are swept away.
- Before entering the mill, the necessary safety measures should be taken (disconnect **main circuit breakers**)

4.3.4 Sampling Procedure

Figure 9 Sample Taking



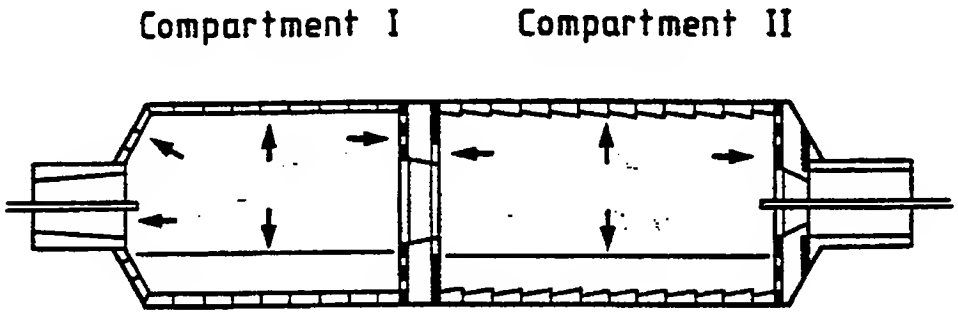
Each sample should be collected from three points in the same cross section of the mill and weigh 1 - 2 kg. To get a representative sample some balls should be removed and the material taken from inside the charge.

One composite sample each 1 m along the mill axis should be taken. The first and the last sampling point should be 0.5 m off the wall / diaphragm as there are often some retained particle fractions or segregated material leading to unrepresentative samples.

Parallel to this sampling inside the mill also a visual inspection should be performed.

4.3.5 Visual Inspection

Figure 10 Visual Inspection within Mill



Following items should be checked within the mill to give important hints on the grinding efficiency:

4.3.6 Material Level and Distribution

The material should be evenly distributed over the ball charge in both compartments. The material level should be:

- **1st compartment**: balls just visible, at the inlet an accumulation off material could be possible.
- **2nd compartment**: balls entirely covered by a material layer of about 20 - 30 mm.

4.3.7 Grinding Media and Charge

Necessary checks are:

- Measurements of height above charge and internal mill diameter as well as exact compartment lengths (d_i, h, l_i) for recalculation of filling degree and absorbed mill power
- Ball sizes and their distribution along the mill axis (especially in mills equipped with classifying liners)
- Shape and conditions of grinding media (worn and/or broken media) as well as foreign bodies (metal scrap) in the charge
- Formation of coating around the grinding media

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4.3.8 Shell Liners and Diaphragms

- Conditions, shape and remaining thickness of liner plates. Formation of coating
- Conditions of blind plates (wear)
- **Diaphragms**: Size and shape of slots, conditions and remaining thickness. If the slots are blocked, cleaning will be necessary, eventual screening out of the scrap in the media charge might be indicated. Gaps between slot plates and deficiencies in the center grates should be repaired.

4.3.9 Water Injection Nozzles

The nozzles should always be open and clean, no damages from impacts. Obstructions with material can be avoided by proper control of the cleaning air in the waterspray system.

4.3.10 Sample Analysis

From each sample a sieve analysis has to be carried out:

Figure 11 Sieves for the Longitudinal Sampling Analysis

1st compartment	Sieves 16, 10, 6, 2, 1.25, 0.5, 0.2 [mm]
2nd compartment	Sieves 1.25, 0.5, 0.2, 0.12, 0.09, 0.06 [mm] Blaine [cm^2/g]

Often not representative results are gained from the sieve analysis when metal parts are not separated prior to sieving or when the homogenizing and the dividing of the composite samples are not properly done.

4.4 Inspections

Besides the visual inspection within the mill at the occasion of the longitudinal sampling, the entire mill system should be inspected in view of mechanical as well as process technological aspects. In many cases the real bottle necks can be detected just by observing closely the mill operation.

Some examples:

* Limited capacity of feeders or conveying elements
* Wrong readings from such elements resulting in incorrect mill control
* Mechanical deficiencies (wear!)
* Insufficient material flows or flow divisions
* Air flow problems and dedusting deficiencies, blockages
* Not representative sampling

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5. EVALUATION OF PERFORMANCE

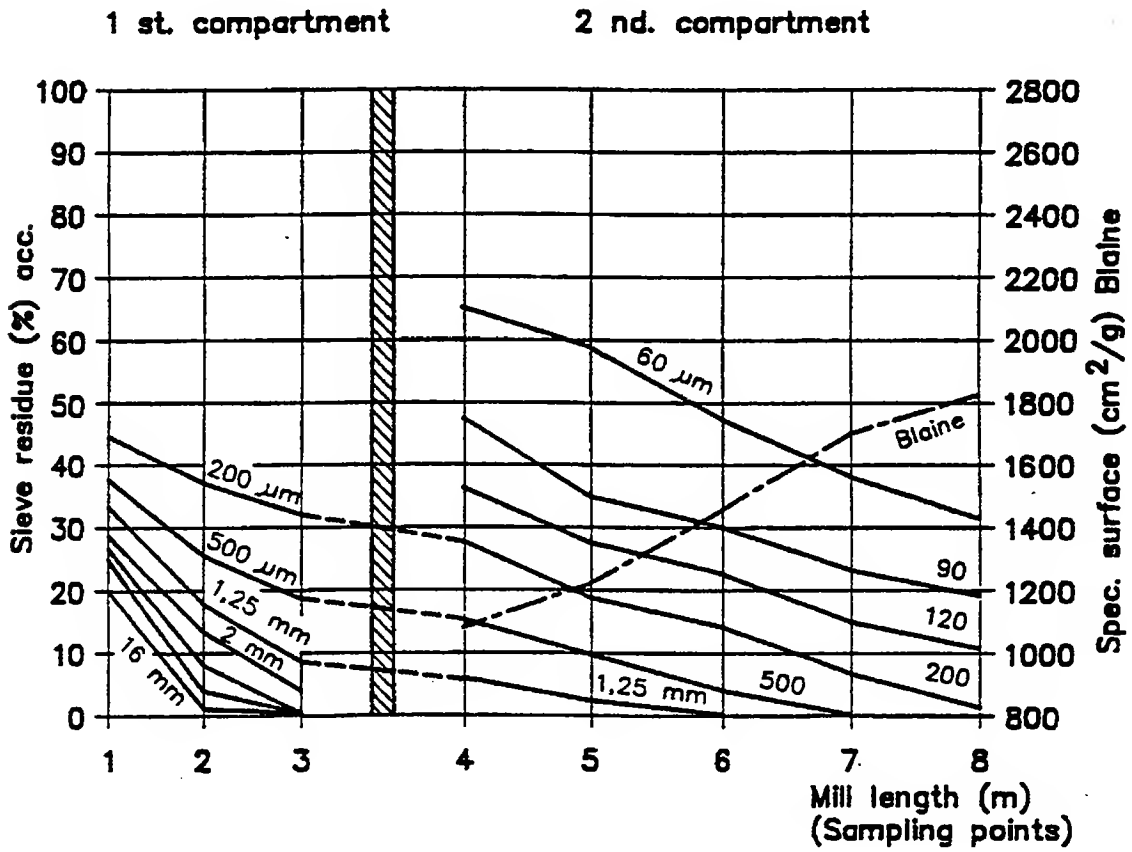
5.1 Interpretation of Test Results

The interpretation of the test results is very much a matter of experience and training. Who is less experienced takes guide lines and general indications for reference. Subsequently some basic guide lines are given. More indications are available in the respective sections of this documentation.

5.1.1 Grinding Efficiency

The main tool for judgment of the grinding efficiency is the grinding diagram as a result from the longitudinal sieving. A sample plot is given in fig. 12.

Figure 12 Typical Grinding Diagram for a Two-Compartment Cement Mill



Above diagram is typical for a two-compartment cement mill in closed circuit producing an ordinary portland cement with a fineness of ~ 3000 [cm^2/g] Blaine.

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For a judgment of the diagram usually the sieve residues and the Blaine value in front of the diaphragms are used. Reference values are:

- In front of **intermediate diaphragm**:
  15 - 25 % R 0.5 [mm]
  max. 5 % R 2.0 [mm]
- In front of **discharge diaphragm**:
  15 - 25 % R 0.09 [mm]
  max. 5 % R 2.0 [mm]

In case the results are not in line with these values, the possible reasons have to be analyzed in connection with all other findings from the performance test. Some of the possible correcting measures are listed in fig. 4.

5.1.2 Separator Efficiency

Good means for a judgment of the separator efficiency are:

* Trompcurve
* Separating efficiency curve
* Spec. separator loads

which can be computed on basis of analysis and measurements from the performance test (see section: separators).

5.1.3 Dedusting / Cooling Efficiency

Judgment criteria are mainly:

* Massflow / Gasflow measurements
* Dust granulometry
* Heat balances

established on basis of performance test results (guide lines see: mill ventilation and cement cooling).

6. GRINDING MEDIA CHARGES

The grinding media charge has a major impact on the grinding efficiency, due to:

- Charge conditions (worn media)
- Charge compositions (media grading)

6.1 Charge Conditions

Badly worn grinding media change the grinding behavior that means it may lead to a different grading of the charge, influence the retention time of material through the mill, disturb the classifying effect in the 2nd compartment and if completely worn down obstruct the diaphragm slots. Therefore conditions of media charges have to be checked on a regular basis and the charge if necessary replaced.

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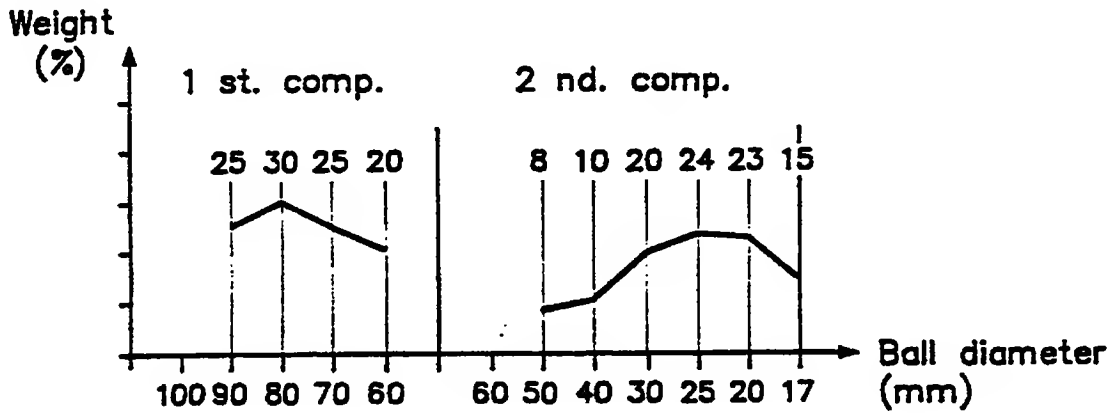
6.2 Charge Composition

The correct charge composition depends on may influencing factors and has to be individually adapted in every case. Also quite different philosophies to determine grading and top size media are applied. Still, to compose a media charge is a matter of experience which so for could hardly be replaced by computer programs.

The main basis for the determination of media charge is the grinding diagram and the overall results of the performance test. The composition of the charge has to ensure an optimal grinding progress within the mill and the correct circulating load in a closed circuit. Usually various cement types are produced with one mill and the media composition is adapted to the type which is produced in larger quantities. Fig. 13 shows a typical media charge composition for a two-compartment cement mill in closed circuit for an OPC of ~ 3000 [cm^2/g] Blaine.

Figure 13 Typical Media Charge Composition



It is good routine to determine a certain media charge composition and first fill in only 80 % of the final charge weight. With this reduced charge a longitudinal sieving can be performed. According to the results of this test, the remaining 20 % of charge can still be corrected and filled in to reach an optimal adaptation.

6.3 Replenishment

Replenishment with grinding media to make up for wear should take place on a regular (running-) time basis (according to the wear rate) and reduction of power consumption. Depending on the criteria applied for the determination of the media charge composition the replenishment method has to be selected. For equilibrium charges only the largest media size of each compartment is needed whereas for a different charge composition portions of the two largest sizes or even more might be necessary to replenish the media charge to the desired weight.

Generally the wear of grinding media proceeds in such a way that the diameter reduces the same amount for any media size as shown in fig. 14.

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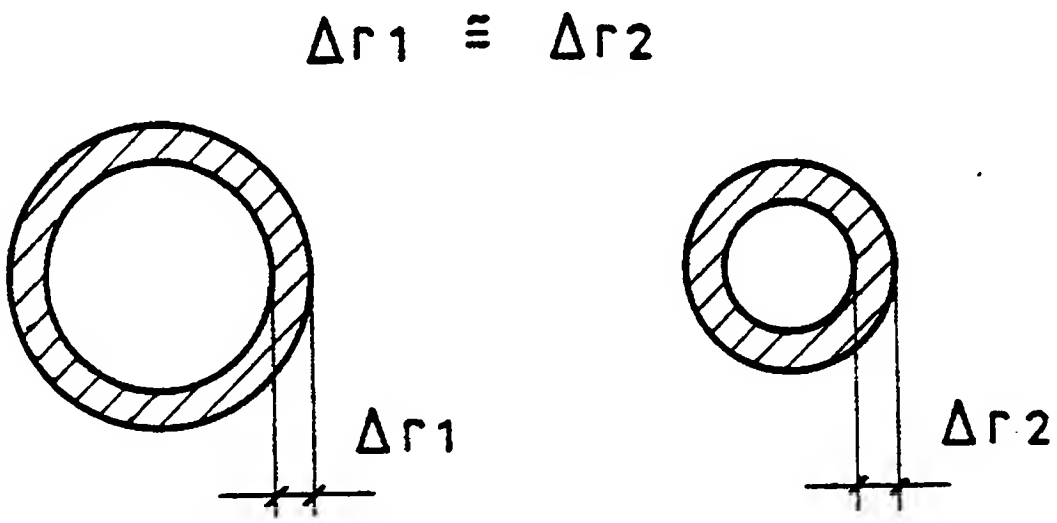
Figure 14 Wear Pattern of Grinding Media



6.4 Recharging

In spite of all theories describing the wear procedure of grinding media it is hardly possible to maintain a determined and optimal media charge over a longer period of operation. It is therefore necessary to discharge the media from time to time and regrade the grinding media. A new grinding media charge can then be put back into the mill where the undersize media and the scrap is sorted-off.

Depending on the wear rate as a function of the specific conditions in grinding system, recharging should be done at least once every or once in two years.

In a second compartment with classifying lining the smaller size grinding media should be filled in close to the discharge end and the larger ones close to the inlet end of the compartment to favor the initial classifying effect. In larger mills a material cushion should be filled in prior to recharge the grinding media for protection of the linings.

7. OPTIMIZATION MEASURES

7.1 Short-term Measures

In most cases immediate adaptations can be carried out by the own maintenance crew under utilization of available equipment and spares from the plant. The measures are very economical as a certain success can be achieved with low costs.

7.2 Medium-term Measures

Larger modifications require usually more detailed investigations and discussions with specialists. Delivery times of parts and equipment lead to more extended project periods. It is obvious that such modifications call for some investments and that is where many optimization programs come to stop. It is then the engineers duty to convince the management of the advantages of such optimization measures!

7.3 New Technologies

The fast development in grinding technologies lead often to a point where investment for a new equipment is more advantageous than a modification. Respective investigations may answer the economical aspects of such measures.

Fig. 15 shows the procedure for the execution of a mill optimization program.

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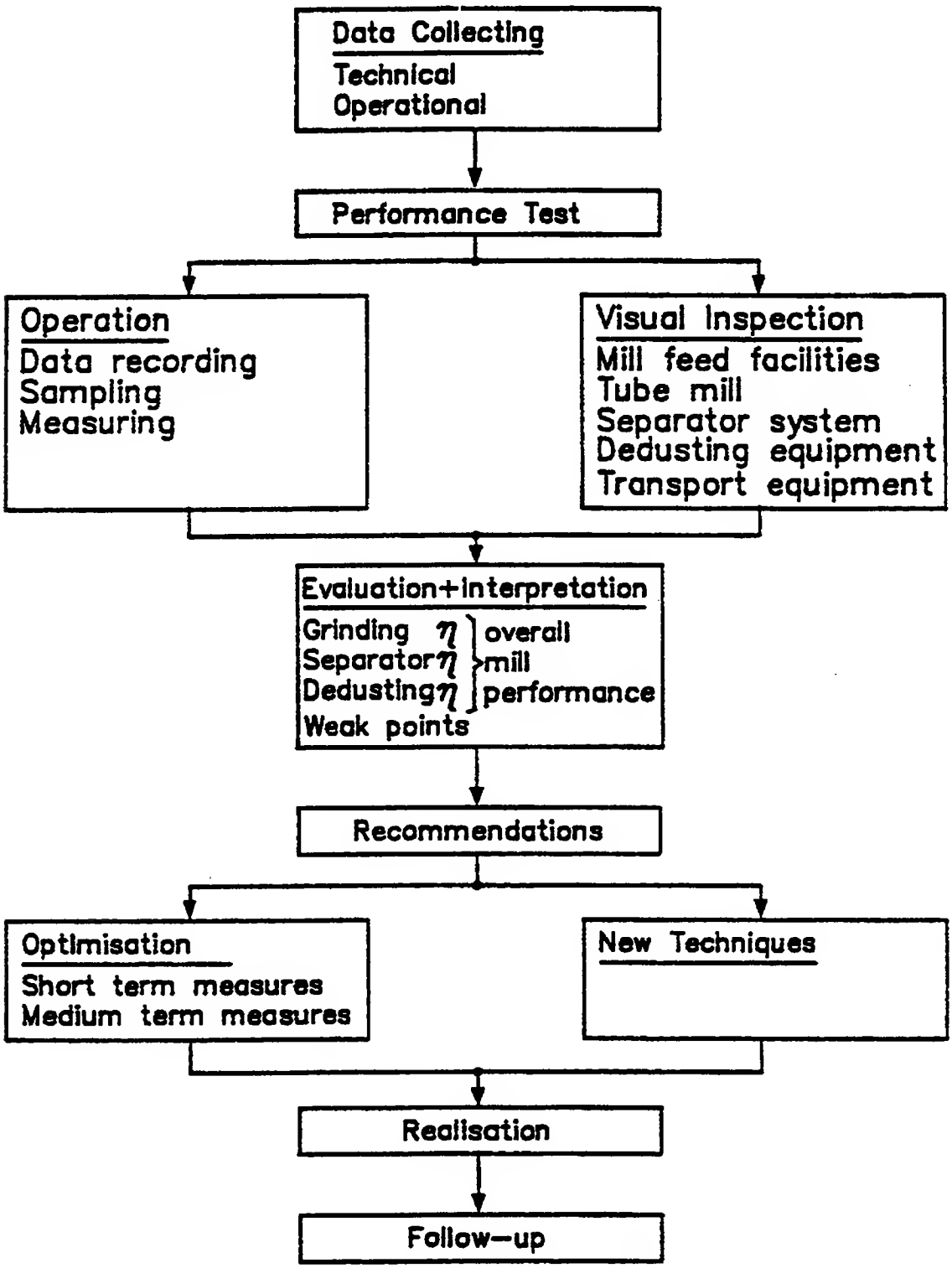
Figure 15 Mill Optimization Procedure



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8. START-UP OF A NEW MILL

The start-up of a new tube mill should be done in several steps with increasing grinding media charge fillings and running in times. The steps can be summarized as follows as a general rule for cement mills:

Figure 16 Steps for Running-in Procedure

Step	% of charge	Running time	Media size	Observations
1	5	15 min.	max. 70 [mm]	
2	50	25 hours	max. 70 [mm]	with material feed
3	75	75 hours	max. 80 [mm]	with material feed
4	90	100 hours	max. 90 [mm]	with material feed
5	100	---	---	normal operation

Prior to charging the mill with grinding media, an amount of material should be filled in for protection of the liners. It is also essential that the mill is not running without material feed except for the first 15 minutes.

During the start-up period the bolts for liner fixation should be retightened several times, for the first time after 8 - 12 running hours.

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Optimization of Grinding Systems
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slegten

CHARGES TYPES DEUXIEME CHAMBRE

Selon P.M.
pour un ciment 95% clinker- 5% gypse

M MICHELET

CIRCUIT FERME

BLAINE	2700 - 3000		3200 - 3500		4200 - 5500	
	charge	appoints	charge	appoints	charge	appoints
Ø60 mm	9	25	8,5	25	8,5	26
Ø50 mm	4,5		4		4	
Ø40 mm	3		2,5		2,5	
Ø30 mm	15,5	50	12,4	44	10,8	41
Ø25 mm	28,6		24,65		23,23	
Ø20 mm	19,2	25	16,45	31	15,49	33
Ø17 mm	20		31,5		35,48	
	100%	100%	100%	100%	100%	100%
poids moyen g.	45,31		38,47		36,61	

CIRCUIT OUVERT

BLAINE	2700 - 3000		3200 - 3500		> 3500	
	charge	appoints	charge	appoints	charge	appoints
Ø60 mm	8,5	30	8	29	5	24
Ø50 mm	5		5		4	
Ø40 mm	3		2		2	
Ø30 mm	5,72	32	4,05	28	2,33	22
Ø25 mm	19,44		16,66		11,67	
Ø20 mm	13,89	38	11,91	43	8,33	54
Ø17 mm	18,33		15,71		11	
ø15 mm	26,12		36,67		55,67	
	100%	100%	100%	100%	100%	100%
poids moyen g.	27,61		24,20		19,73	

Chapter 8

Roller Mills

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Roller Mills

Hp. Fisch
VA 94/4167/E

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1. INTRODUCTION

The vertical roller mill is the appropriate equipment for grinding and drying of wet materials. Grinding and drying can very efficiently be executed in one machine:

Many materials can be ground in vertical roller mills such as:

* raw materials
* coal
* pozzolana / trass
* slag
* (cement)

This paper deals in first place with roller mills for raw material processing. For grinding coal, slag, cement, etc., adaptations to mills and systems have to be made.

2. FUNCTIONS OF THE ROLLER MILL

The vertical roller mill fulfills 4 main functions in one compact piece of equipment:

- Grinding
- Drying
- Separation
- Transport

The process is called a drying / grinding process, where most of the material / product is transported pneumatically by drying gases.

- **Grinding:**
  The material is ground between rollers and grinding table while passing from the center of the table to the nozzle ring. The comminution method belongs to the most efficient grinding processes applied in the cement manufacturing.
- **Separation:**
  The ground and dried material is lifted up with the drying gases. In the separator, the too coarse particles (tailings) are rejected to the grinding table. The fines leave the mill and are conveyed to a dust collector.
- **Drying:**
  The process air consists mostly of waste gas from a kiln or cooler or is supplied by a hot gas generator. Drying occurs during transport through the grinding and separating process stage.
- **Transport:**
  The drying gases are utilized to serve as conveying media. The first transport stage is the internal circulation and the second the separator. At last, the product is extracted from the separator and pneumatically conveyed to cyclones or a filter where the product is collected and fed to a silo. The clean gases are exhausted to the ambient and/or recirculated to the mill.

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Figure 1 Functions.

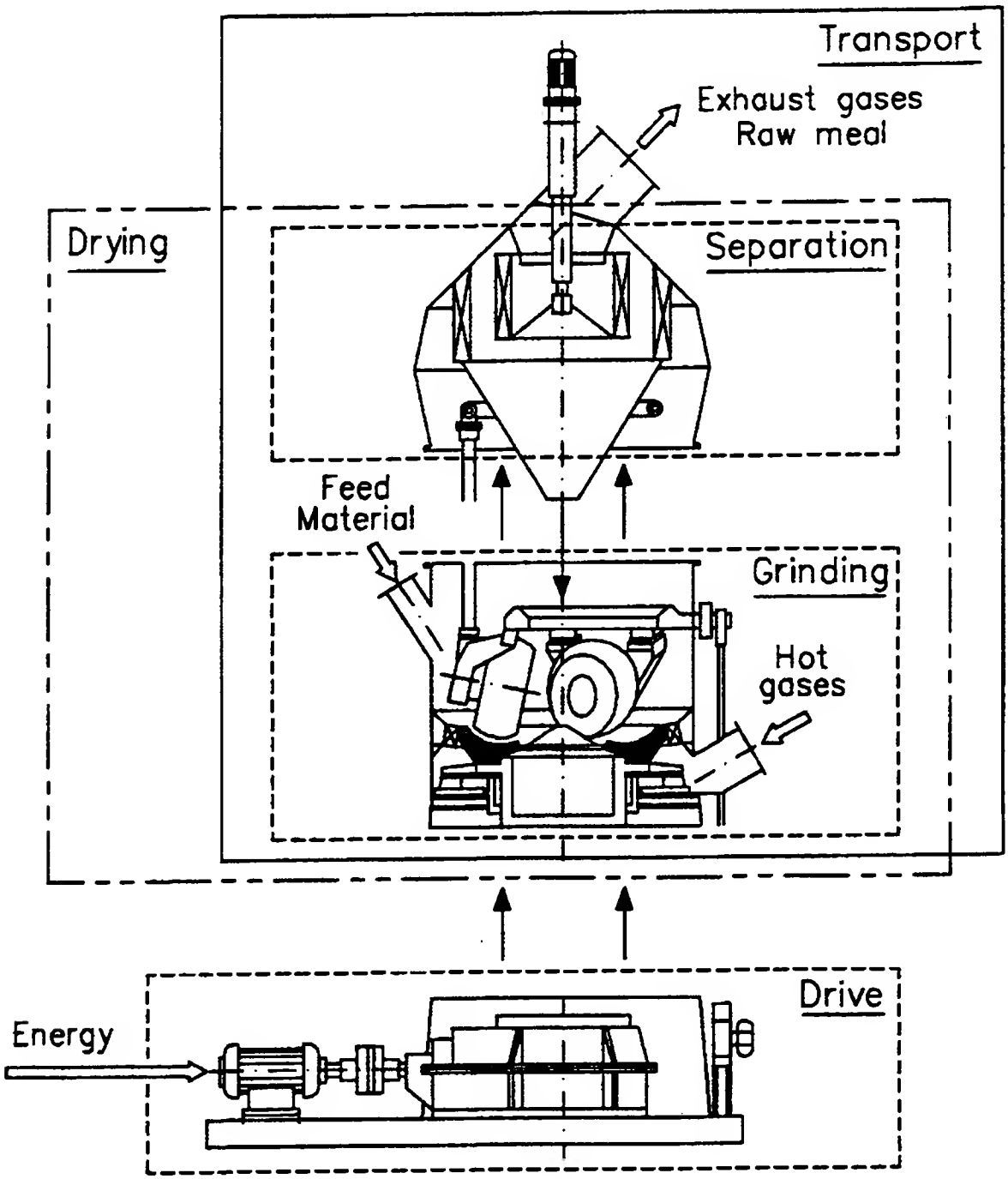

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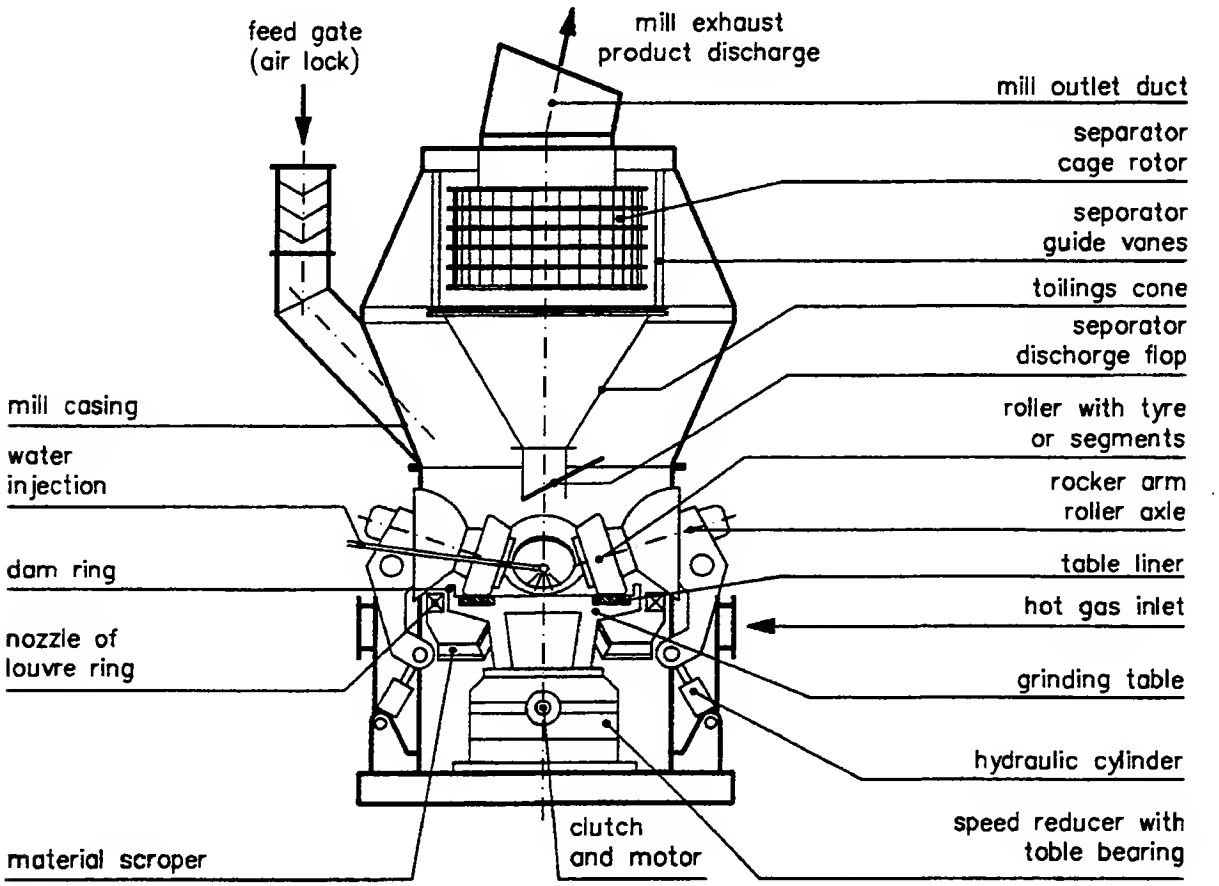
Figure 2 General Design.



3. GENERAL DESIGN AND WORKING PRINCIPLES

- **Material feed:**
  The wet material is fed through an air sealed feed gate onto the grinding table. The feed gate has to avoid false air inlet to the mill and seal against a fairly high negative pressure in the mill body. For wet and sticky materials triple gates (heatable) are preferred, for general use rotary airlocks are commonly applied, depending on supplier's philosophies.

- **Grinding:**
  The material (fresh feed, recirculating material and separator tailings) passes from the table center under the rollers. The material is drawn in-between roller and grinding track and is comminuted. Depending on the roller diameter, table speed, roller pressure and the material characteristics (granulometry and properties) a certain max. particle size can be drawn under the rollers (max. size = 5 - 8 [%] of roller diameter). Higher bed thicknesses require higher grinding pressures and thus absorb more power. More power is also absorbed with increasing material moistures.
  The ability to form a stable grinding bed is essential for a stable mill operation. Dam rings are often utilized for adjustment of the grinding bed thickness. They serve as retention device for the material on the grinding table.

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- **Material circulation:**
  The material, which flows over the dam ring is caught by the vertical gas flow from the nozzle ring and lifted up. Coarser particles fall back to the grinding table and finer ones are swept up to the separator for being classified. The internal circulation rate depends mainly on the grindability of the ground material and can amount up to 15 - 25 cycles. Reducing the gas speed in the nozzle ring (adjustment of the open area) leads to falling through of larger particles. The fall through material has to be extracted with scrapers and mechanically recirculated to the mill feed.

- **Separation:**
  The use of modern separators in roller mills is state of the art. A sharp separation improves the raw meal quality and avoids over grinding (saving of energy). Coarse tailings, fed through the tailings cone to the center of the grinding table, helps the formation of a more stable grinding bed. The raw meal fineness is easily controlled by adjustment of the cage rotor speed.

- **Drying:**
  Drying occurs mainly where the hot gases exit the nozzles and enter in contact with the moist material. Fine particles have an extended retention time within the drying gases (up to the dust collector), which ensures a good drying performance. The drying rate, resp. drying need is directly reflected in the mill exhaust temperature.

- **Water injection:**
  At certain conditions, roller mills need water injection to stabilize the grinding bed. Injection nozzles should spray onto the material in front of each roller, proportioned according to the throughput.
  Water injection facilities with dosing valves can be installed in the mill casing to cool down the kiln gases in case of excess gas temperature and lack of material moisture.

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Figure 3 Working Principle.

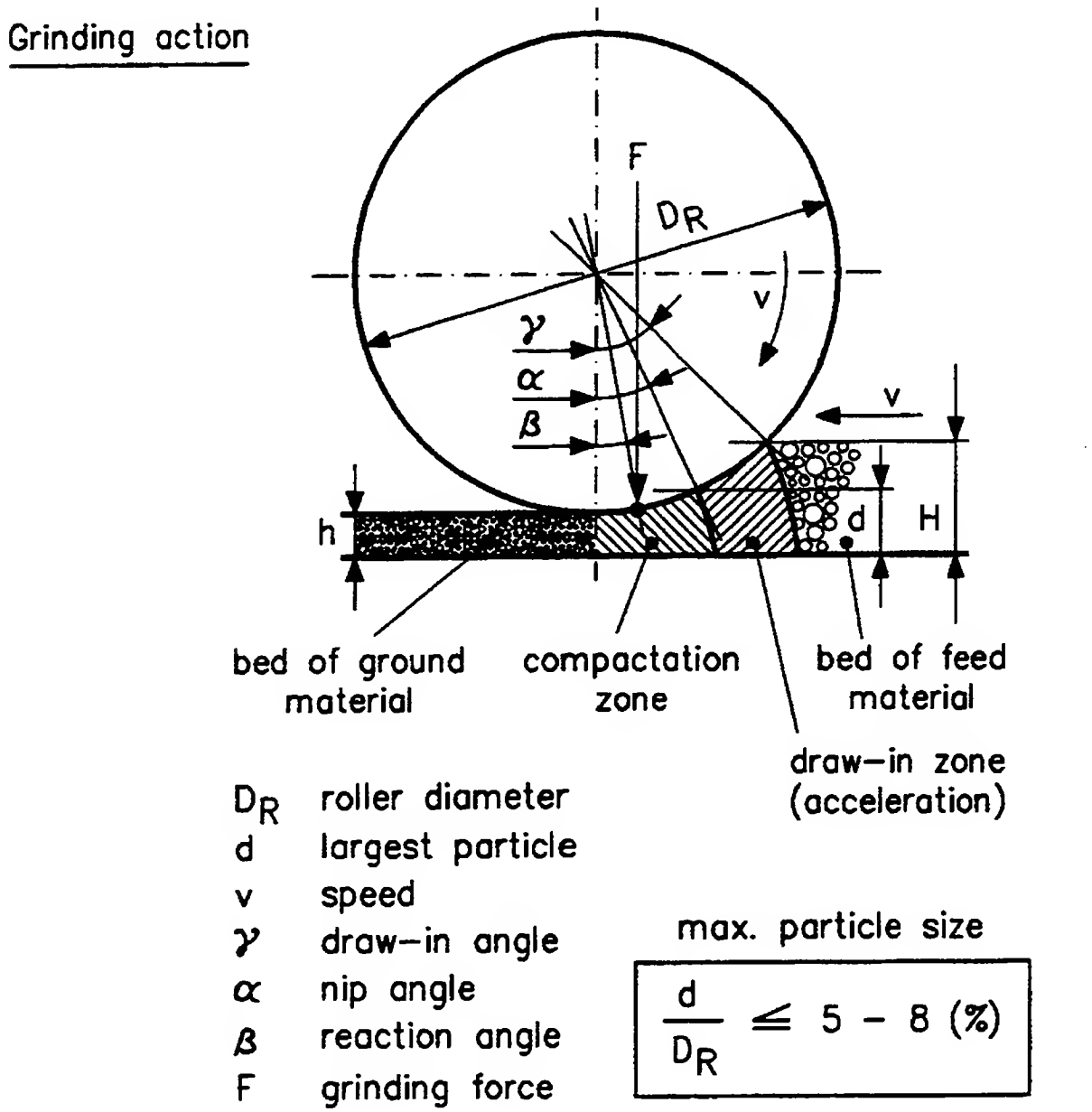

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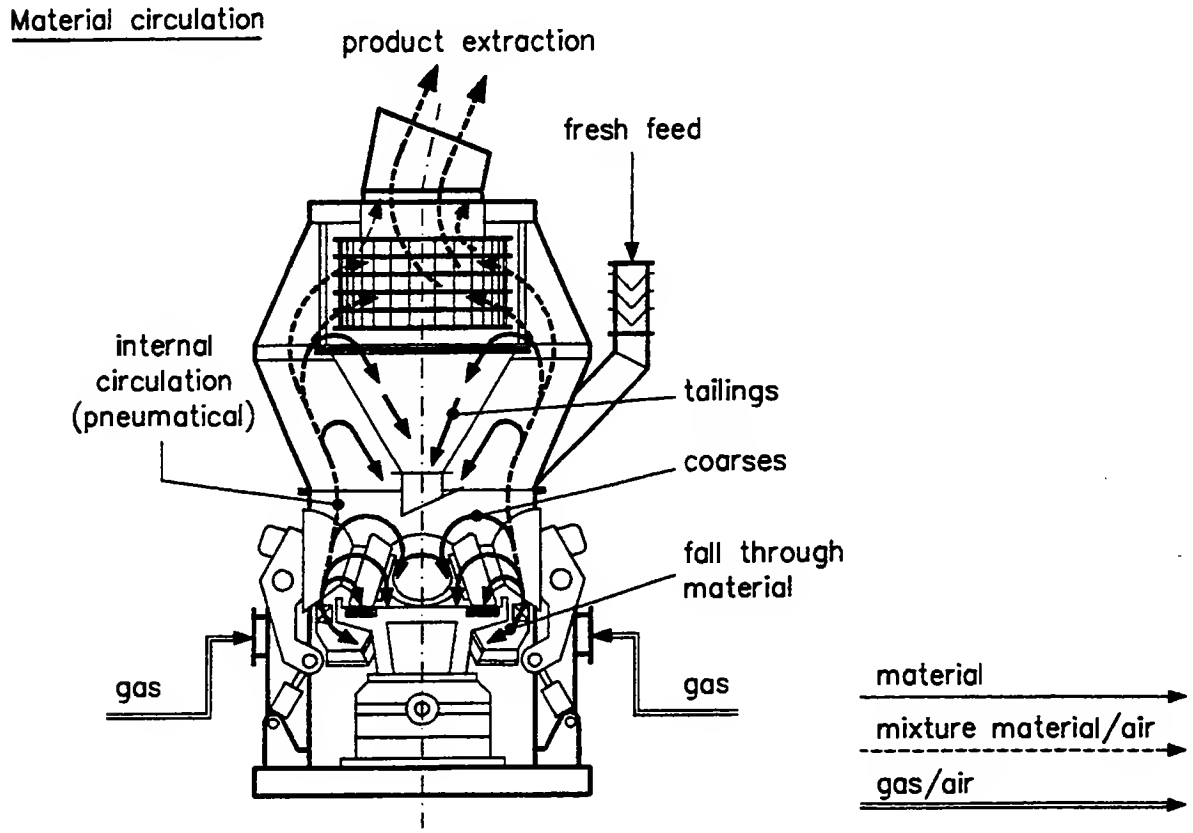
Figure 4 Internal Circulation.



4. DIMENSIONING AND DIMENSIONING CRITERIA

Two main aspects have to be considered when sizing a vertical roller mill:

* grinding
* drying

Which ever of the two is critical determines mainly the size of the mill.

- For the grinding needs, the grindability of the processed material (as a function of the material properties and the product fineness) requests a certain size of grinding table and rollers, as well as eventually the table speed.
- For the drying, the necessary gas flow rate (as a function of material moisture and available gas temperature) determines the casing diameter, the open section of nozzles and the separator size.

Mill throughput $\dot{M}$: is proportional to $D_M^{2.5}$ for given conditions and mill type.

Mill power absorbed P_{abs}: is a result of the required mill throughput and the grindability of the material under given conditions. The mill size is determined by solving formula 2) for D_M. The friction factor and the spec roller pressure are empirical values gained through experience and/or lab tests. The mill parameters are supplier specific and given in chapter 7. The absorbed power increases with higher material moisture, higher press forces and higher grinding bed thickness.
Installed power $P_{inst} = 1{,}1 \times P_{abs}$ [kW]

Specific roller pressure k: is defined by division of the total roller press force **F** by the projected roller section $D_R \times W_R$. Each mill type has its operating range. The achieved spec. roller pressure is related to the operating conditions.

Roller pressure force F: consists of the proper roller weight **G** and the exerted force by the hydraulic pressure device F_2. Force F_1 applies for rocker arm type mills, where the force from the hydraulic cylinder has to be converted to F_2 with the lever arm ratio.

Table speed n: is a function of D_M for a constant centrifugal force of the material on the grinding table. Typical speed constant **c** are given for each supplier. In certain cases, lower speeds are used for fine and dry feed materials.

Drying capacity: In general, the maximum drying capacity is around 20 [%] H_20. For higher moisture contents a predrying facility is necessary. The required gas flow rate through the mill has to be calculated e.g. through a heat balance. The maximum admissible mill inlet temperature is < 450 [C].

Gas speeds: The mill has to be sized to meet the required speed ranges:

- Nozzle ring: ≥ 30 [m/s], higher speeds can be adjusted by covering nozzles
- Mill casing: 4.5 - 7 [m/s] for vertical transport
- Separator: 4.5 - 6 [m/s] through the cage rotor (gross area)

Dust load: The gas flow rate must be selected to achieve a dust load of 500 - 600 [g/m^3] raw meal at the mill outlet. The never exceed value for the separator and dust collector is 750 [g/m^3].

Separator speed: should be adjustable in the range of 10 - 25 [m/s] periferical cage rotor speed for raw mills.

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Figure 5 Dimensioning

Mill throughput $\dot{M}$:

$$\dot{M}=\frac{P^{abs}}{q}$$ (t/h) ①

Absorbed mill power P_{abs}:

$$P^{abs}=i\cdot\mu\cdot k\cdot D_R\cdot W_R\cdot D_M\cdot\pi\frac{n}{60}$$ (kW) ②

Specific roller pressure k:

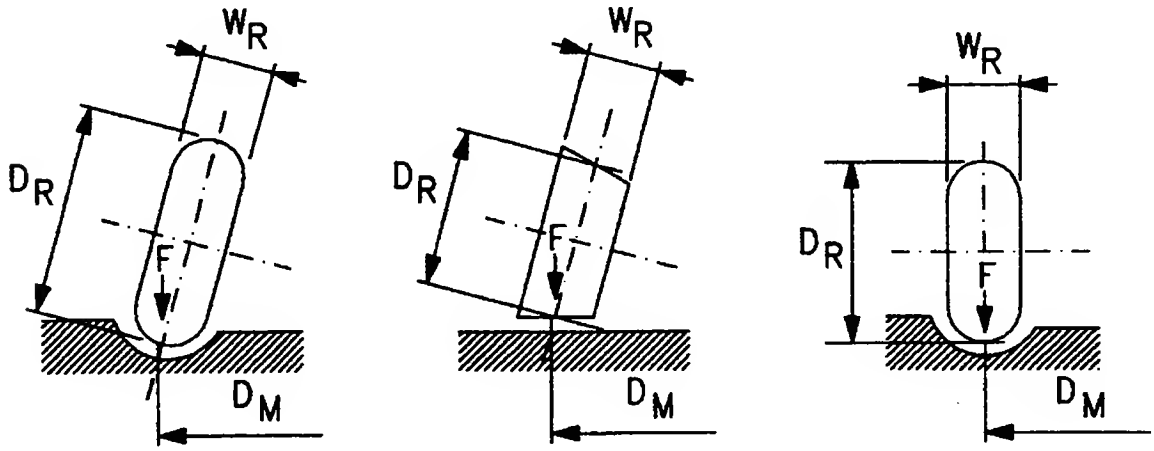
$$k=\frac{F}{DR\cdot WR}$$ (kN/m^2) ③

Roller press force F:

$$F=F_2+G_R$$ (kN) ④

Table speed n:

$$n=\frac{c}{\sqrt{D_M}}$$ (min^{-1}) ⑤



Symbol explanation

Symbol	Unit	Designation	Symbol	Unit	Designation
$\dot{M}$	(t/h)	Mill throughput	D_R	(m)	- roller
q	(kWh/t)	Spec. energy consumption	W_R	(m)	Roller width
P_{abs}	(kW)	Mill power, absorbed	F	(kN)	Roller press force
I	(-)	Number of rollers	F_1	(kN)	Mech. force at hydr. piston rod
μ	(-)	Friction factor	F_2	(kN)	Mech. force at roller
k	(kN/m^2)	Spec. roller pressure	G_R	(kN)	Roller weight
D	(m)	Diameter	c	(-)	Speed constant (supplier specific)
D_T	(m)	- table	n	(min^{-1})	Speed of grinding table
D_N	(m)	- nominal	v	(m/s)	any velocity
D_M	(m)	- grinding track			

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5. OPERATIONAL ASPECTS

Grinding:
During operation the prime objective is to ensure an optimum and stable grinding bed which allows the optimum grinding production at the lowest absorbed mill power. The main influence factors are:

- **Material granulometry**: - A mixture of coarse and fine particles form a stable material bed. Too coarse material (car on gravel road) causes rough roller movements. Too fine material (car on a sandy ground) tends the rollers to slide or slip.
  Too dry material is usually moistened by water injection to form a compact bed.
- **Roller pressure**: - High pressures yield a higher comminution work in one material passage and hence lead to lower circulation loads and vice versa. Pressure and circulating loads influence the particle size distribution of the product, higher circulating loads produce wider and lower circulating loads narrower particle size distributions. At higher pressures also the absorbed motor power increases. The optimal conditions have to be established by trials and experience.
- **Dam ring**: - The dam ring allows an adjustment of the bed height and is more important for mills with flat tables to retain the material on the grinding table. As the wear at rollers and table segments proceed, the grinding bed height increases. To keep the bed height constant, the dam ring height needs then to be lowered. When exchanging roller and table wear parts, the dam ring has generally also to be renewed accordingly.
- **Louvre ring**: - Based on the rollers arrangement and table design, the material reaches in uneven flows the louvre ring. By proper adjustment of the nozzle openings (covers, inserts, adjustment devices) more gas can be guided through the nozzles where more and coarse material has to be lifted up.

External circulation:

- The material is lifted up over the louvre ring by the gas flow and pneumatically recirculated. To lift up the larger particles, too much gas velocity is needed which causes high pressure loss across the louvre ring.

In order to reduce the pressure drop in this area, the open nozzle section is opened to provide speeds in the range of 30 - 50 [m/s] through the nozzles, with the result of reducing the fan power. As a result, coarse particles fall through the nozzles, have to be collected by a scraper attached to the rotating grinding table, and are lifted up mechanically (bucket elevator) to the mill feed. Bucket elevators are dimensioned for the same capacity as the mill throughput (100 [%]), even if during normal operation a lower rate is externally recirculated.

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Figure 6 Grinding Aspects.

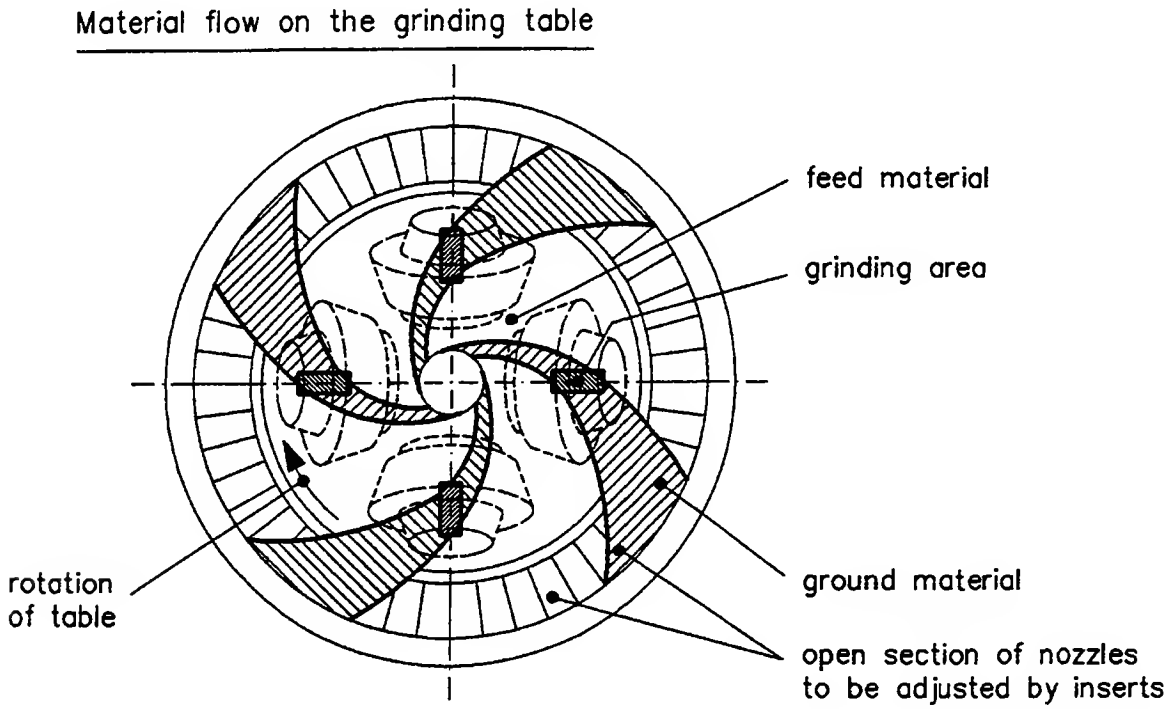

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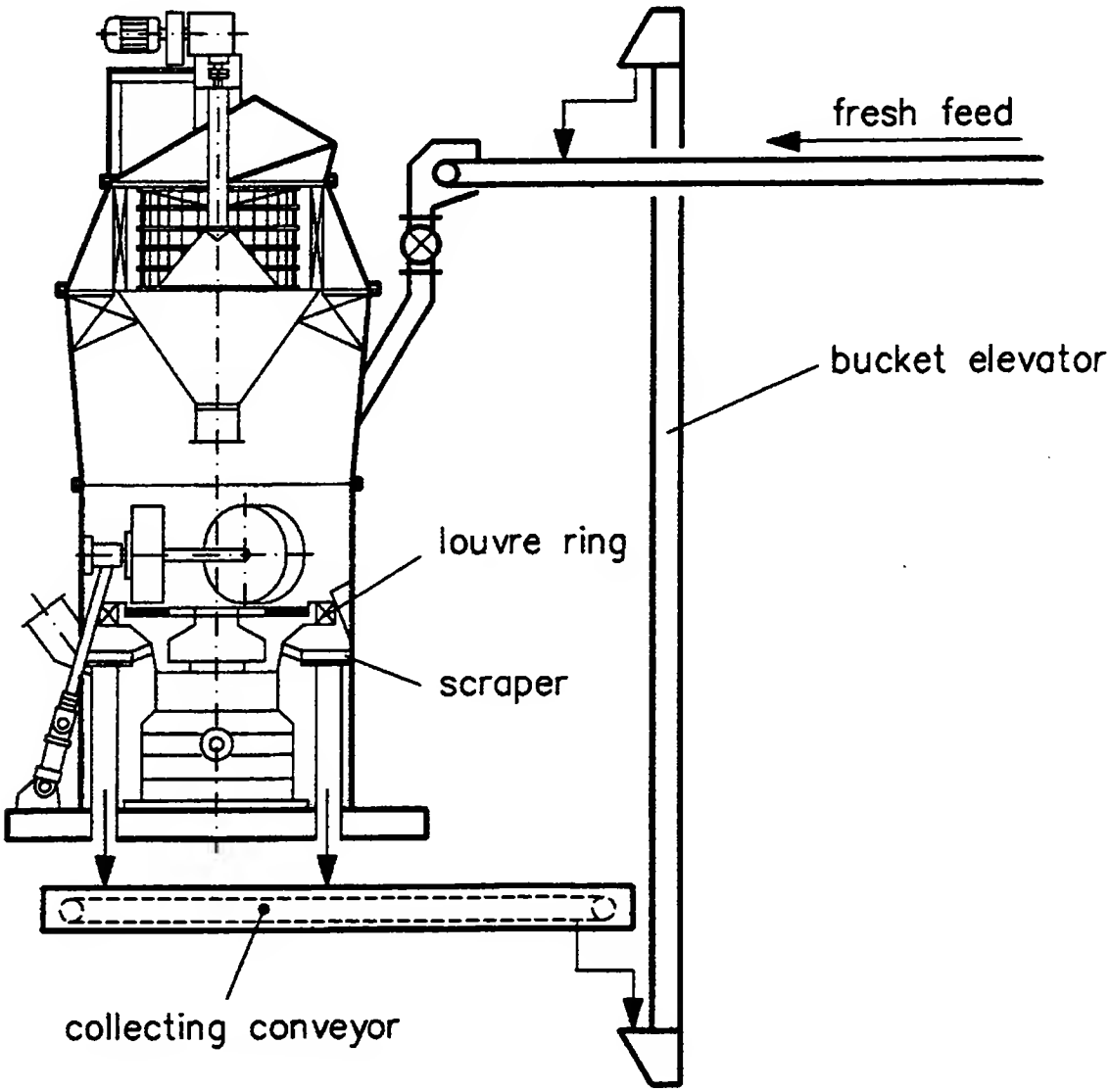
Figure 7 External Circulation.



Separation:

- The separator has to classify the material lifted up from the grinding table. The separator acts mainly in the coarse fraction of the product. The fines are produced by the grinding process. The target is to produce a narrow particle size distribution (PSD) with a low amount of coarse (% R 200 µm) and fine particles (% < 10 µm).
- There are three generations of separators:
  1. Generation: Static separators
  2. Generation: Conventional rotor type separators
  3. Generation: Cage rotor separators with guide vanes
- **Static separators** are cheap and simple equipments, but produce a wide PSD. Fineness adjustment is done by adjusting the direction of the vanes. The efficiency drops with increasing fineness (more tangential vane position, higher pressure loss, lower gas flow rate due to fan characteristic).
- **Conventional rotor type separators** adjust the fineness by the rotor speed. They produce a wide PSD. Low pressure loss. Tailings fall spreaded over the grinding table.
- **Cage rotor separators with guide vanes** are based on the same considerations as commonly used for cement grinding, but have to operate with higher air speeds through the cage rotor (4.5 - 6 [m/s]) for raw material (speed related to the gross cylindrical cage area (D x π x H). The guide vanes are positioned during the commissioning phase and then remain fixed.
  In new roller mills only cage rotor separators with guide vanes should be installed as they produce a narrow PSD, represented as a steep slope in the range of 90 - 200 [µm]. Good ratios [%] R 200 [µm] / [%] R 90 [µm] are between 0.02 - 0.05 [-]. This means, while the target residue on 90 [µm] can be maintained, a much lower residue on 200 [µm] can be achieved. As in general, the quartz is found in the coarse fraction, a lower residue on 200 [µm] points to finer ground quartz particles, which in turn favors the burnability.

  Increased gas speeds through the separator rotor provoke narrower, lower gas speeds wider PSD, resp. steeper or flatter slopes in the RRSB-chart. The PSD is also strongly influenced by the grindability of the material and the applied roller pressure.

  On the other hand, cage rotor separators, due to the better classifying efficiency, produce less fines (= saving of energy) and produce coarser tailings (= more stable grinding bed).

Drying:

- The gas flow rate through the mill, resp. the gas mass, has to be maintained constant, to keep stable grinding/separation conditions. As the gas volume changes with the temperature (gas density), it is important to keep the mill exit temperature constant.

  When processing material with higher moisture content, the mill exit temperature must be higher, and lower for material with less moisture.

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Figure 8 Separators.

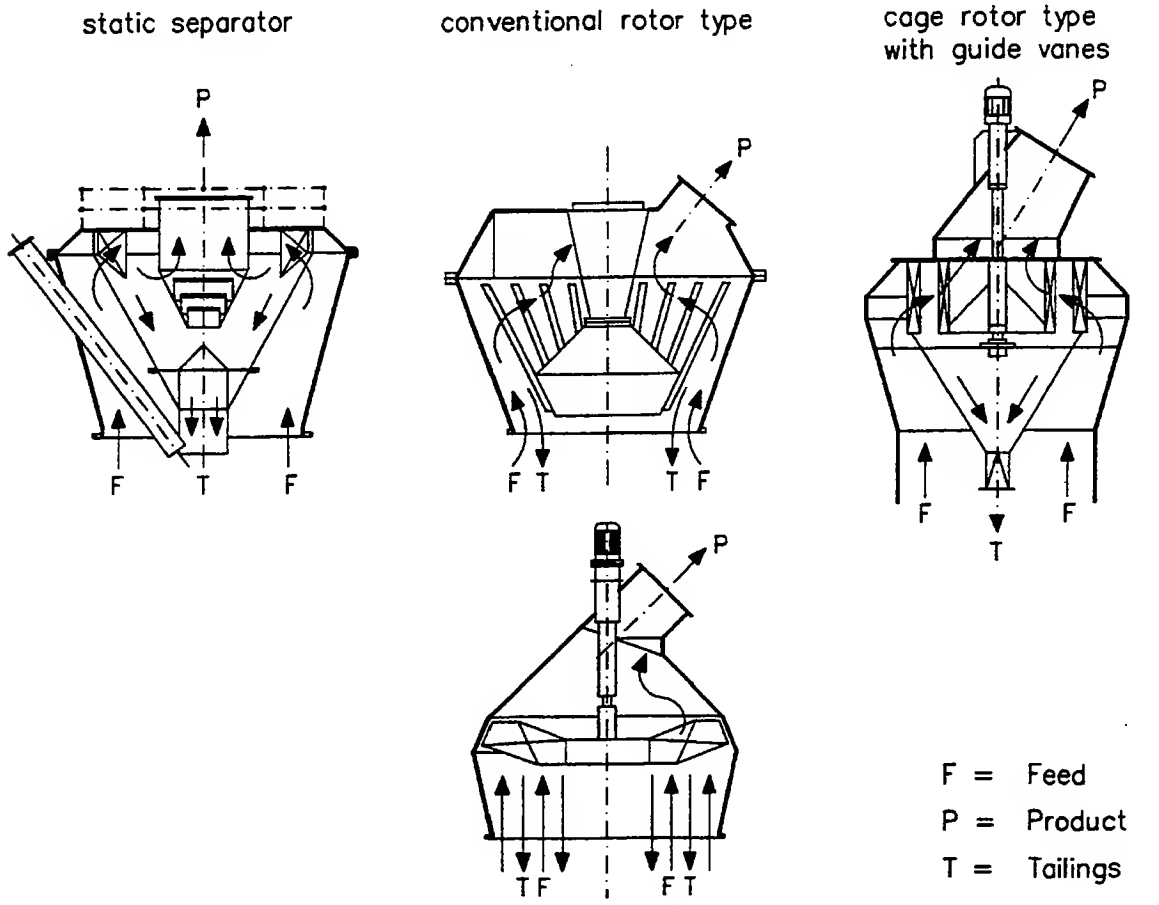
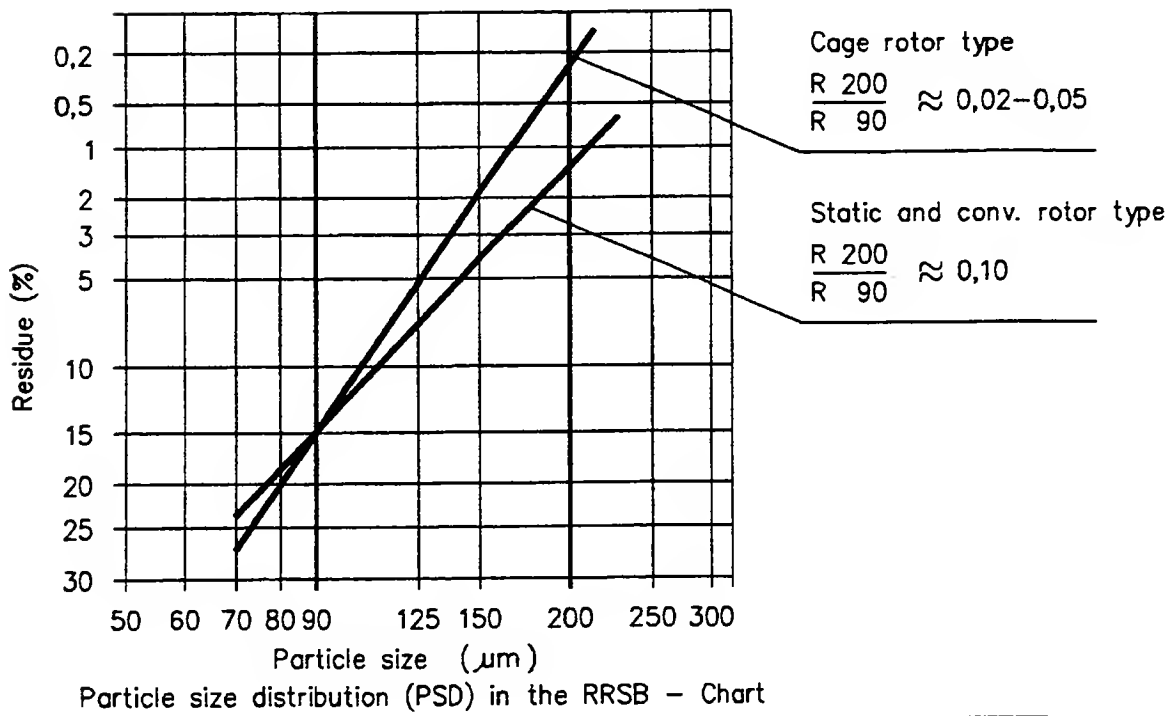


Figure 9 Particle Size Distribution.



Particle size distribution (PSD) in the RRSB – Chart

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6. ROLLER MILL PERFORMANCE

6.1 Mill Throughput

For a given mill, the mill production rate depends mainly on:

* produced fineness
* grindability of feed material
* conditions of mill and system
* mill operation and control

Assuming that the feed material and its properties are given, the mill production rate changes in first order with changing fineness. The finer the product, the lower the possible mill throughput. On the other hand the feed granulometry to the mill has in general little or no influence on the production rate, specially in the coarse range. The impact of the feed granulometry is strongly operational (distribution of particles).

6.2 Mill Energy

The spec. mill energy consumption is a result of the absorbed mill power divided by the mill production. The absorbed mill power was discussed before.

The spec. energy consumption varies largely with the produced fineness. From the graph (fig. 10) the tendencies can be seen in case of raw materials. The graph shows the dependencies of the spec. energy consumption in relation to fineness and material hardness (grindability ranges for raw materials as defined by "Holderbank"). Spec. energy consumptions for roller mills are lower than for tube mills as the process is more efficient. Further the curves for roller mills are flatter than for tube mills, this means, the spec. energy consumption varies less for the same change of fineness.

6.3 Partial Load

The tendency in modern cement plant concepts is to adapt the raw mill production rate to the needs of the kiln. Therefore the roller mill often operates at partial loads. At lower mill throughputs the spec. energy consumption increases.

6.4 Fan Energy

The energy consumption of a roller mill system consists mainly of two large consumers, the mill and the mill fan. The energy consumption of the mill fan is generally in the same order of magnitude as for the mill itself. The fan energy can be influenced by two prime factors:

* gas flow rate
* pressure loss over mill and system

The gas flow rate can hardly be changed, therefore the pressure loss is predominant for the portion of energy consumed, where the pressure loss over the mill is the most important. The pressure loss of large mills increases compared to small mills. A maximum effort should be made to lower the pressure losses. Fig. 11 shows a normal range of pressure loss for vertical roller mills for raw material.

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Figure 10 Roller Mill - Performance / Raw Material

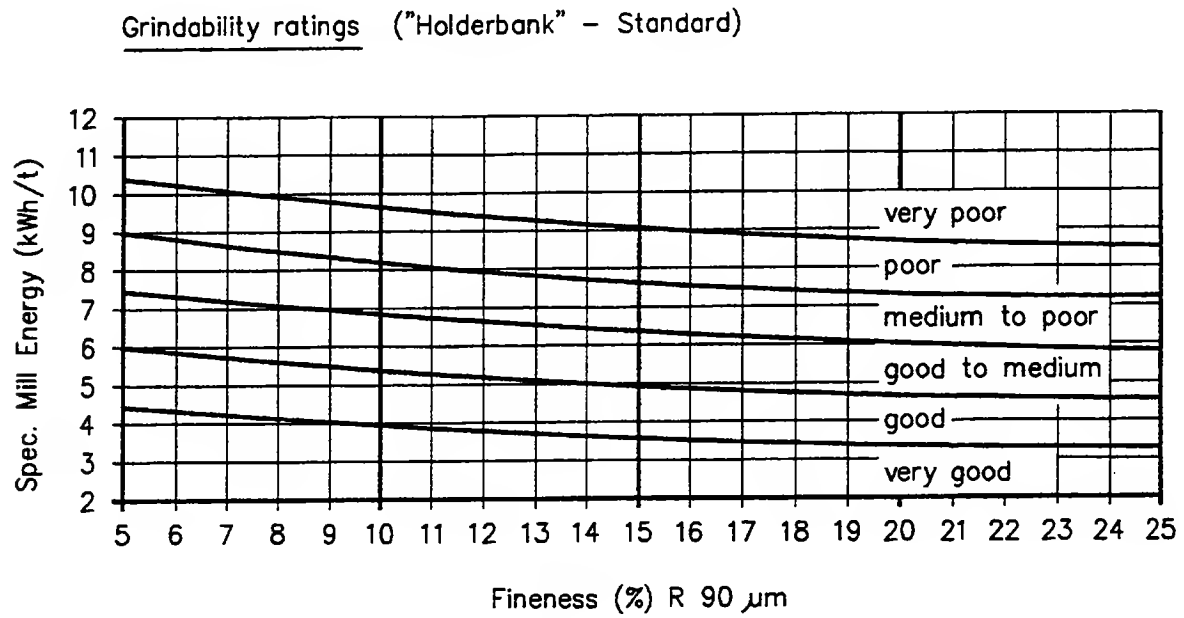
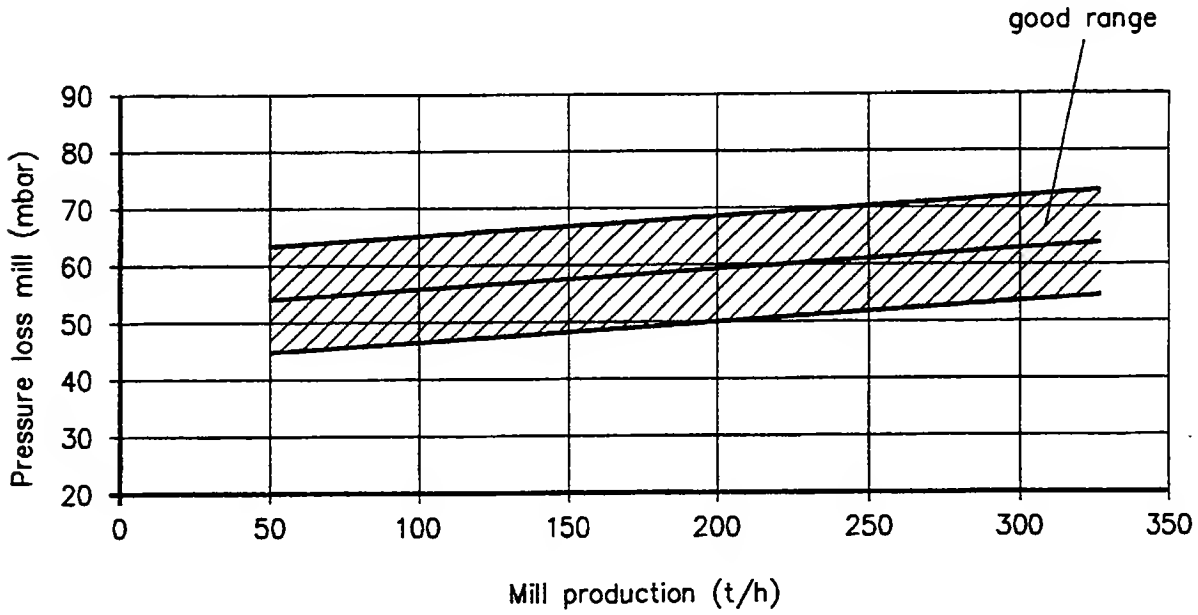


Figure 11 Roller Mill - Pressure Loss / Raw Material



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7. DESIGN AND FEATURES OF VARIOUS SUPPLIERS

7.1 Gebr. Pfeiffer (fig. 12)

- Make:

MPS 3750 A
- 3750 → Diameter [mm] *)
- A → Series

- Series: *)
  A - old series - Diameter = D_M (grinding track)
  B - new series - Diameter = D_B (outside grinding track)
  C - version for cement
  K - version for coal

- Application:
  - Raw material
  - Coal
  - Pozzolano / Trass
  - Cement

- Sizes: up to MPS 4850 A, 350 [t/h] raw material

- Dimensioning parameters:

D_M	= 0.8 x D_B	[m]
c	= 44.5 ... 47.5	[-]
k	≤ 450	[kN/m^2]
m	» 0.1	[-] raw material

- Design features:
  - Roller and grinding bed 15 deg inclined, concave grinding path
  - 3 rollers for all sizes
  - Rollers suspended at a pressure frame
  - Hydraulic actuated at a pressure frame
  - Pull rods inclined to absorb torque, mill housing not affected with horizontal forces
  - High-efficiency separator type SLS

- Operation:
  - Start-up with auxiliary drive, rollers statically on grinding table
  - Normally large size mill casing and low pressure losses of gas flow
  - Series B operates with higher gas flow rates and higher roller pressures than series A

- Maintenance:
  - Swing-lift device for retraction and servicing of the rollers

- Specialties:
  - Lower part of mill (without separator) to be used as a pregrinder for clinker in connection with a ball mill

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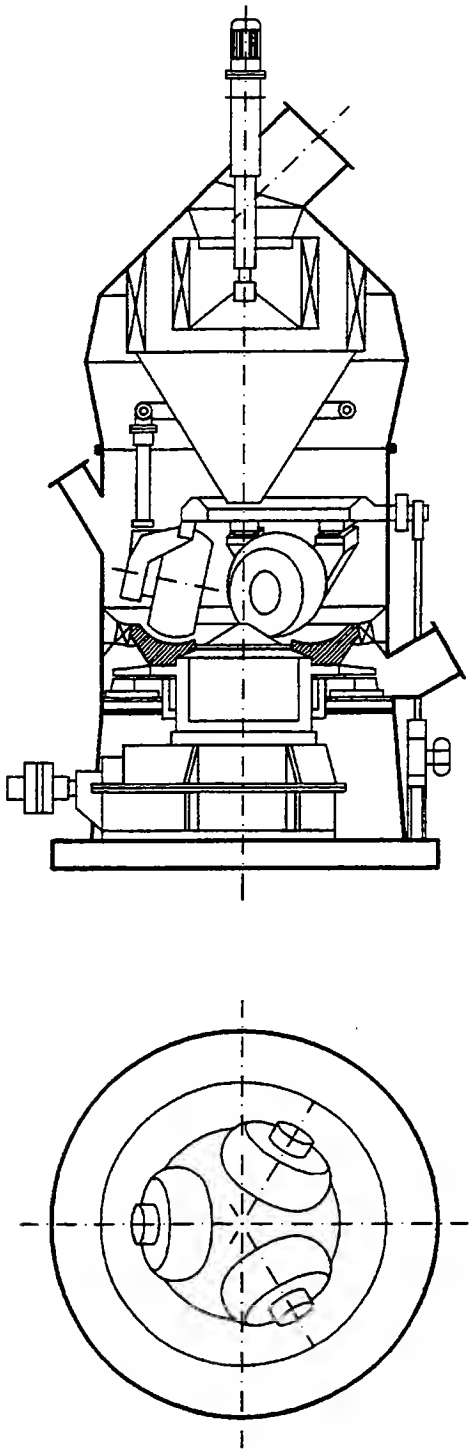
- Licensees:
  - Former license to FLS
  - Babcock
  - Allis-Chalmers

Figure 12 Gebr. Pfeiffer



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7.2 Loesche (fig. 13)

- Make:

LM	38.	4	50
|	|	|	↓
|	|	|	Module
|	|	|	
|	|	→	No of rollers (2,3,4)
|	—	→	Table diameter (nominal) in [dm]
—	—	→	Loesche-Mill

- Application:
  - Raw material
  - Coal
  - Pozzolana
  - etc.

- Dimensioning parameters:
  (Loesche mills vary in geometrical ratios for the different mill sizes.)

levers rocker arm	F_2	$\approx 0.8 \ldots 0.85 \times F_1$	[kN]
diameter grinding path	D_M	$\approx 0.81 \ldots 0.83 \times D_N$	[m]
speed constant	c	= 50.5 ... 53.5	[-]
spec. roller pressure	k	≤ 880	[kN/m^2] operation
friction factor	μ	≈ 0.1	[-] raw material

- Sizes: up to LM 59.4 for 700 [t/h] raw material

- Design features:
  - Airlock in general as triple gate, larger mills double triple gate
  - High-efficiency separator, jalousie type LJKS
  - Conical rollers 2, 3 or 4 according to mill capacity
  - Rocker arm with hydraulic cylinders to exert the roller pressure
  - Hydraulic cylinders with nitrogen accumulators
  - Modular design; hydropneumatic units can be combined with various mill sizes according to throughput or drying needs.

- Operation:
  - Rollers can be lifted from grinding track for low torque start-up of the mill
  - Two of four rollers can be unloaded for partial capacity operation of special operating conditions
  - Generally high pressure loss across mill, can be optimized with external circulation
  - Generally high false air rates due to rocker arm openings in mill casing

- Maintenance:
  - Practical swing-out device for roller servicing and replacement
  - Rocker arm sealings need carefully be maintained

- Licensees:
  - UBE Japan
  - formerly also Fuller, today expired due to take-over by FLS

- Future:
  Mill has been adapted, specially through UBE, for grinding blast furnace slag and cement

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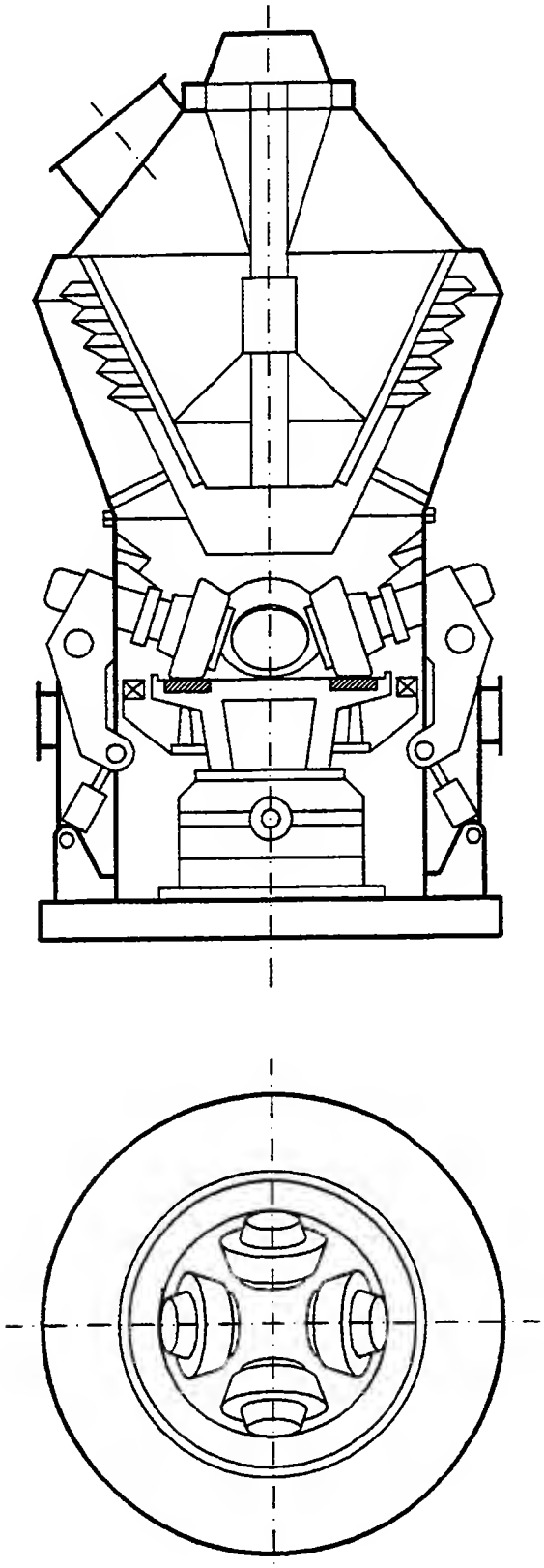
Figure 13 Loesche



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7.3 Fuller - Loesche (fig. 14)

- Make:

LM	38.	4	30
|	|	|	↓
|	|	|	Module
|	|	|	
|	|	→	No of rollers (2,3,4)
|	—	→	Table diameter (nominal) in [dm]
—	—	→	Loesche-Mill

- Application:
  - Raw material
  - Coal
  - Pozzolana
  - etc.

- Dimensioning parameters:
  (Fuller-Loesche mills vary in geometrical ratios for the different mill sizes.)

levers rocker arm	F_2	$\approx 0.8 \ldots 0.85 \times F_1$	[kN]
diameter grinding path	D_M	$\approx 0.81 \ldots 0.83 \times D_N$	[m]
speed constant	c	$= 50.5 \ldots 53.5$	[-]
spec. roller pressure	k	≤ 880	[kN/m^2] operation
friction factor	μ	≈ 0.1	[-] raw material

- Sizes: up to LM 59.480 for 700 t/h raw material
  Fuller has many different roller sizes for the same mill size. Therefore the dimensions of rollers have to be indicated for the calculation of a specific mill type.

- Design features:
  - In general the same as the Loesche mill
  - High-efficiency separators type O-SEPA (license of ONODA) are installed. Also features of the SEPAX separators are available since the take-over through FLS.

- Operation:
  - Mainly same as Loesche mills

- Specialties:
  Fuller offers often roller mill systems with fresh air dampers. These systems are generally very difficult to operate. It is recommended to install in any case a gas recirculation duct for a good controllability of the systems.

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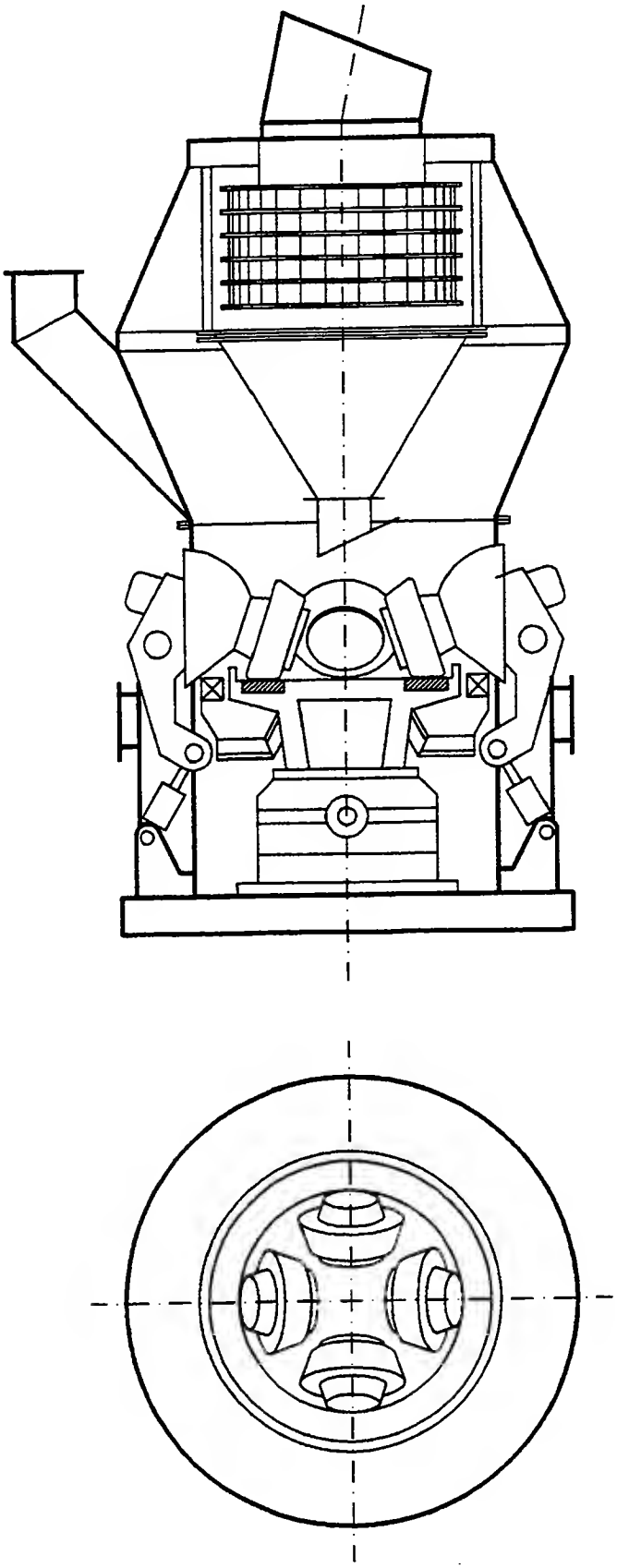
Figure 14 Fuller - Loesche



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7.4 FLS (fig. 15)

- Make:

  ATOX 50
  ↓
  Table diameter $D_T = D_N$ (nominal) [dm]

- Application:
  - Raw material
  - Coal

- Dimensioning parameters:
  Atox mills are totally geometrical:

D_M	$= 0.8 \times D_T$	[m]
D_R	$= 0.6 \times D_T$	[m]
W_R	$= 0.2 \times D_T$	[m]
c	= 50.1	[-]
k	= 800	[kN/m^2] dim. / ≤ 700 [kN/m^2] operation
μ	≈ 0.1	[-] raw material

- Sizes: available up to ATOX 60, 660 [t/h] of raw material, built up to ATOX 50

- Design features:
  - No. of rollers: always 3
  - Axles of the rollers interconnected in a rigid center yoke
  - Hydraulic cylinders with pull rods connected to each axle tip
  - Pull rods arranged outside grinding area
  - Roller axis with 3 degrees lag of the table center in rotating direction
  - Tangential thrust absorbing torque arms and bearings fixed to the mill body
  - High-efficiency cage rotor separator SEPAX type (RAR...) with adjustable guide vanes
  - Nozzles of louvre ring adjustable with inserts

- Operation:
  - Rollers can be lifted-up for start-up of grinding table
  - Compact mill housing with reduced false air leakage
  - Standard systems include a gas recirculating duct
  - External material circulation installed as normal

- Maintenance:
  - Roller segments uniformly worn due to flat rollers and table
  - Roller segments can be turned around
  - Roller segments exchange possible by lifting-up hydraulically rollers with yoke

- Future:
  - Tests with grinding of cement performed
  - FLS favors commercializing the OK mill (Kobe/Onoda) for cement grinding

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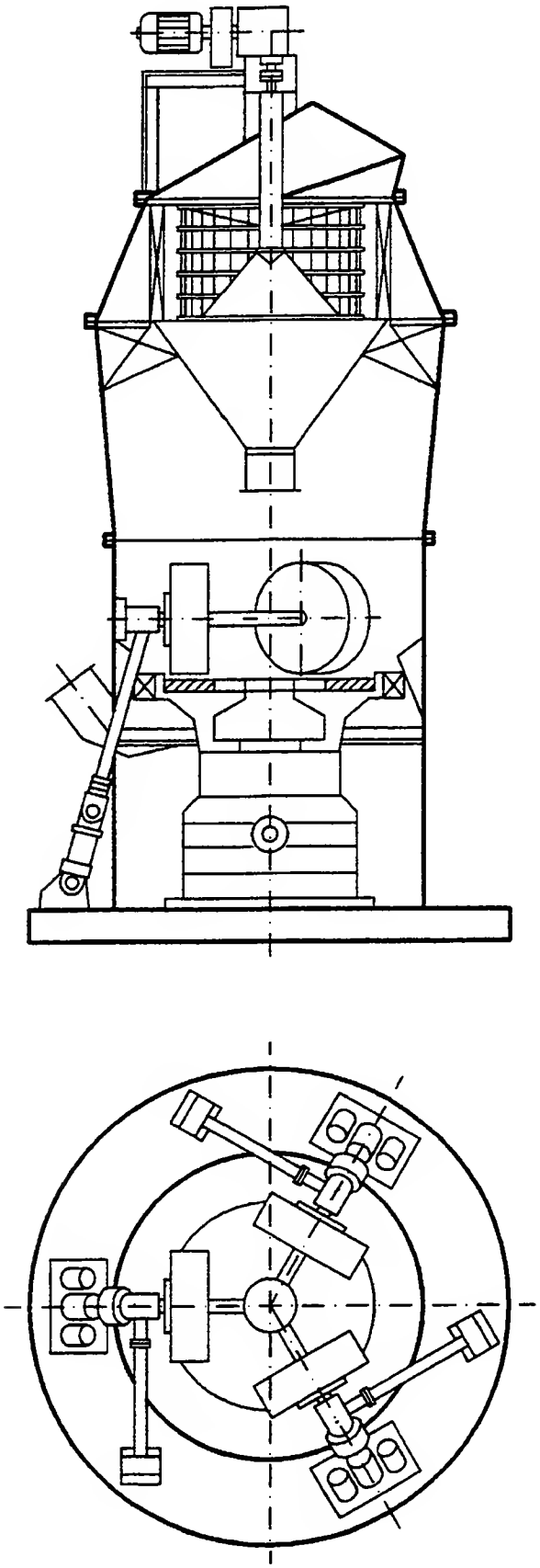
Figure 15 FLS - ATOX



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7.5 Polysius (fig. 16)

- Make:

RM 46/ 26/ 370
| | | ↓
| | | Diameter separator rotor [cm]
| | |→ Diameter rollers D_R [dm]
| |— → Diameter grinding table D_T [dm]
|
|— — → R-Raw Mill
. K-Coal Mill

- Application:
  - Raw material
  - Coal
  - etc.

- Dimensioning parameters:

D_{M1}	$\approx 0.75 \times D_N$	[m]
D_{M2}	$\approx 0.35 \times D_T$	[m]
c	= 38 ... 42	[-] related to $D_M = \frac{D_{M1}+D_{M2}}{2}$
k	≤ 1100	[kN/m²]
μ	≈ 0.1	[-]

- Sizes: up to RM 60/29

- Design features:
  - Mill feed sealing through rotary air lock
  - 2 twin rollers for all sizes with double grooved grinding table
  - Each roller pair with a yoke
  - Hydraulic cylinders with pull rods, fixed to hooks at yoke tips
  - Pull rods arranged outside grinding area
  - One pull rod is generally fix, the other is the working side which adjusts the roller pressure
  - High-efficiency separator type SEPOL

- Operation:
  - Start-up of mill with auxiliary drive, as rollers can not be lifted hydraulically
  - Nozzle ring openings adjustable to adapt gas speed to operating conditions (normally v = 20 ... 50 [m/s] adjustable)
  - External circulation preferred, installed capacity ≤ 200 [%] of mill throughput

- Maintenance:
  - Rollers (twin rollers with yoke) turnable for more even wear
  - Rollers retraction by crane through large doors in mill casing
  - Nozzles for louvre ring adjustable from outside
  - Pull hooks at the roller yokes adjustable in position for distribution of grinding force to the two rollers

- Future:
  - Roller mill tested and adapted for grinding of cement

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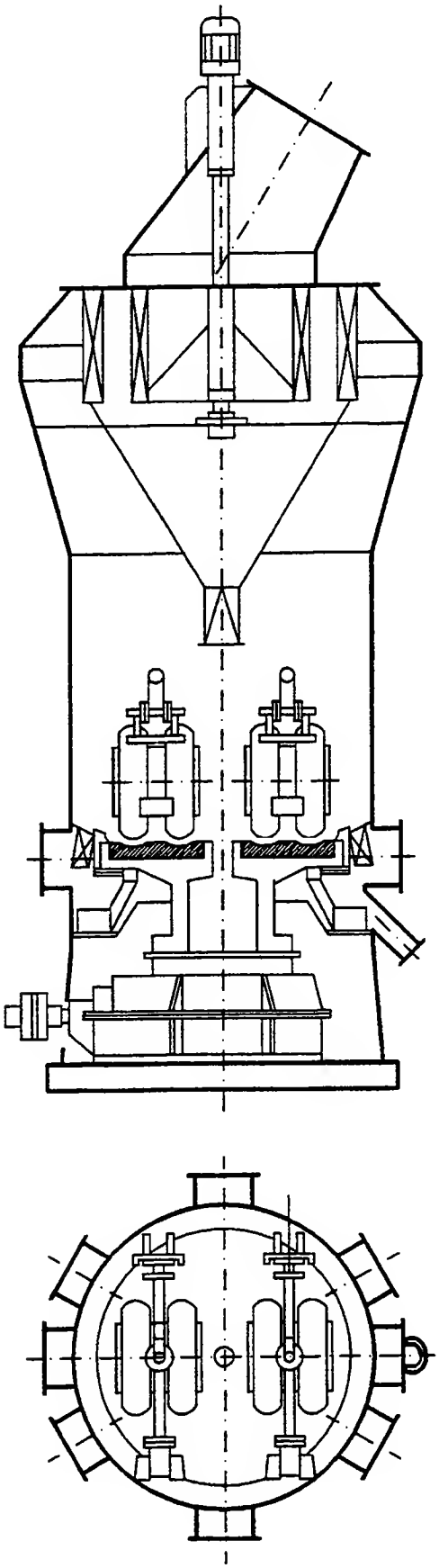
Figure 16 Polysius



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8. MAINTENANCE AND SERVICING

Today's roller mills for raw grinding have to run through for a long time (weeks, months) without interruption to cope with the kiln and plant's needs. The reliability has been greatly improved, so that usually only planned stops are necessary.

8.1 Wear

What concerns the wear of the grinding tools (roller tyres / segments and table liners) one does hardly talk about [g/t] of wear, but service lives of "years'. To change the main wear parts, a mill stop is needed. This should be one of the yearly stops. Spare rollers / tyres / segments and table liners are preferably in stock and can be exchanged / replaced upon a planned stop. In certain cases (e.g. Polysius, FLS) the rollers can be turned around after half the service life.

Roller exchanging facilities:

- Loesche, Fuller, Ube, Mitsubishi, Raymond:
  Rocker arm type mills add a servicing hydraulic cylinder in order to swing out the roller. The casing opening has to be prepared for. The swung-out roller can be lifted by an overhead crane.
- Polysius
  There are crane rails fix installed within the mill. The large side doors are opened, a connection of the rails with the outside crane is done and the crane hoist can enter the mill. The twin rollers form a unit together with a yoke. Yoke and rollers are lifted up and moved at a crane out of the mill. Sufficient space for the handling of the roller set (quite heavy for large mills) has to be provided.
- Gebr. Pfeiffer
  Pfeiffer mills can be fitted with a swing-lift device. A support with a central column is fitted to the rollers and the swing-lift device outside the mill. The swing lift device can haul the roller out of the mill where a crane will take over the roller.
- FLS
  A crane rail is introduced into the mill, directly over one roller. A hoist lifts the roller up and moves the load over the rail to an external rail connection to the servicing area.

8.2 Inspections

A continuous operation of a roller mill can only be achieved if the critical areas are regularly inspected and serviced.

- During operation:
  - Vibrations of gear drive and mill casing
  - bearing and oil temperatures
  - lubrication systems
- During stop:
  - wear parts (table, rollers, nozzles, dam ring, bearings exposed to material/gas flow, feed gate, etc.)
  - bearings, roller bearings, bearing sealings
  - air locks, sealing air system for bearings
  - lubrication systems (filters, oil conditions)
  - hydraulic pressure system, include. accumulators
  - gear device, clutch

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Figure 17a Servicing Facilities - Loesche, Fuller, UBE, Mitsubishi

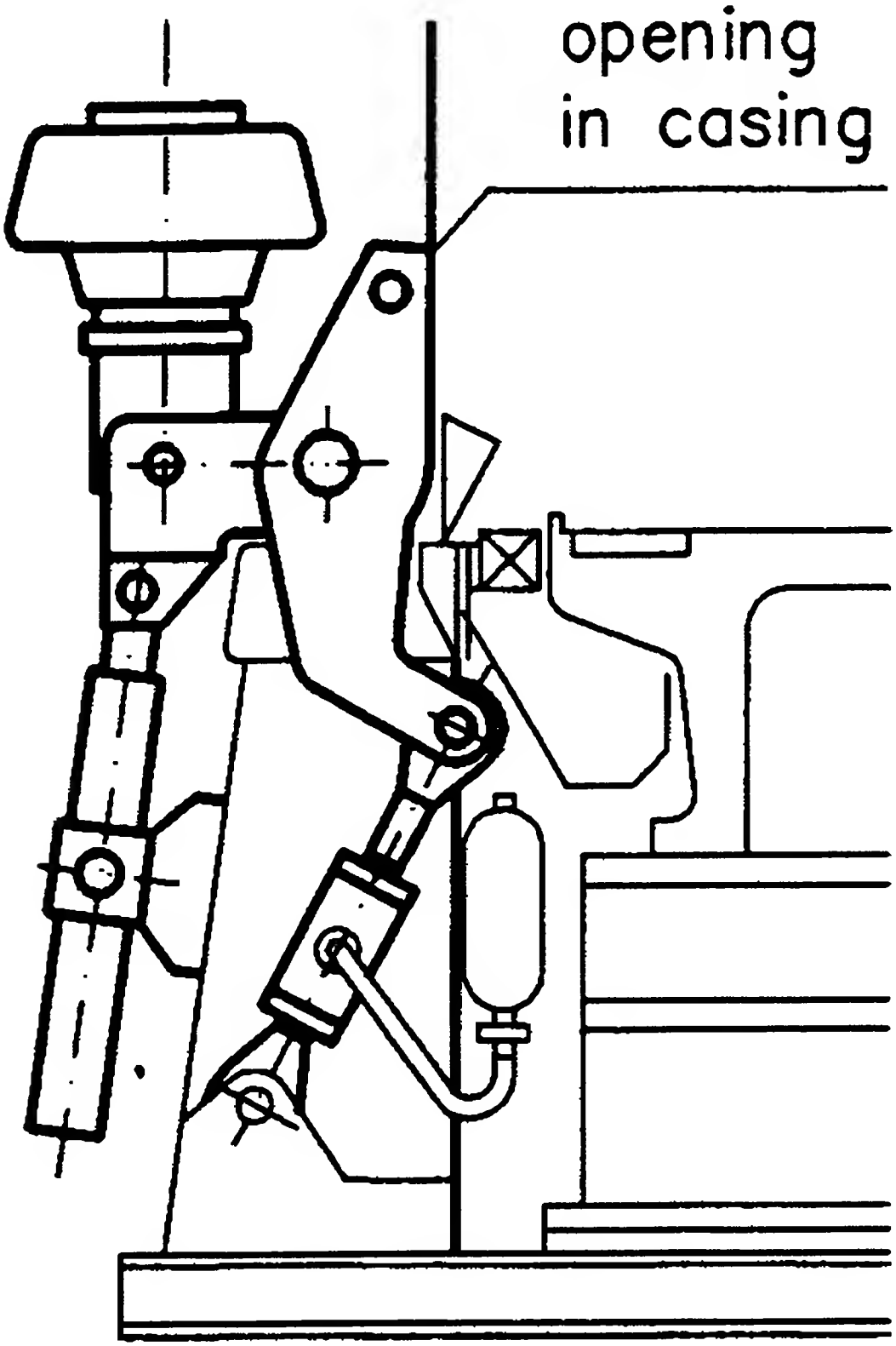

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Figure 17b Servicing Facilities - Gebr. Pfeiffer

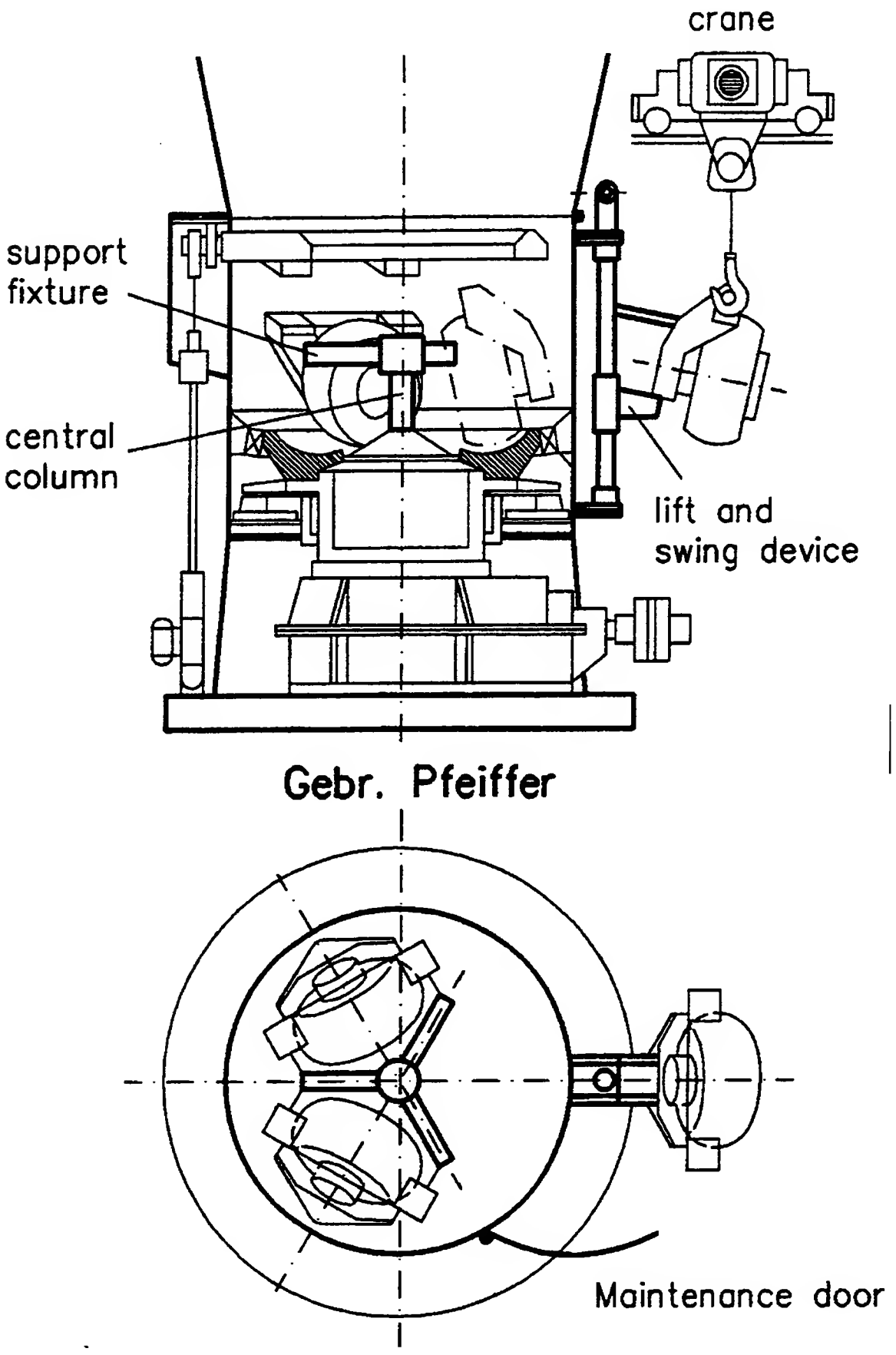

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Figure 17c Servicing Facilities - Polysius

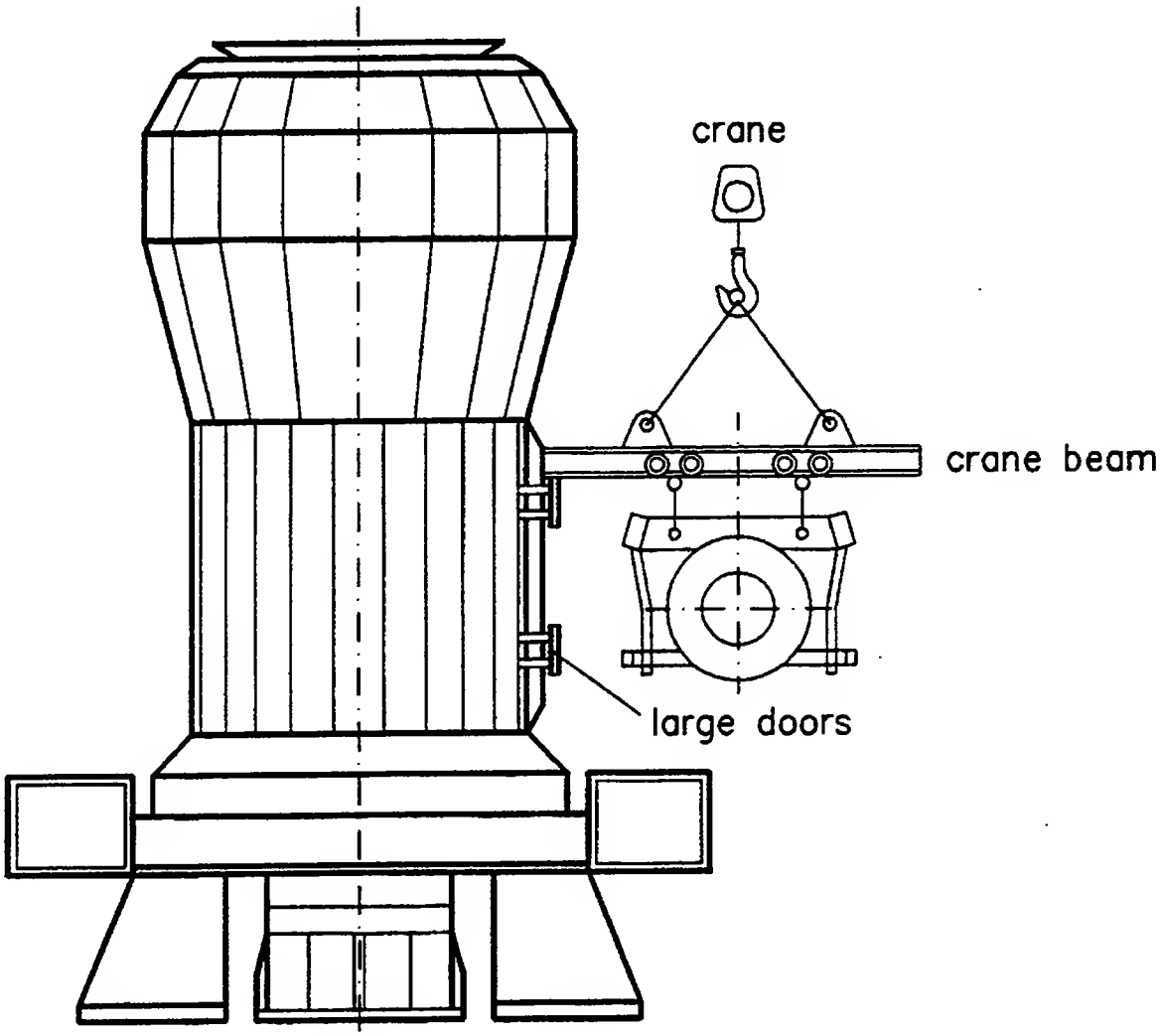

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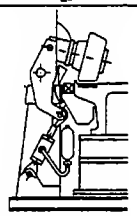
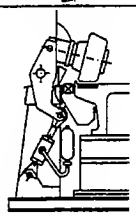
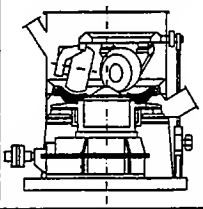
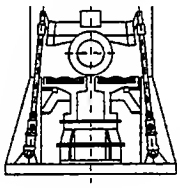
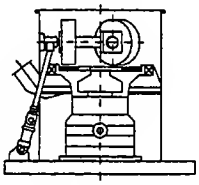
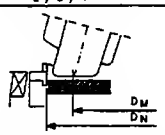
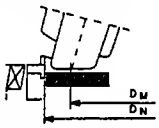
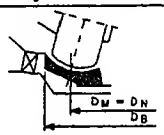
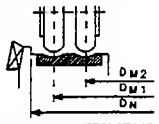
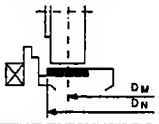
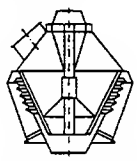
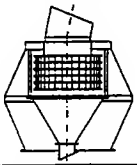
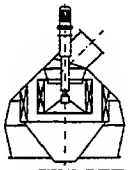
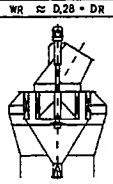
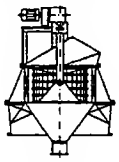
Figure 18 Suppliers - Design Roller Mills

SUPPLIER	Loesche	Fuller – Loesche	Gebr. Pfeiffer	Polysius	FLS
Make	LM	LM	MPS	RM	ATOX
Mill design					
Tension device					
– Type	Rocker arm	Rocker arm	Pressure frame	Yoke	Center yoke
– Arrangement	Outside	Outside	Pull rods / outside	Pull rods / outside	Tension rods / outside
– Roller exchange	Aux. tilting device	Aux. tilting device	Swing – lift device	Crane	Crane
Nr. of Rollers	2 / 3 / 4	2 / 3 / 4	3	2 x 2 (twins)	3
Grinding tools					
Roller inclination	15 deg	15 deg	15 deg	—	—
Grinding path	$D_M \approx 0.83 \cdot D_N$		$D_M \approx 0.80 \cdot D_B$	$D_{M1} \approx 0.75 \cdot D_N$	$D_M \approx 0.80 \cdot D_N$
				$D_{M2} \approx 0.35 \cdot D_N$	$D_R \approx 0.60 \cdot D_N$
Roller width	$W_R \approx 0.39 \cdot D_R$	$W_R \approx 0.34 \cdot D_R$ 1)		$W_R \approx 0.28 \cdot D_R$ 1)	$W_R \approx 0.20 \cdot D_R$
Separator					
Type	LAKS	HEC (O-Sepa) Fuller	SLS	SEPOL	SEPAX
		SEPAX (FLS)			RAR

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Chapter 9

Separators

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Separators

W. Zeller
VA 95/4299/E

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1. STATIC SEPARATORS

1.1 Cyclone

1.1.1 Principle of Operation

Air with dispersed material enters the cyclone through the inlet. Coarse particles precipitate at the wall due to centrifugal forces, while fine particles are carried with the air and leave the cyclone through the immersion tube.

1.1.2 Mechanical Elements

- immersion tube
- inlet volute and cylindrical section
- conical part
- discharge device (rotary valve, pendulum flap)

1.1.3 Dimensioning

Cyclone nominal diameter (rule of thumb)

$$\mathbf{d} = \sqrt{\frac{4 \cdot V}{n \cdot \pi \cdot 2.6 \cdot 3600}}$$

d	[m]	nominal diameter cyclone
V	[m^3/h]	total airflow
n	[n]	number of cyclones

1.1.4 Operation

- pressure drop around 10 - 15 mbar, dedusting efficiency 75 - 80 %
- product fineness adjustment possibilities:
  - diameter of immersion tube (smaller diameter - finer separation)
  - immersion tube depth (longer tube - finer product)
  - gas flow (higher flowrate - finer product, but higher pressure drop)

Applications

- air-swept tube mills
- roller mills
- hammer crusher-dryers, flash dryers
- cyclone-air dynamic separators (see 2.2)

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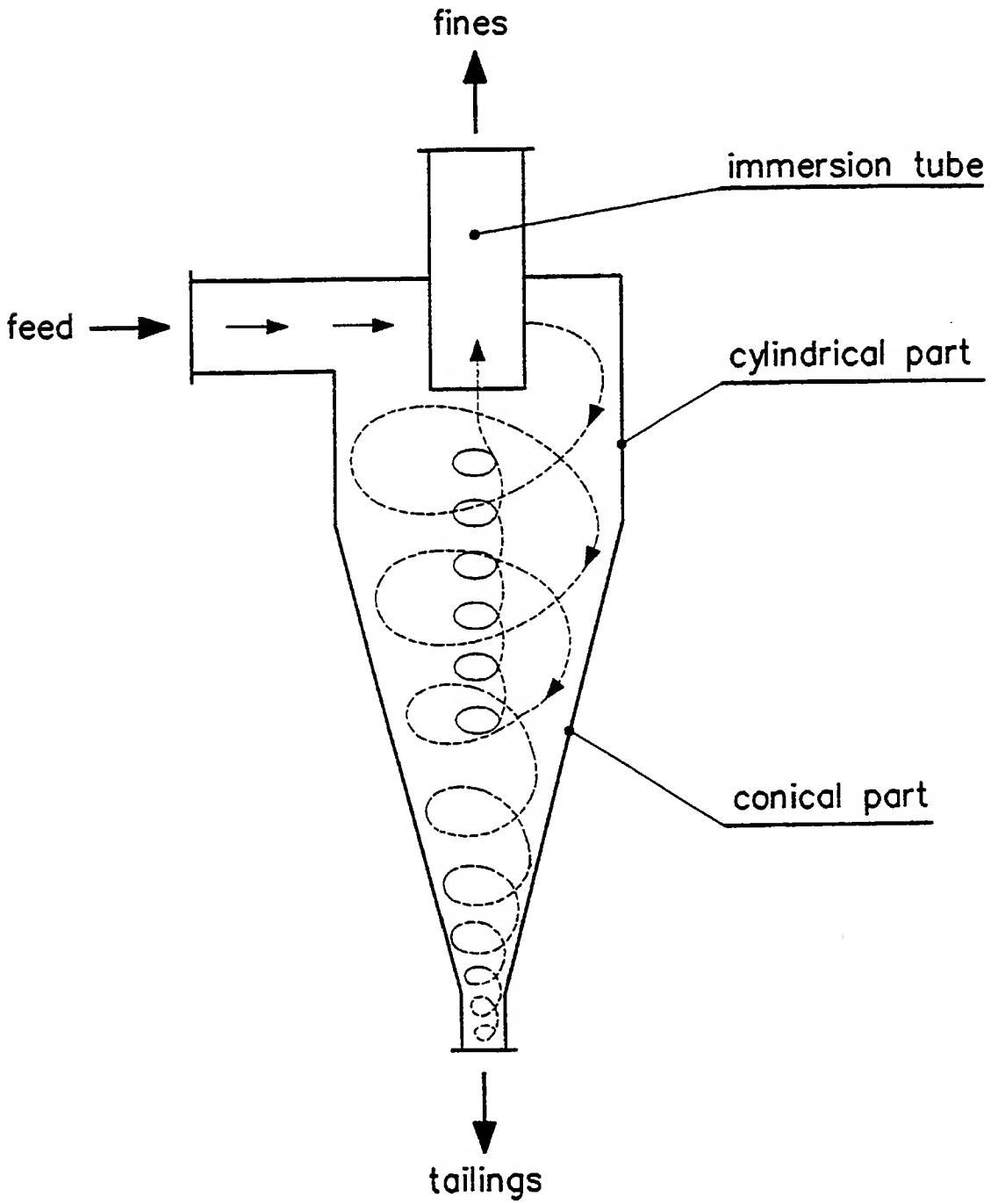
1.1.5 Design

Figure 1 Cyclone



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1.2 Grit Separator

1.2.1 Principle of Operation

Dust-laden air enters the separator from below and flows through a large number of adjustable blades. If the blades are set radially, coarse particles are precipitated only by the action of inertial forces since they cannot follow the 90 ° deflection of streamlines when entering the blades and the immersion tube.

If the guide vanes are set at an angle to the radial direction, a vortex motion of the fluid is created and separation due to centrifugal forces takes place. The effectivity of centrifugal separation is limited since the tangential velocity is not very large compared to the radial velocity.

If the angle is increased, the tangential velocity increases also and the cut size is lowered. At angles of around 60 ° the particles begin to hit the blade tips and due to friction the tangential velocity decreases.

This explains why with angles larger than about 60 ° the cut size goes up again and is generally limited to about 90 µm with this type of classifiers.

1.2.2 Mechanical Elements

- immersion tube
- adjustable blades
- blades adjustment ring
- tailings cone
- housing

1.2.3 Dimensioning

Specific air load (feed) 500 - 1000 g/m^3

$$\mathbf{d = 0.0127 \cdot \sqrt{V}}$$

d	[m]	nominal diameter
V	[m^3/h]	airflow

1.2.4 Operatin

- Pressure drop 10 - 15 mbar, dedusting efficiency 75 - 80 %
- coarser product: shorten immersion tube, set blades radially
- finer product: lengthen immersion tube, set blades tangentially

Applications

- air-swept ball mills
- vertical roller mills
- hammer crushers

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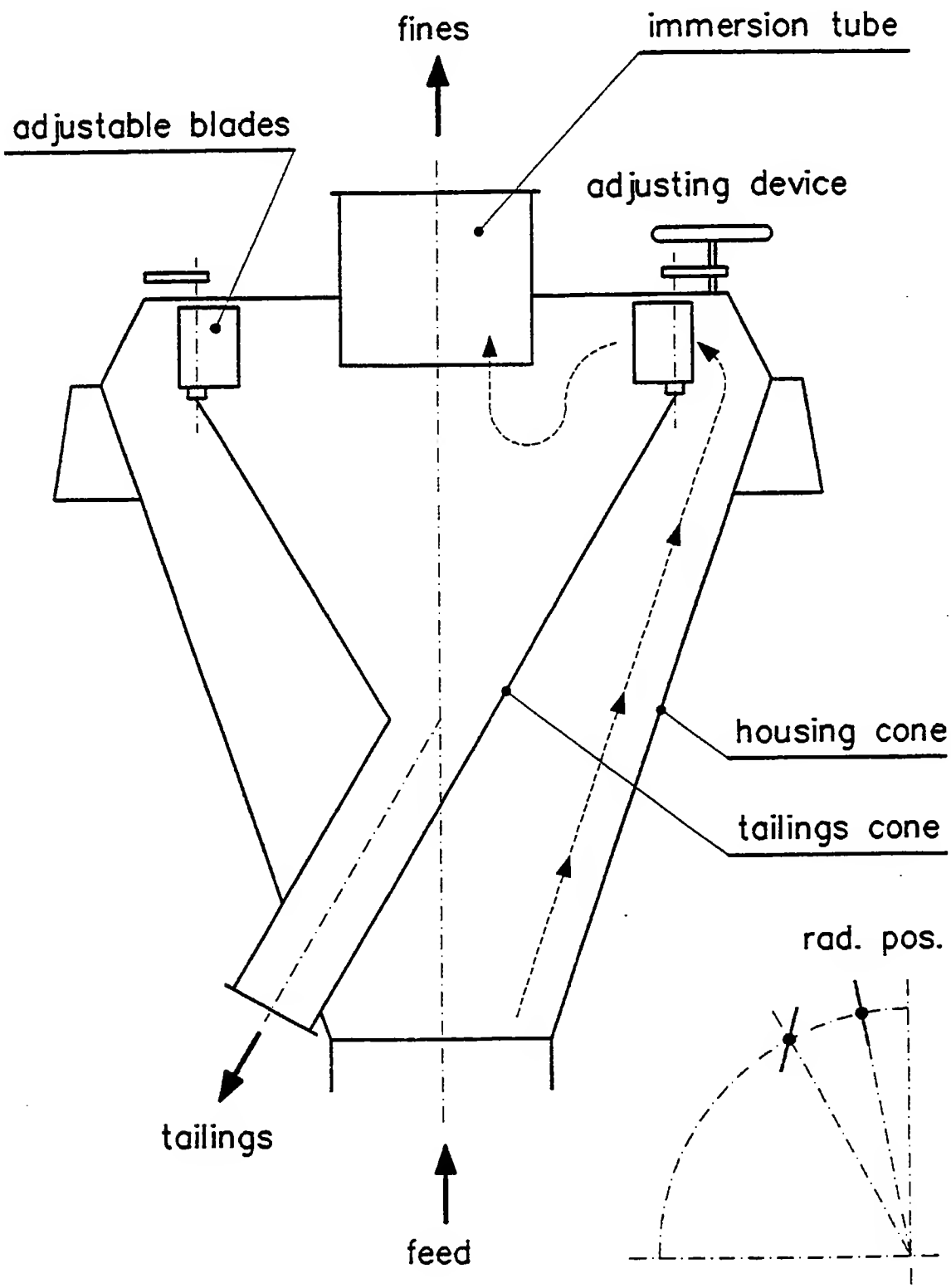
1.2.5 Design

Figure 2 Grit Separator



1.3 VS-Separator

1.3.1 Principle of Operation

The feed is admitted to the VS-Separator from above. The material flows through the classifying zone over inclined plates. Classifying air is introduced into the classifying zone transversely to this fresh material flow. The classifying air can consist of either circulating air, hot gases for drying or fresh air for cooling. The actual separation zone is located between the inclined plates and an array of baffle plates. The fines entrained by the air flow are moved through the space in between the baffle plates and discharged at the air exit opening.

Due to gravitational force the coarse fraction is discharged from the classifier in downward direction.

1.3.2 Mechanical Elements

- housing
- inclined plates four guiding the feed material with individually replaceable plates
- baffle plates for flow direction with individually replaceable elements
- air inlet/outlet box
- tailings outlet

1.3.3 Dimensioning

Guide values for 1700 cm^2/g product fineness in closed circuit with roller press:

- specific air load (feed) ~ 4 kg/m^3
- specific separator load ~ 6 m^3/s per m^2 projected open separating area
- designation: VS-459 → 4.59 m^2 projected open separating area

1.3.4 Operation

- Setting of a suitable air velocity enables control of cut size
- product fineness up to 4'500 cm^2/g is possible in finish grinding systems
- pressure drop 6 to 15 mbar

Applications

- high pressure comminution circuit with a roller press for clinker and raw material

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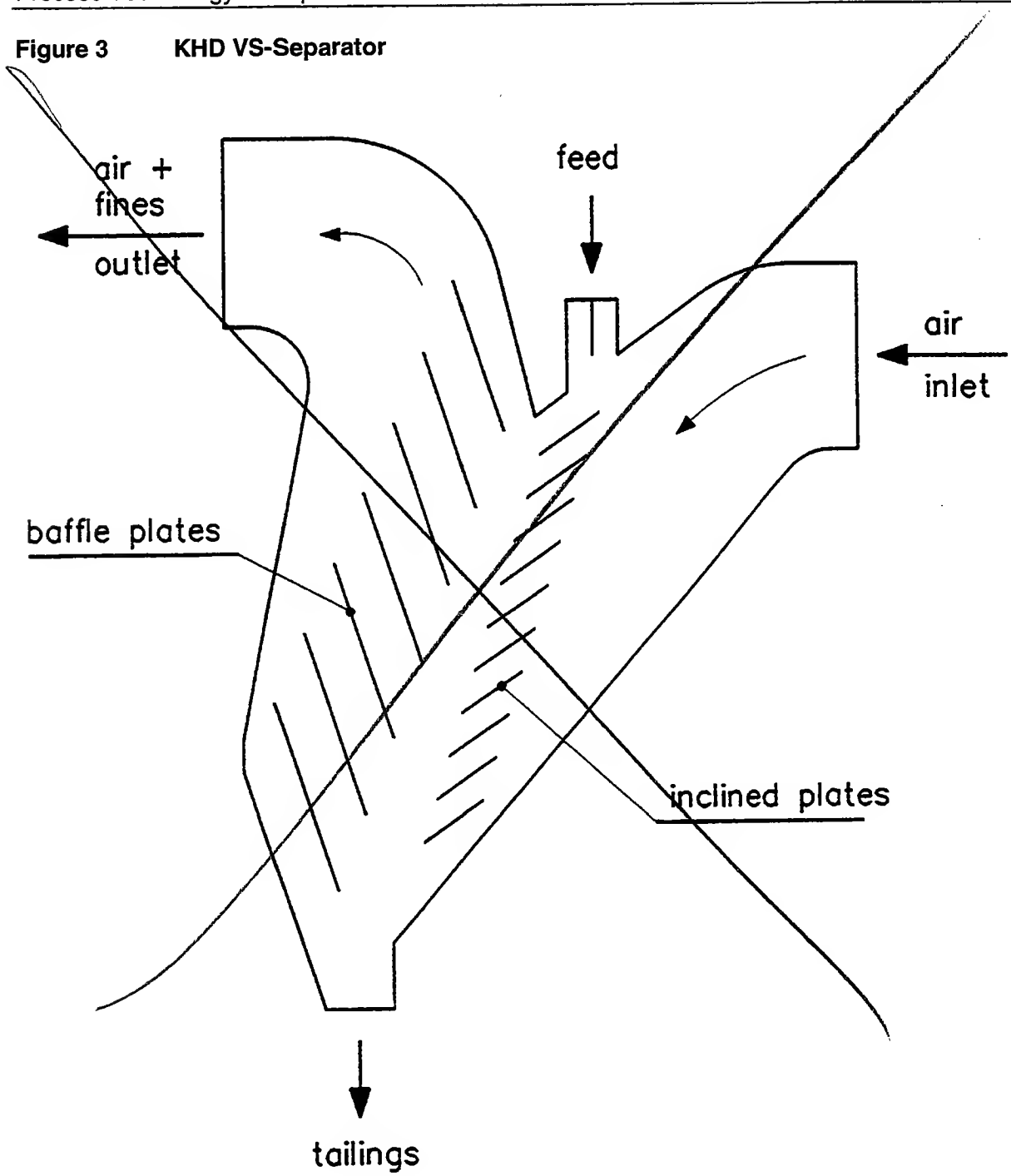
Figure 3 **KHD VS-Separator**



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2. DYNAMIC SEPARATORS

2.1 Classifiers with Counterblades and Internal Fan

2.1.1 Principle of Operation

Feed material is introduced through a chute onto a distributor plate that disperses the particles in the airflow. Air with dispersed particles flows up and passes the rotating counterblades. Coarse particles are centrifuged out to the surrounding wall and fall down into the tailings cone. Air with fines flows through the fan to the fines chamber. Here the fine material is separated from the conveying air and collected in the outer cone. Air together with some amount of fine material is returned to the separating zone through the air vanes.

2.1.2 Mechanical Elements

- housing
- feed spouts
- distributor plate
- counterblades
- internal fan
- return air vanes
- tailings cone
- fines collecting chamber
- fineness control valves (Sturtevant)
- gearbox and motor

2.1.3 Dimensioning

Cement Fineness [cm^2/g]	**Specific Separator Load** [$t/h.m^2$]
~ 2500	2.2 - 3.6
~ 4500	1.0 - 1.5

$$\text{specific separator load} = \frac{finished\,product[t/h]}{nominal\,cross\,area[m^2]}$$

- the nominal cross section area refers to the outer diameter of the separator
- for large separators the higher values are applicable
- the indicated values are guide values and vary from supplier to supplier

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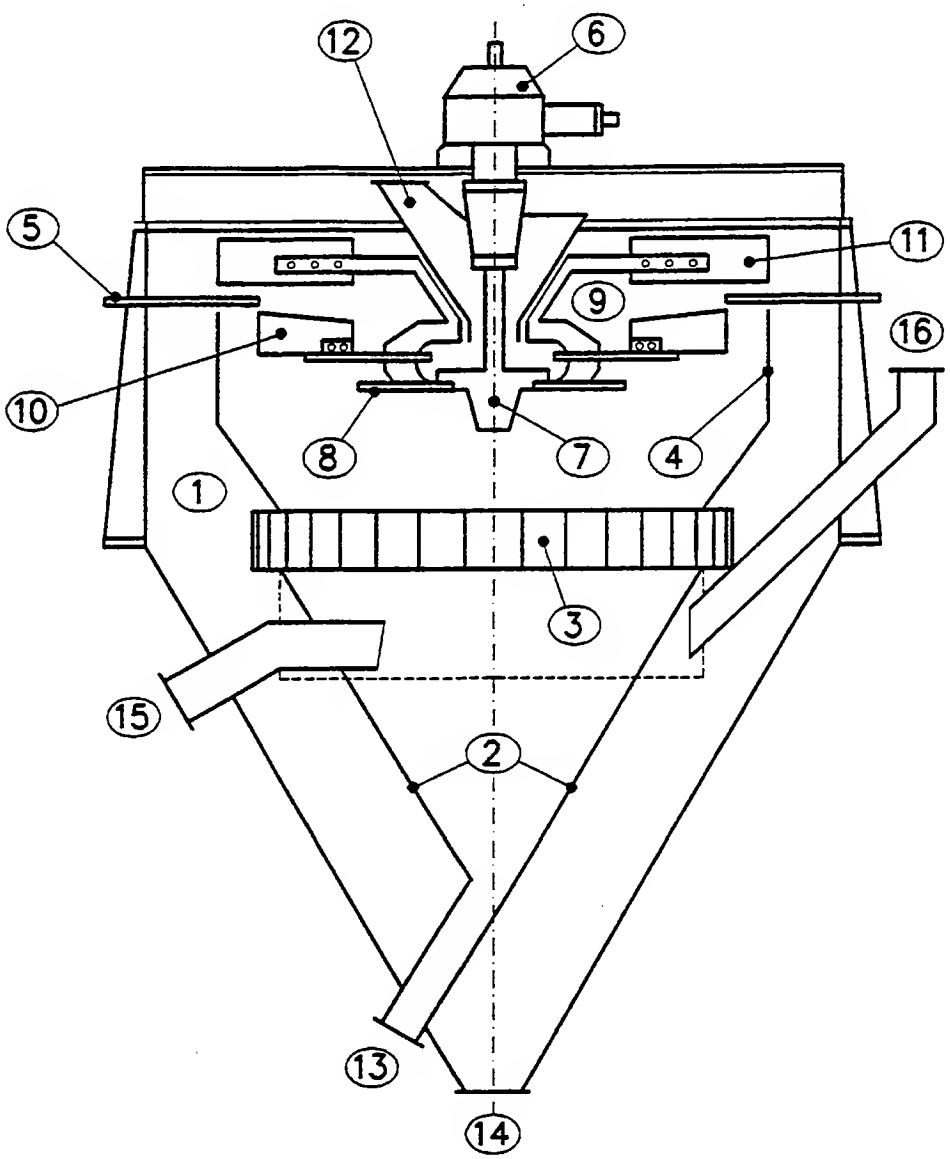
Figure 4 Sturtevant Whirlwind Air Separator



(1) fines chamber	(9) fan cone
(2) tailings cone	(10) counterblades
(3) air vane	(11) main fan blades
(4) inside drum	(12) feed spout and intake cone
(5) fineness control valve rod	(13) tailings outlet
(6) gear reducer	(14) fines outlet
(7) main shaft and distributing hub	(15) air inlet
(8) distributing plate	(16) air outlet

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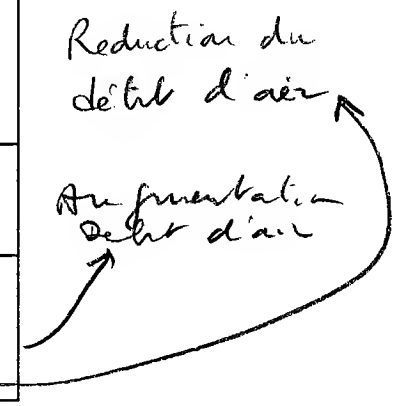
2.1.4 Operation

Fineness range 3000 - 6000 cm^2/g

Adjustment possibilities:

- speed of counterblades
- radial position, number and size of counterblades
- radial position of fan blades (Sturtevant, Raymond)
- radial position of fineness control valves (Sturtevant, Raymond)

adjustment	**product fineness**
counterblades:	
increased speed	finer
increased number	finer
increased radius (moved outward)	finer
increased size	finer
fan blades:	
increased radius (moved outward)	coarser
internal fan damper:	
outward position (larger opening)	coarser
inward position (smaller opening)	finer



2.1.5 Design

- Sturtevant Separator (fig. 4)
  - the separator shown in fig. 4 has additional air inlets/outlets for drying/cooling purposes
  - similar makes: Raymond
- Heyd Separator (fig. 5)
  - the Heyd separator from Chr. Pfeiffer has two individual drives for fan and counterblades
  - (fixed speed for fan, variable speed for counterblades and distributor plate), this makes adjustment easier and improves the separator efficiency by fine grinding.
  - similar makes: Polysius (Turbopol), Fuller, Schmidt

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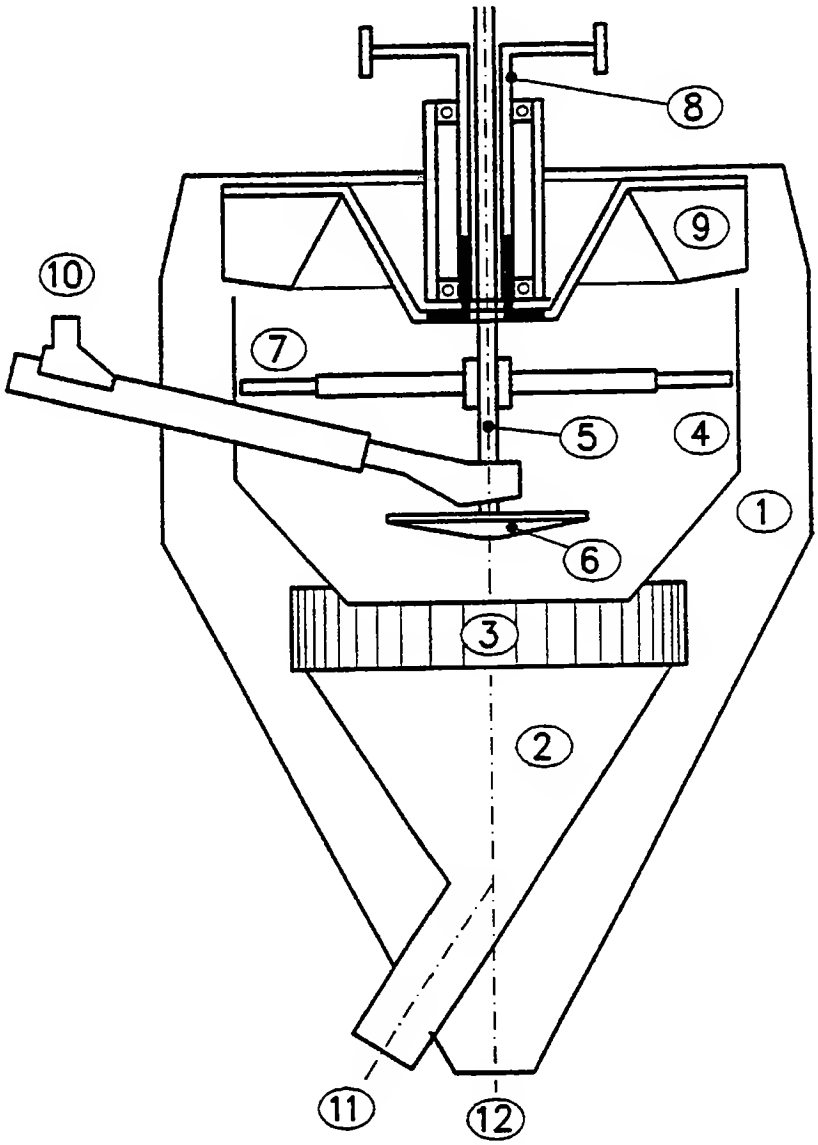
Figure 5 Heyd-Type Separator



1. fines chamber
2. tailings cone
3. air vane
4. separation chamber
5. distributor+counterblades
6. distributor plate
7. counterblades
8. fan shaft
9. fan blades
10. feed spout
11. tailings outlet
12. fines outlet

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2.2 Classifiers with Counterblades and External Fan

2.2.1 Principle of Operation

Material is introduced through feed spouts and dispersed in the circulating air by the rotating distributor plate. Particles fine enough to pass the rotating counterblades are conveyed by air to external cyclones where they are precipitated and discharged. Coarse particles move downward to the coarses cone. The air is recirculated to the separator via fan air vanes.

Inleaking false air is removed via dedusting filter and fan in order to maintain a constant underpressure inside the system.

2.2.2 Mechanical Elements

* housing
* feed spouts
* tailing cone
* distributor plate
* counterblades
* drive unit
* cyclones
* external fan
* additional ducts

2.2.3 Dimensioning

Cement Fineness [cm^2/g]	**Specific Separator Load** [$t/h.m^2$]
2600	~ 11
3000	~ 8
4000	~ 5
4500	~ 4

$$\text{specific separator load} = \frac{finished\,product[t/h]}{nominal\,cross\,section\,area[m^2]}$$

The indicated values are guide values and vary from supplier to supplier.

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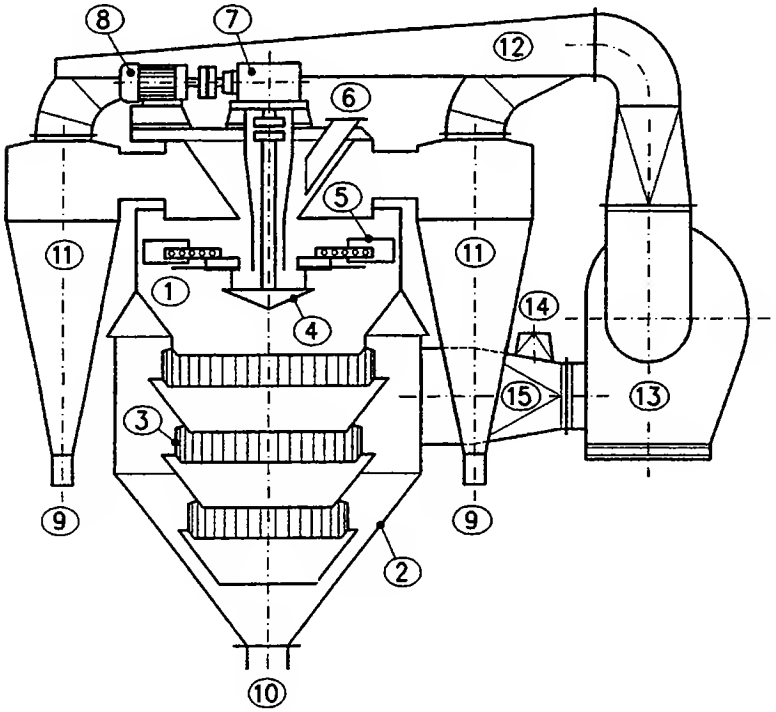
Figure 6 Cyclone Air Separator (WEDAG)



① separation chamber
② tailings cone
③ air vanes
④ distributor plate
⑤ counterblades
⑥ feed spout
⑦ gearbox
⑧ motor
⑨ fines outlet
⑩ tailings outlet
⑪ cyclones
⑫ air duct to fan
⑬ fan
⑭ dust collecting pipe to filter
⑮ return air duct

2.2.4 Operation

Fineness adjustment possibilities:

- speed of counterblades
- number of counterblades
- airflow rate

adjustment	product fineness
counterblades:	
increased speed	finer
increased number	finer
fan damper:	
more open	coarser
more closed	finer
fan speed:	
higher speed	coarser
lower speed	finer

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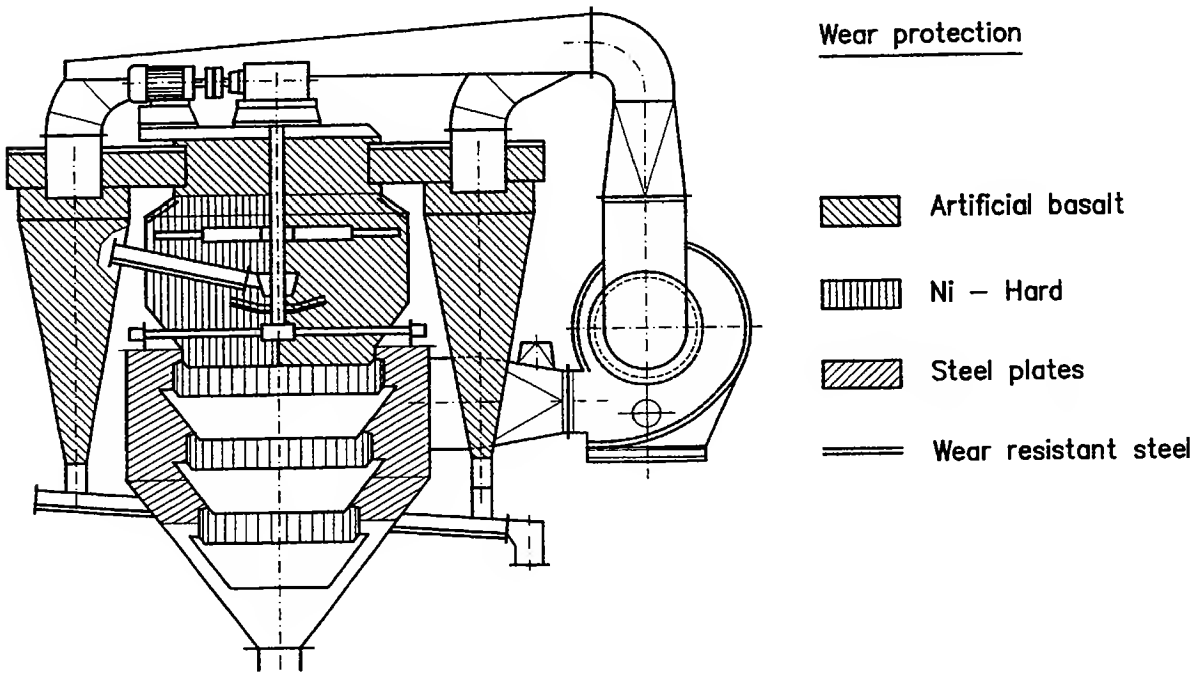
2.2.5 Design

Fig. 6 shows a Wedag ZUB separator with external air circuit and usually a number of planetary cyclones. In figure 7 a typical example for the wear-protection of separator parts is given for a Polysius Cyclopol separator.

The advantages of this type of classifier over the internal air circulation types can be summarized as follows:

- independent adjustment of separator speed and airflow gives a wider range of regulation and better fines separation efficiency because of more circulating air and lower particle concentrations
- a number of small cyclones gives a more efficient fines separation from the circulating air. This gives less fines in the return material and therefore a higher efficiency .
- fan runs in less dusty environment
- adjustment of fineness can be made without stopping the separator for change of counterblades

Figure 7 Cyclone Air Separator (Polysius Cyclopol)



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2.3 Classifiers with Rotor Cage and External Fan (High-efficiency separators)

2.3.1 Principle of Operation

The central part is a rotating circular cage comprising a top distributor plate and a ring of vertical blades or rods that constitute the rejector.

Material is introduced through feed spouts and drops on the rotating dispersion plate on top of the rotor cage. From there it descends in the form of a circular curtain in front off the rotor blades. Air is introduced tangentially through guide vanes into the separating zone. Fine particles leave the separator with the air through the rotor and are precipitated in external cyclones or bag filter. Coarse particles are rejected by the rotor blades, pass the separation zone and leave the separator through the discharge device of the tailings cone.

2.3.2 Mechanical Elements

* feed chutes
* cage with rotor blades, shaft and distributor plate
* air inlet volute
* air guide vanes or conical louvre rings
* tailings outlet
* fines outlet
* gearbox and motor

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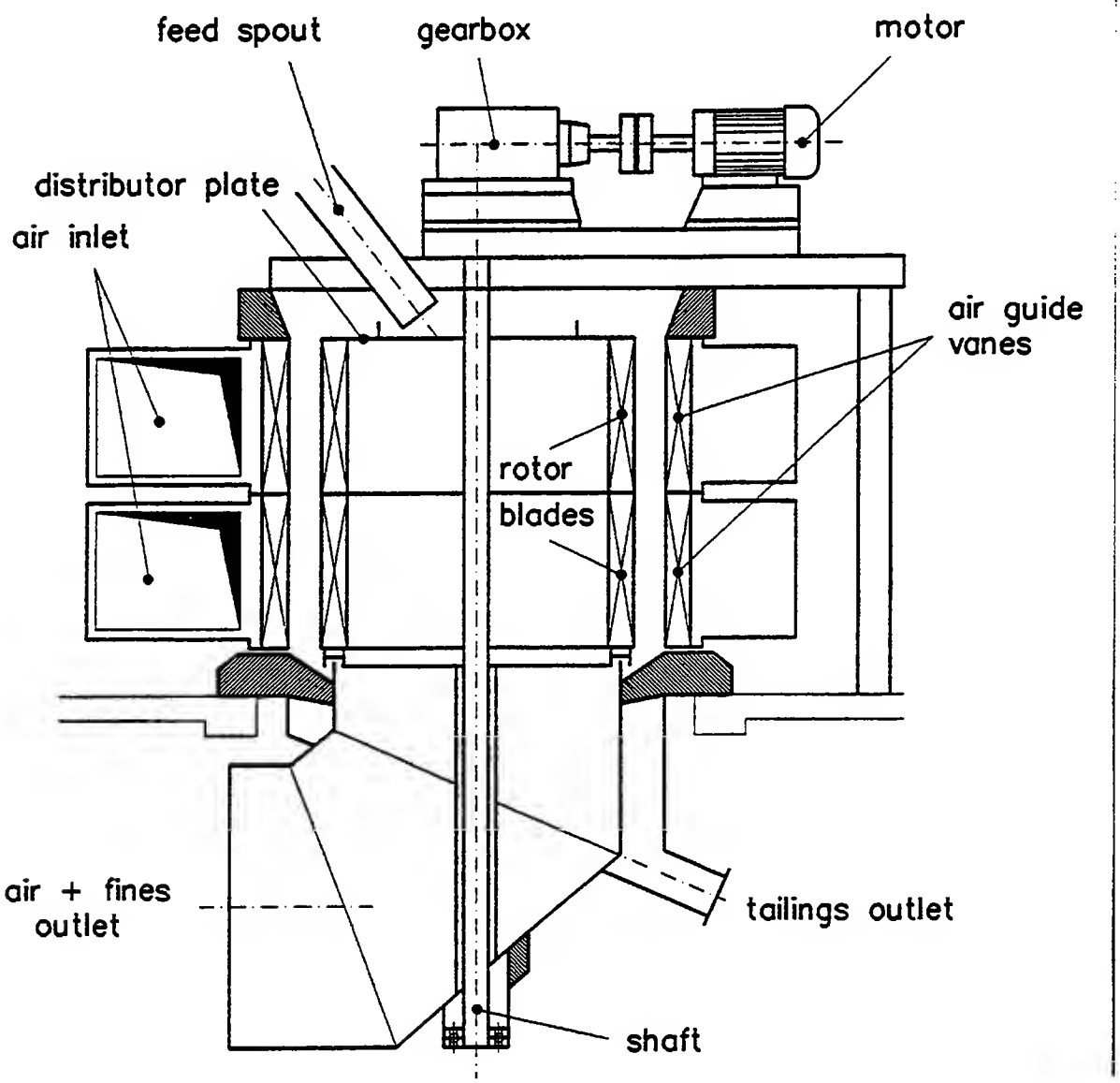
Figure 8 O & K Cross-Flow Separator



2.3.3 Dimensioning

Dimensioning criteria for cement with around 3000 Blaine (see also fig. 9)

- specific rotor load ~ **10 - 12 t/h.m^2** or less
- specific air load (feed) < **2.5 kg/m^3**
- specific air load (product) < **0.75 kg/m^3**
- peripheral speed **5 to 35 m/s**, depending on Blaine
- specific installed power **0.4 - 0.5 kWh/t**

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2.3.4 Operation

The separator is operated with constant airflow, fineness is controlled by the speed of the separator cage. Only in extraordinary cases where motor power or gearbox reduction ratio pose limits to separator speed, the airflow may be decreased to obtain the desired fineness.

To avoid dust emissions, it is mandatory to maintain a certain underpressure (around - 5 mbar) in the separator system. Since no ducting is perfectly tight, this underpressure causes an air inleak that has to be removed and dedusted by a filter. In case of single pass design this is the main dedusting filter. With cyclone air separators an additional dedusting filter has to be provided.

The recommended design value for the dedusting system air flow rate is 10 % of separating air amount.

Characteristic figures:

◆ installed separator	about 0.4 - 0.5	kWh/t
◆ installed fan power	about 2 - 2.5	kWh/t
◆ pressure drop across separator	20 - 25	mbar
◆ pressure drop across cyclones	about 10 - 15	mbar
◆ pressure drop across bagfilter	about 10 - 15	mbar
◆ pressure drop ducting	about 5	mbar

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Figure 9 Rotor Type Separator

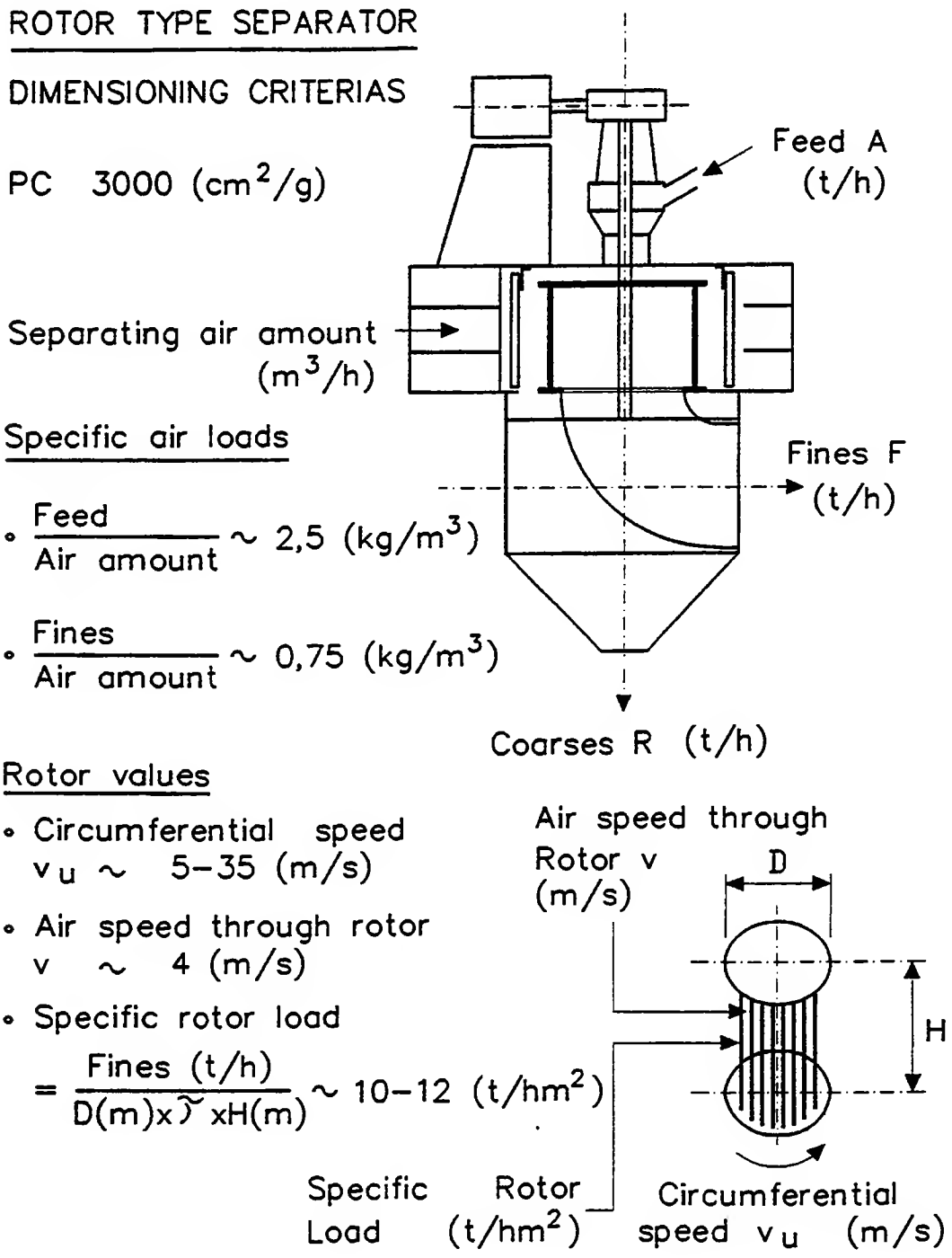

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Figure 10 Fuller (Onoda) O-Sepa

Characteristics

Supplier **Type**	**Fuller (Onoda)** **O-Sepa**
Rotor: Diameter/Height	~ 1.7 : 1
Number of sep. air inlets primary air secondary air tertiary air	3 mill exhaust, ambient air ambient air air into cone
Number of sep. air outlets	1
Location air outlet	top of separator
Feed	central from top (2 or 4 inlets)
Fineness adjustment	rotor speed separating air amount
Standard sizes	12
Designation	N-500 ↓ Separating air amount m^3/min.

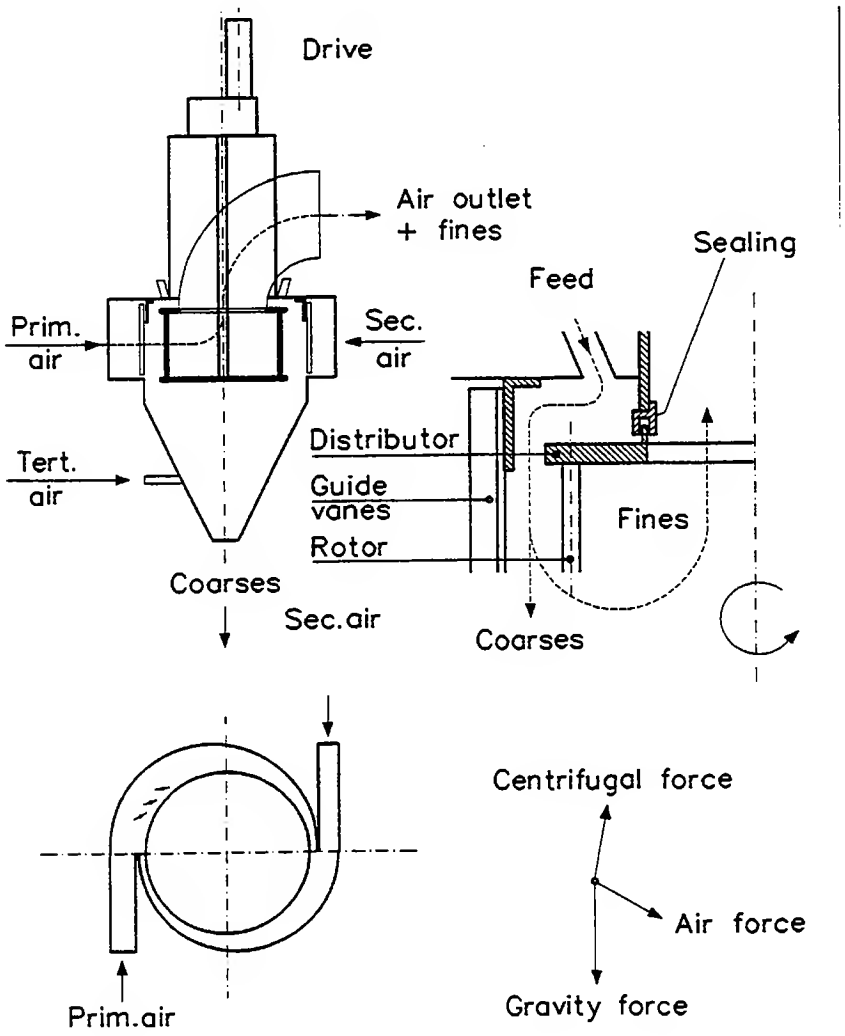

Figure 11 Polysius Sepol (Rotor Type Separator, Makes)

Characteristics

Supplier **Type**	**Krupp-Polysius** **Sepol**
Rotor: Diameter/Height	1.5 : 1
Number of sep. air inlets	1
Number of sep. air outlets single pass with cyclones	depending on system 1 1 to 6
Location air outlet	bottom part
Feed	central from top
Fineness adjustment	rotor speed separating air amount
Standard sizes	21
Designation	150 / 4 → number of cyclones ↓ rotor diameter [cm]

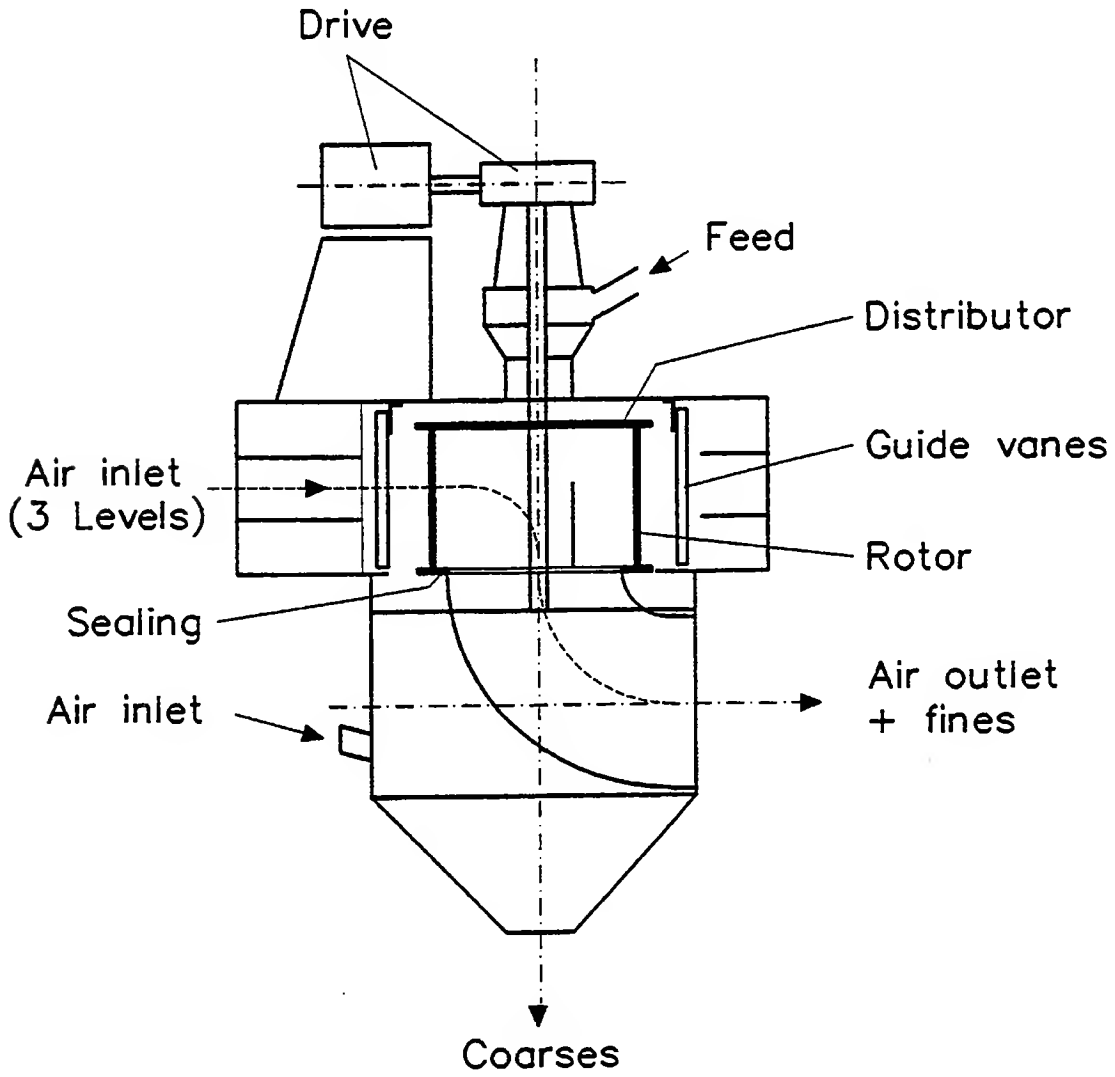

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Figure 12 FLS Sepax

Characteristics

Supplier **Type**	**F.L. Smidth** **Sepax**
Rotor: Diameter/Height	~ 2.4 : 1 small separators ~ 1.2 : 1 large separators
Number of sep. air inlets	1
Number of sep. air outlets single pass with cyclones	depending on system 1 4
Location air outlet	top of separator
Feed	from bottom (dispersed in air/gas)
Fineness adjustment	rotor speed separating air amount
Standard sizes	17
Designation	Sepax 2 - 450 | | → diameter casing [cm] | → 2 with cyclones | → 1 single pass

The separator shown on the right of fig. 12 is equipped with integrated desagglomerator for application in roller press circuits.

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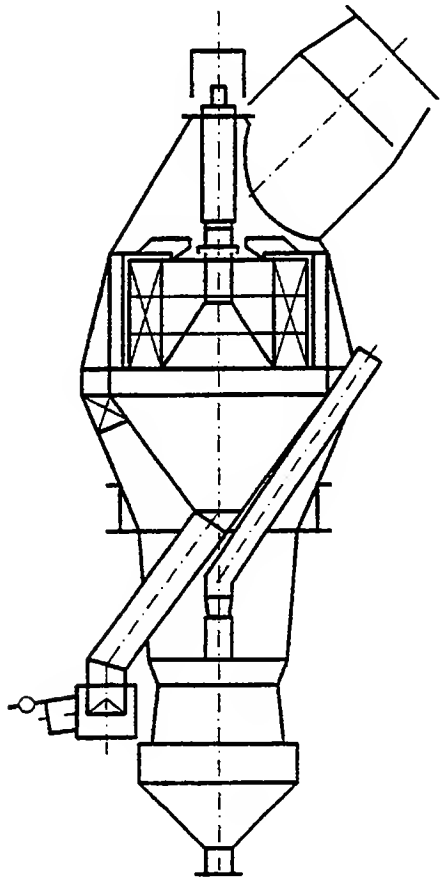
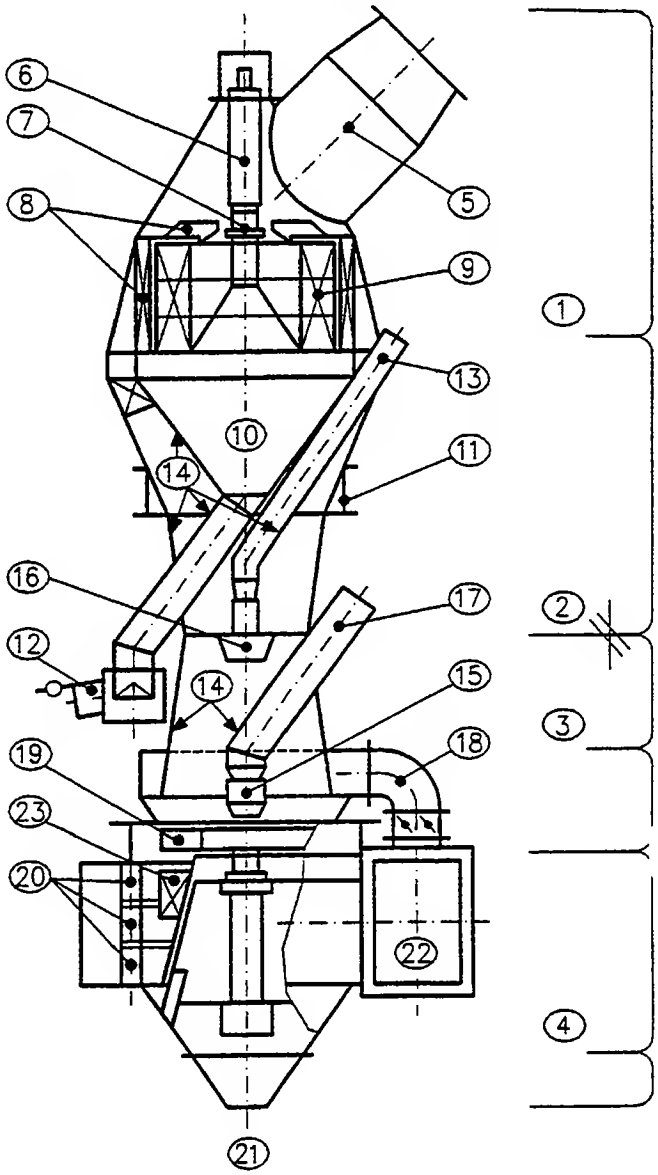
compact version



integrated desagglomerator
(roller press applications)



① separator part
② optional duct ext. to fit layout
③ desagglomerator
④ grit separator
⑤ fines outlet
⑥ bearing housing
⑦ shaft rotor joint
⑧ guide vane
⑨ rotor blade
⑩ reject cone
⑪ support
⑫ reject outlet valve
⑬ feed inlet
⑭ densit wearcast 2000
⑮ air lock
⑯ spreader plate
⑰ feed from press
⑱ air by-pass
⑲ desagglomerator rotor
⑳ guide vane sections
㉑ outlet to press
㉒ air inlet
㉓ rotor blades

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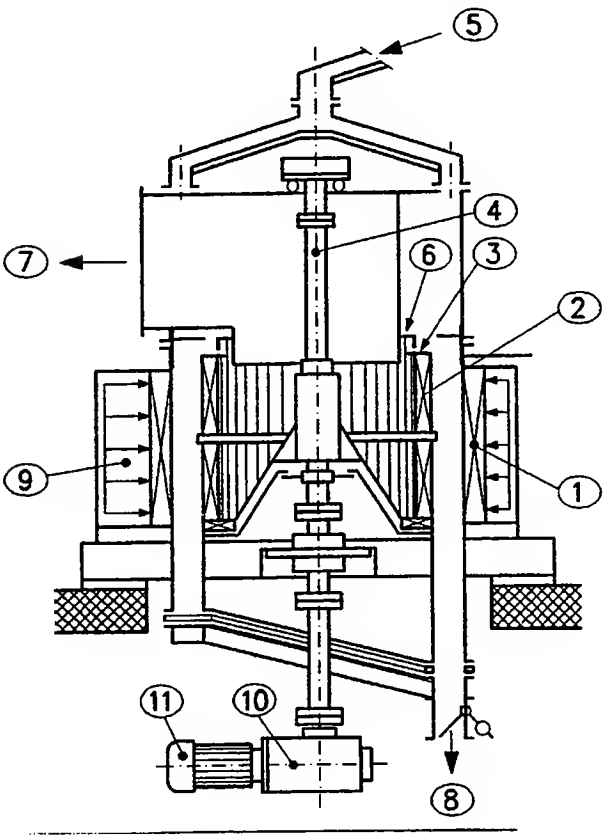
Figure 13 KHD Sepmaster SKS - Z

Characteristics

Supplier **Type**	**KHD** **Sepmaster**
Rotor: Diameter/Height	~ 1.65 : 1
Number of sep. air inlets	1
Number of sep. air outlets single pass with cyclones	depending on system 1 2 to 8
Location air outlet	top of separator
Feed	central from top (SKS-Z, SKS-D) from bottom (SKS-LS)
Fineness adjustment	rotor speed separating air amount
Standard sizes	14
Designation	SKS 175 / 4 | → number of cyclones



① guide vanes
② rotor blades
③ distributor plate
④ rotor shaft
⑤ feed spout
⑥ sealing
⑦ air + fines outlet
⑧ tailings outlet
⑨ air inlet
⑩ gear box
⑪ motor

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Figure 14 KHD Sepmaster SKS-D and Sepmaster SKS-LS

KHD Sepmaster SKS-D

The design is specially tailored for closed-circuit operation with roller presses.

The sturdy desagglomerator is mounted directly above the separator and equipped with its own drive. Thus, both separator and desagglomerator can be individually adjusted for optimal performance.

KHD Sepmaster SKS-LS

The separator is laid out for material feeding together with air/gas.

It is applied mainly for combined drying and grinding of materials with high moisture content.

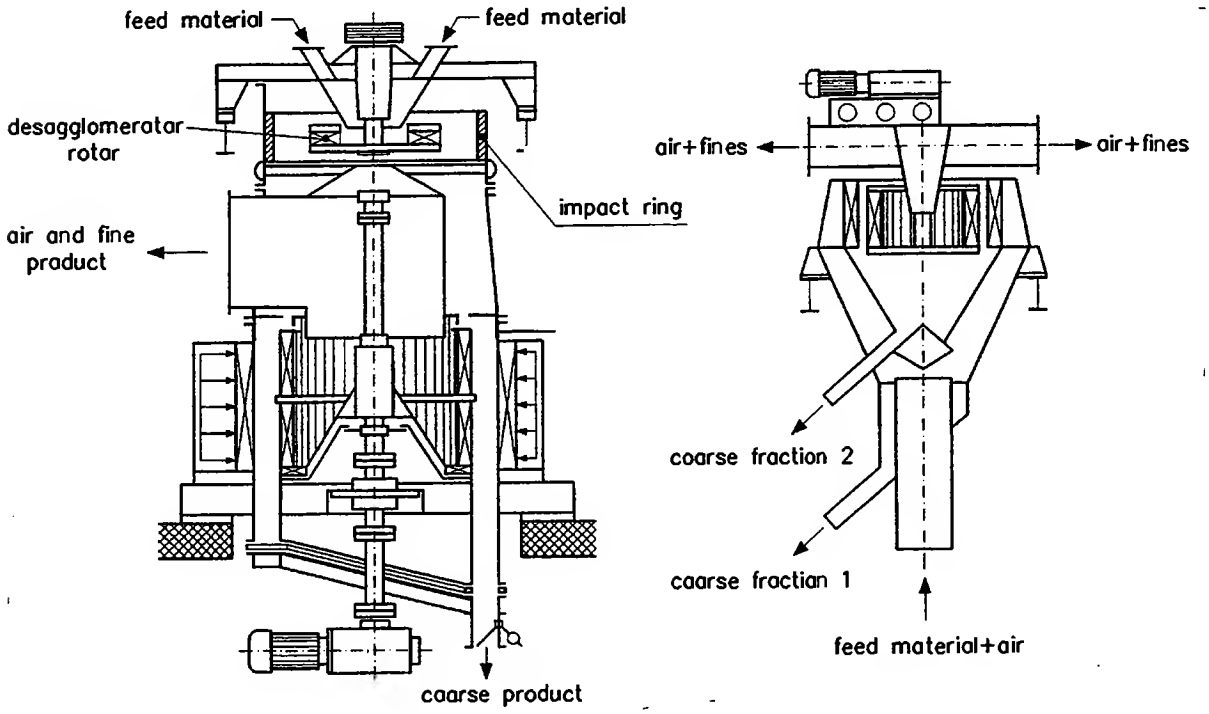

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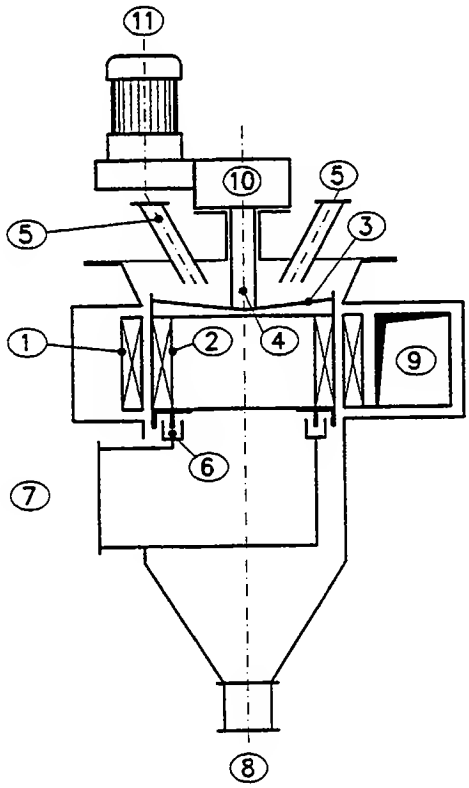
Figure 15 Pfeiffer QDK

Characteristics

Supplier **Type**	**Pfeiffer** **QDK**
Rotor: Diameter/Height	~ 1.5 : 1
Number of sep. air inlets	1
Number of sep. air outlets single pass with cyclones	depending on system 1 1 (cyclones installed separately) 1 to 8 (cyclones installed around sep.)
Location air outlet	bottom part
Feed	central from top
Fineness adjustment	rotor speed separating air amount
Standard sizes	17
Designation	11-N / 2 cyclones |→ diameter of separating zone [dm]



① guide vanes
② rotor blades
③ distributor plate
④ rotor shaft
⑤ feed spouts
⑥ sealing
⑦ air + fines outlet
⑧ tailings outlet
⑨ air inlet
⑩ gear box
⑪ motor

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Figure 16 FCB TSV Classifier

Characteristics

Supplier **Type**	**FCB** **TSV**
Rotor: Diameter/Height	~ 4 : 1 small separators ~ 2.5 : 1 large separators
Number of sep. air inlets	1
Number of sep. air outlets	1
Location air outlet	top of separator
Feed	from top from bottom (feed dispersed in air/gas) mixed (top/bottom feed)
Fineness adjustment	rotor speed separating air amount
Standard sizes	16
Designation	TSV - 4000 |→ rotor diameter [mm]

The type with top feed is applied in e.g. bucket elevator ball mill circuits.
The type with mixed feed (top + bottom) is applied in e.g. grinding circuits with air-swept and semi-airswept tube mills and with the HOROMILL.

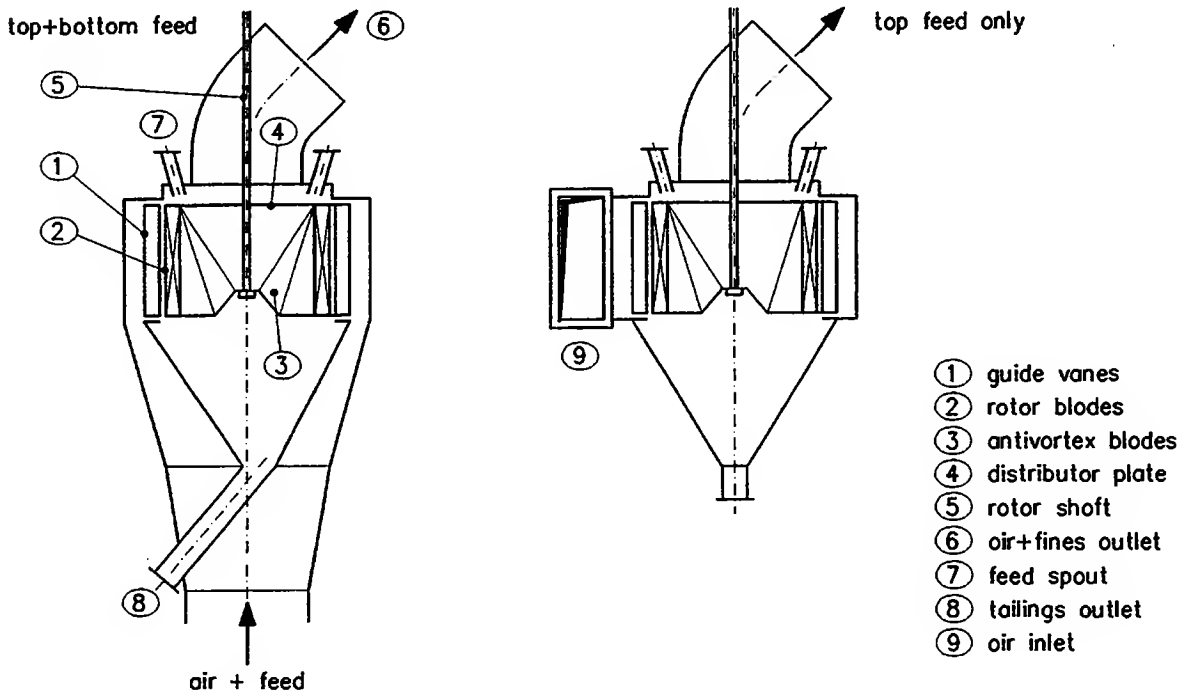

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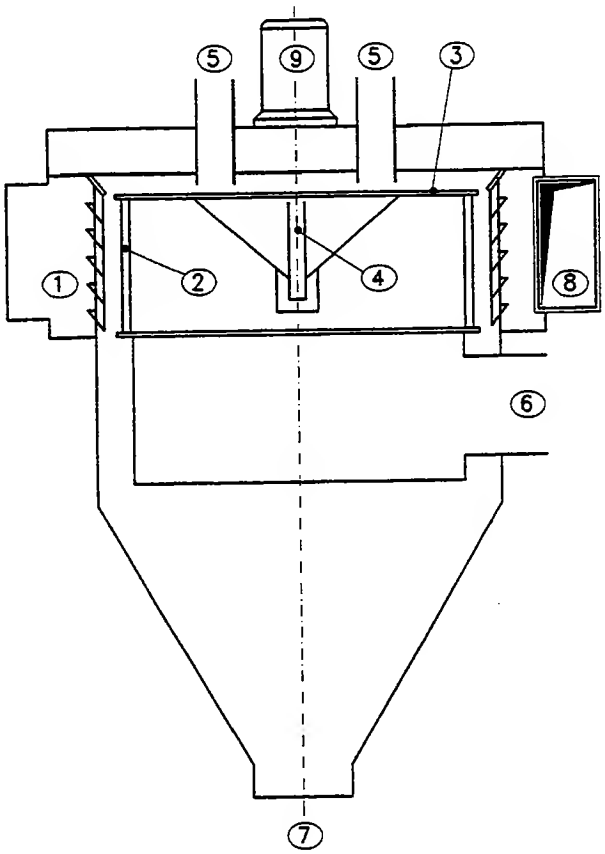
Figure 17 Sturtevant SD (Side Draft) Classifier

Characteristics

Supplier **Type**	**Sturtevant** **SD (Side Draft)**
Rotor: Diameter/Height	~ 1.3 : 1 small separators ~ 2.4 : 1 large separators
Number of sep. air inlets	1 to 8 1 with cyclones 8 with only fresh air
Number of sep. air outlets	depending on system
Location air outlet	bottom part
Feed	central from top
Fineness adjustment	rotor speed separating air amount
Standard sizes	10
Designation	SD - 20 - 4 | | → number of cyclones | → casing diameter [ft]



① conical deflector plates
② cage rods
③ distributor plate
④ rotor shaft
⑤ feed spouts
⑥ air + fines outlet
⑦ tailings outlet
⑧ air inlet
⑨ rotor drive

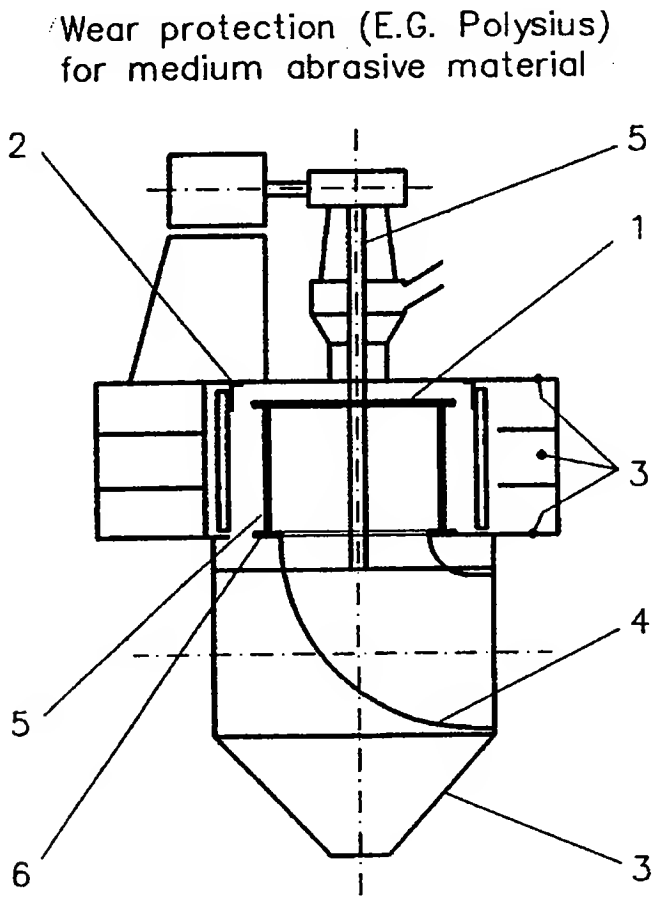
2.4 Wear Protection

The following parts of a rotor type separator are exposed to higher wear and require a protecting material or are manufactured of a wear-resistant material:

* air inlets
* air outlets
* distributor plate
* rotor
* drive shaft

In fig. 18 typical examples for wear-protected parts are given for the Sepol separator from Polysius and the O-SEPA separator from Fuller.

Figure 18a Separator Wear Protection (e.g. Polysius) for Medium Abrasive Material



1 Compound steel plate (Duromax 260)
2 Vaudit, compound wear plate (mild steel + hard facing)
3 Tiscral
4 Basalt
5 St 37-2
6 Castolin, (hard facing)

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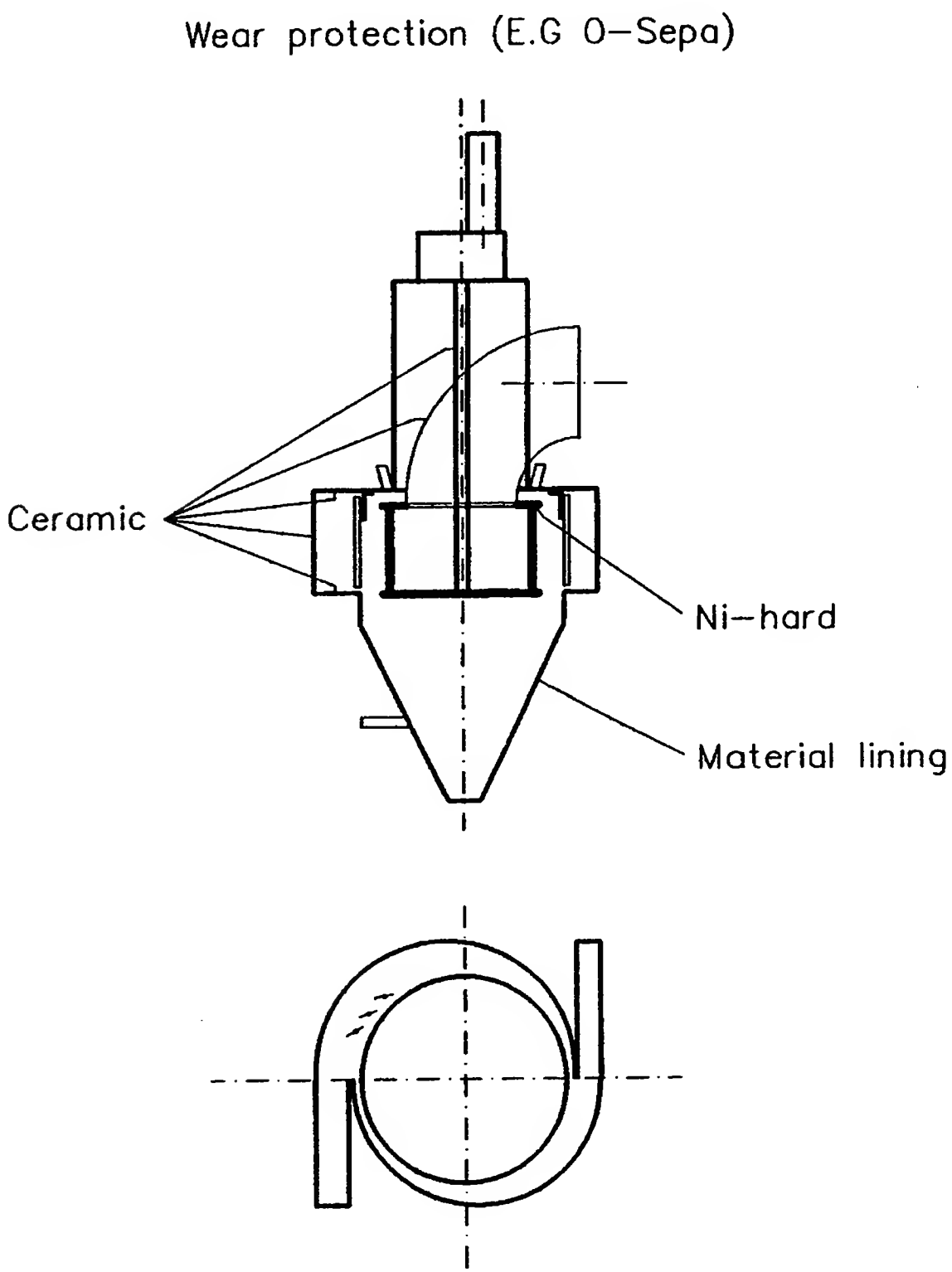
Figure 18b Separator Wear Protection (e.g. O-Sepa)



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2.5 Installation Arrangements

Basic solutions (fig. 19):

* cyclone air separator
* single pass separator with separate mill ventilation
* single pass separator combined with mill ventilation

Selection criteria:

* required cooling capacity
* condition of existing mill venting system
* available space
* investment and operating costs

	cyclone air separator	**single pass combined mill ventilation**	**single pass separate mill ventilation**
cooling effect	very low	medium	high
space requirement	approx. same as for mechanical separator	low for separator additional space requirement for bagfilter depends on capacity of existing mill filter	low for separator large bagfilter required
mill ventilation system	existing system is kept	depending on capacity and condition the existing mill filter is: replaced enlarged kept as it is	existing system is kept

Figure 19a Cyclone Air Separator

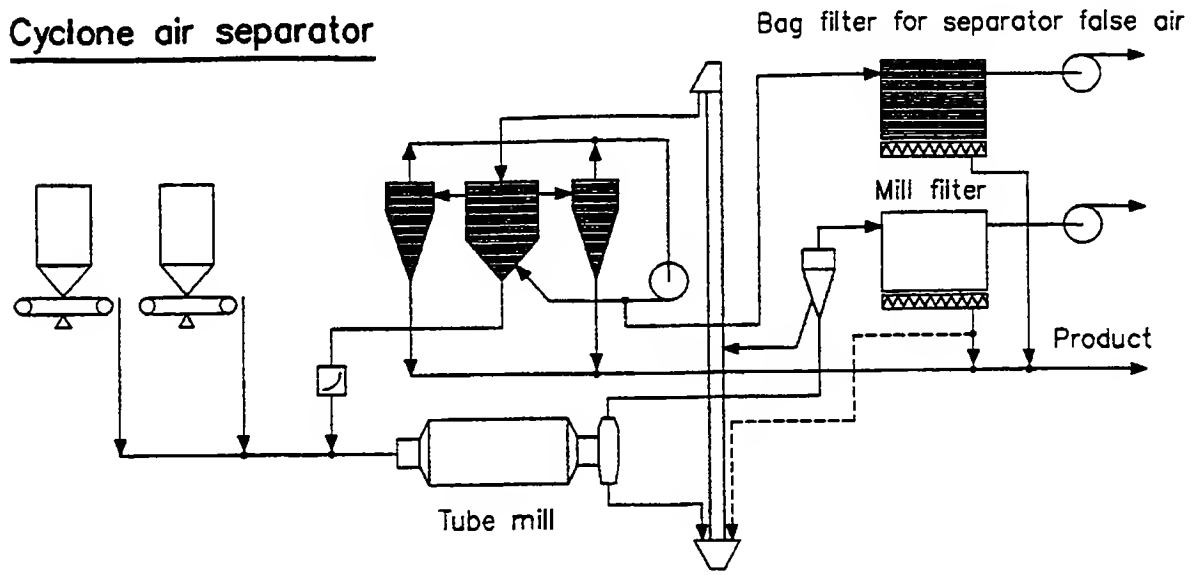

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Figure 19b Single Pass Separator Combined with Mill Ventilation

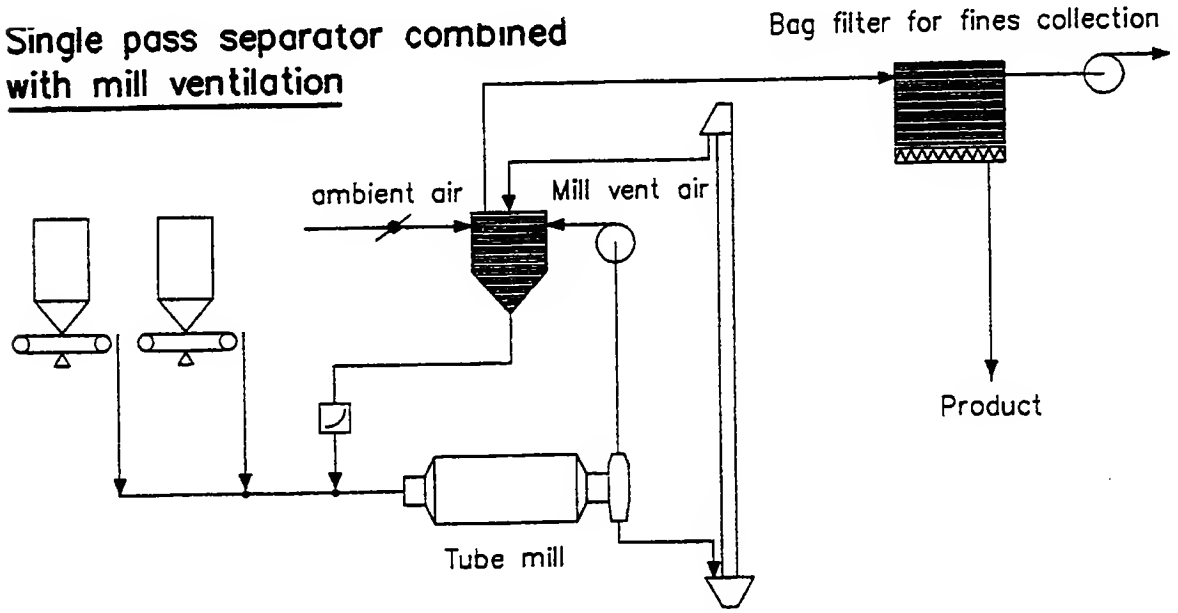
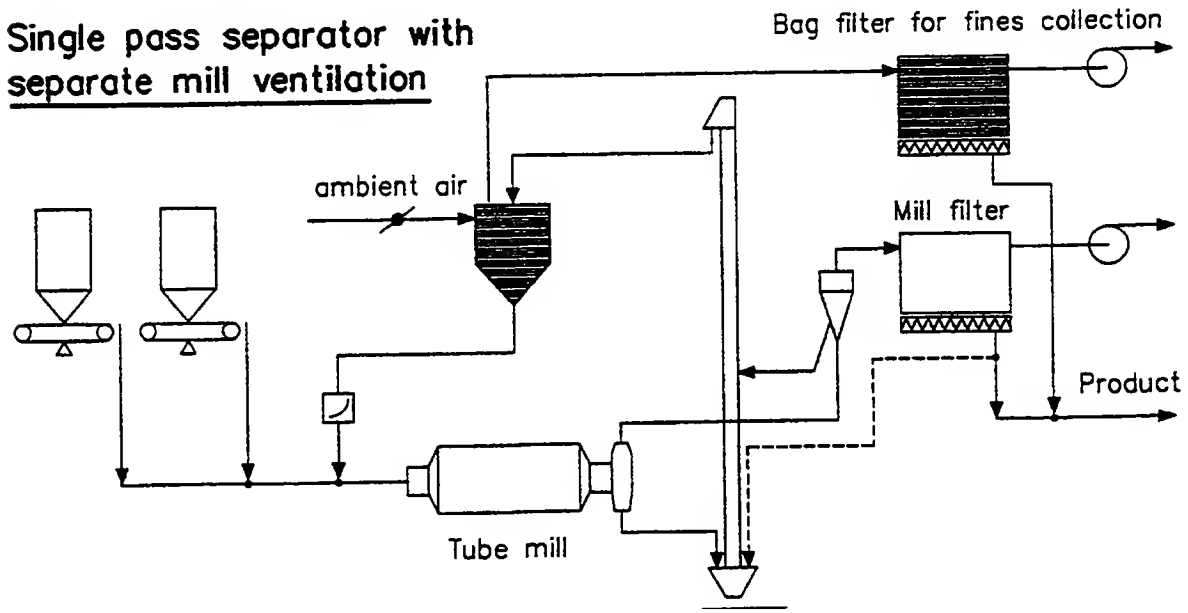


Figure 19c Single Pass Separator with Separate Mill Ventilation



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1. INTRODUCTION

The ideal situation in grinding would stipulate that a particle of feed would be discharged from the grinding circuit as soon as it has been reduced to the required size. Thus, the grinding forces would be applied only to the oversize particles. However, due to the action of tumbling mills, any attempt to reduce all the feed to a finished product in one step results in costly overgrinding.

To control top size of discharge from the grinding circuit, sizing apparatus such as an air separator is employed, and the oversize constitutes the circulating load to the mill. In this manner, particles greater than a specified maximum size are prevented from leaving the circuit, and particles below the desired size are not recirculated through the mill.

The separation itself has an essential influence on the grinding performance in the mill, and therefore it is necessary to evaluate characteristics performance figures.

Proper functioning is mainly influenced by:

- Separator adjustment and technical condition, e.g. distributor plate speed, number and position of spin rotor blades, wear on fan and spin blades, air in-leaks, etc.
- Separator feed, e.g. feed rate, particle size distribution, moisture content, density, etc.
- Separation air, e.g. temperature, density, viscosity, moisture content, etc.

Four levels of representation of separator performance are recognized, in which:

a) Only the total mass of the coarse (or fine) stream is considered relative to the mass of feed, expressed by the term 'yield'. This is the simplest form of assessment.

b) Cumulative size distributions of feed, fine and coarse streams are included, leading to the term 'recovery' or 'efficiency'.

c) The extent to which a feed size class appears in the fine and coarse stream is calculated for all size classes and leads to the term 'classification function' or 'Tromp curve' with its characteristics parameters cut point, sharpness off cut and by-pass.

d) Stochastic errors are taken into account (not dealt with in this paper).

Figure 1 Mechanical Air Separator

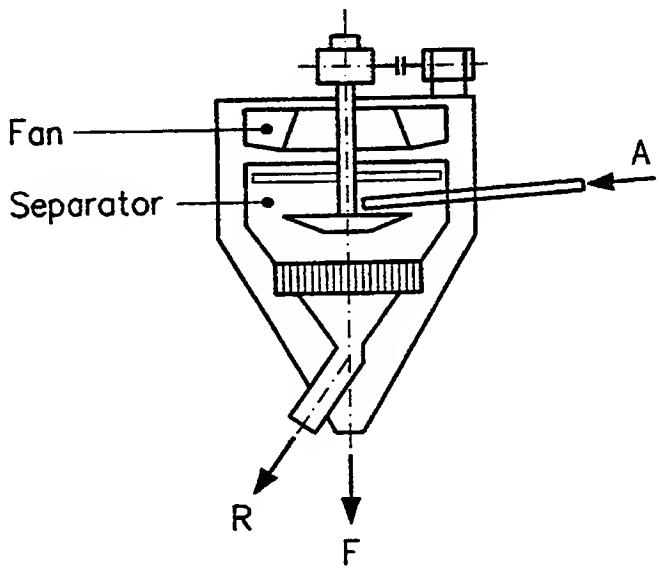

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Figure 2 **Cyclone Air Separator (Rotor Type)**

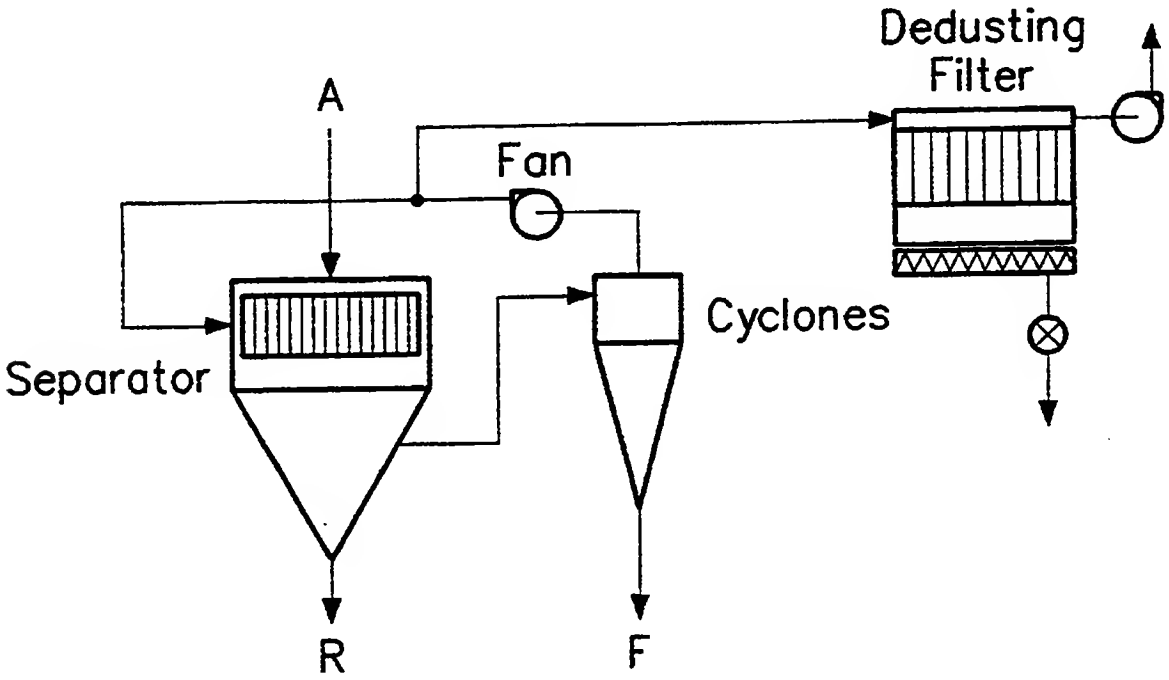
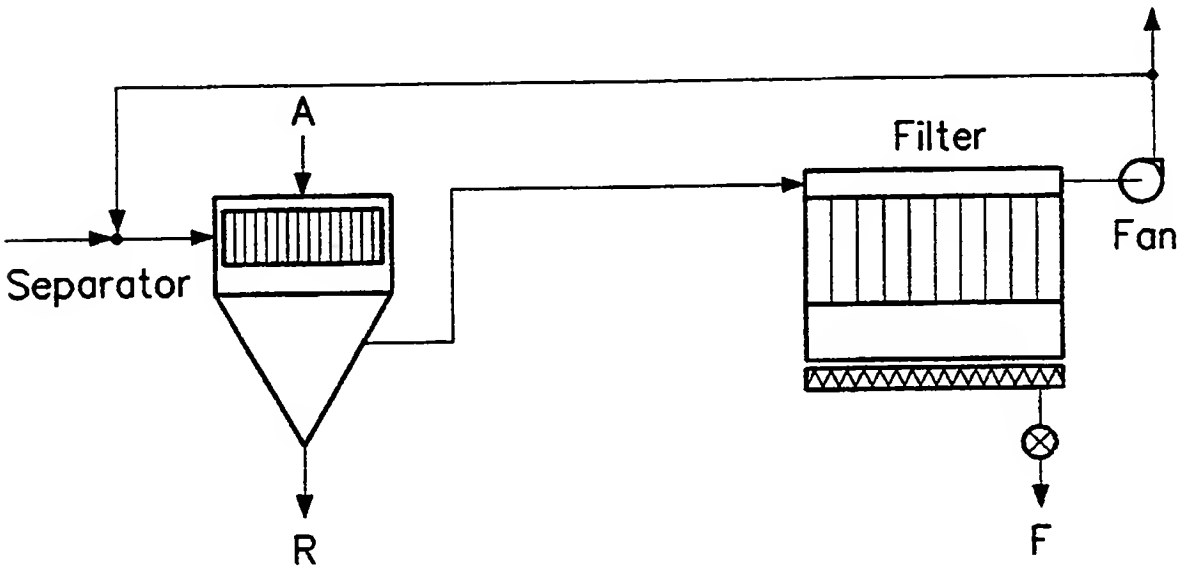


Figure 3 **Single Pass Separator (Rotor Type)**



2. GENERAL CALCULATING METHODS

2.1 Definition of a Closed Grinding Circuit

If the oversize in the mill discharge is recycled to the mill after a classification stage, the mill is considered to be operating in a closed circuit.

Figure 4 Symbols used for Mass Balances

Description	Mass flow [t/h]	Fineness [%] passing	Spec. surface [cm^2/g], Blaine
Mill feed	M	-	-
Separator feed (= mill discharge)	A	a	B_A
Fine fraction (Product)	F	f	B_F
Coarse fraction (Tailings)	R	r	B_R

2.2 Basic Equations (Mass Balance)

In steady-state operation, the mass flow of feed is equal to the sum of mass flows of fine and coarse streams.

$$A = F + R \qquad \text{eq. (1)}$$

A mass balance can also be formulated for a portion of the streams finer than a certain particle size x [µm] (see **figure 5**).

Let a, f and r denote the percentage finer than size x in the feed, fine and coarse stream. As long as no comminution takes place in the separator, the amounts of material less than size x entering and leaving the separator are equal, hence:

$$A.a = F.f + R.r \qquad \text{eq. (2)}$$

The values of a, f and r are found by a particle size analysis (e.g. sieve analysis, laser diffraction analysis, etc.).

Yet another formulation of the mass balance can be made for a portion which is limited by a lower and upper particle size x1 and x2 (see **figure 5**).

Let a, f and r denote the percentage of material finer than size x2 and larger than size x1 in the feed, fine and coarse stream. As long as no comminution takes place in the separator, the amounts of material between x1 and x2 that enter and leave the separator are equal, hence:

$$A.\Delta a = F.\Delta f + R.\Delta r \qquad \text{eq. (3)}$$

The values of a, f and r are found by a particle size analysis.

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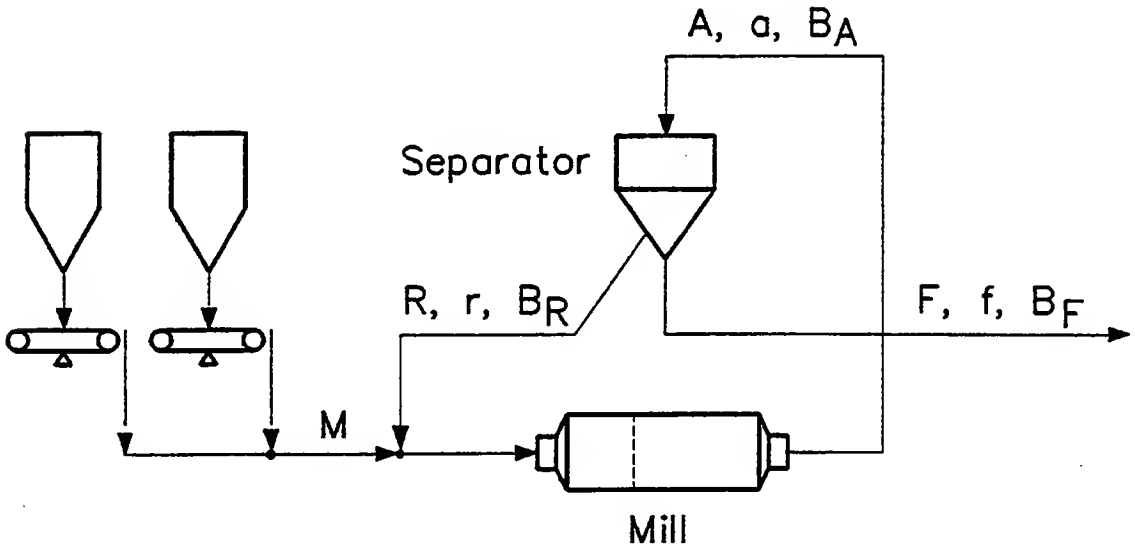
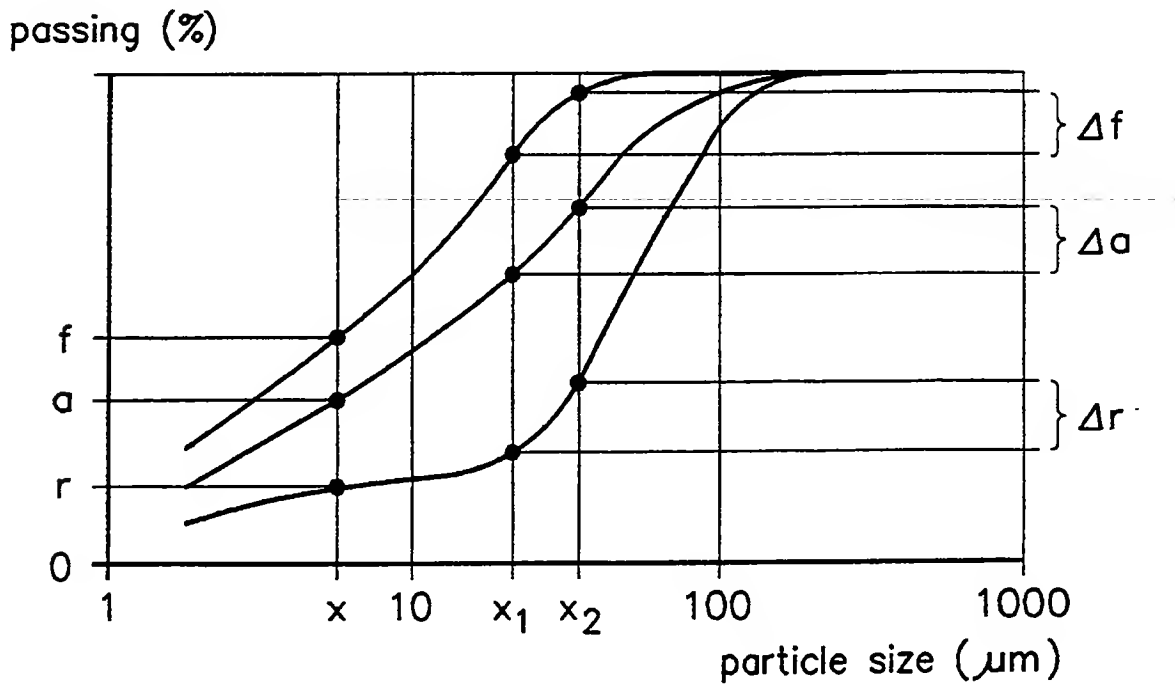
Figure 4 Symbols



Basic equations

A	= F + R
A · a	= F · f + R · r
A · Δa	= F · Δf + R · Δr

Figure 5 Definition Fineness



2.3 Circulating Load

The circulating load in closed-circuit grinding is defined as the mass of the separator feed A divided by the mass of the fine fraction F, or :

$$u=\frac{A}{F} \qquad \text{eq. (4)}$$

The actual value of the circulating load depends on various factors, such as mill design, grinding efficiency, product fineness, etc., but as a guide the following values can be given:

Cement mill	low fineness	u = 1.5 to 2
	high fineness	u > 2
Raw mill		u = 2 to 2.5

2.3.1 Calculation of Circulating Load

Formula (4) can be used directly if the mass flows of fines and feed (or rejects) are given. Most mills are equipped with weigh feeders, so M is known, which is equal to F in steady-state operation.

If no weighing equipment for separator feed or rejects is installed, u must be determined using particle size analysis data and formula (1), (2).

A = F + R (1)	A.a = F.f + R.r (2)	u = A/F (4)

$$u=\frac{(f-r)}{(a-r)} \qquad \text{eq. (5)}$$

It is recommended to use the u-values calculated by this formula with caution, because inevitable errors in the determination of the particle size distributions affect the result considerably.

2.4 Separator Efficiency

Efficiency takes some account of size distribution, to the extent that it is defined as the recovery of a feed size class (0 to x µm) into the fines stream.

$$\eta(x)=\frac{F\cdot f}{A\cdot a}\cdot 100[\%] \qquad \text{eq. (6)}$$

$$\eta(x)=\frac{f}{u\cdot a}\cdot 100[\%] \qquad \text{eq. (7)}$$

If x is chosen as the maximum particle size present in the feed, η(x) is equal to 1/u.

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Figure 6 shows efficiency curves for two different values of u.

Figure 6 Circulating Load, Efficiency

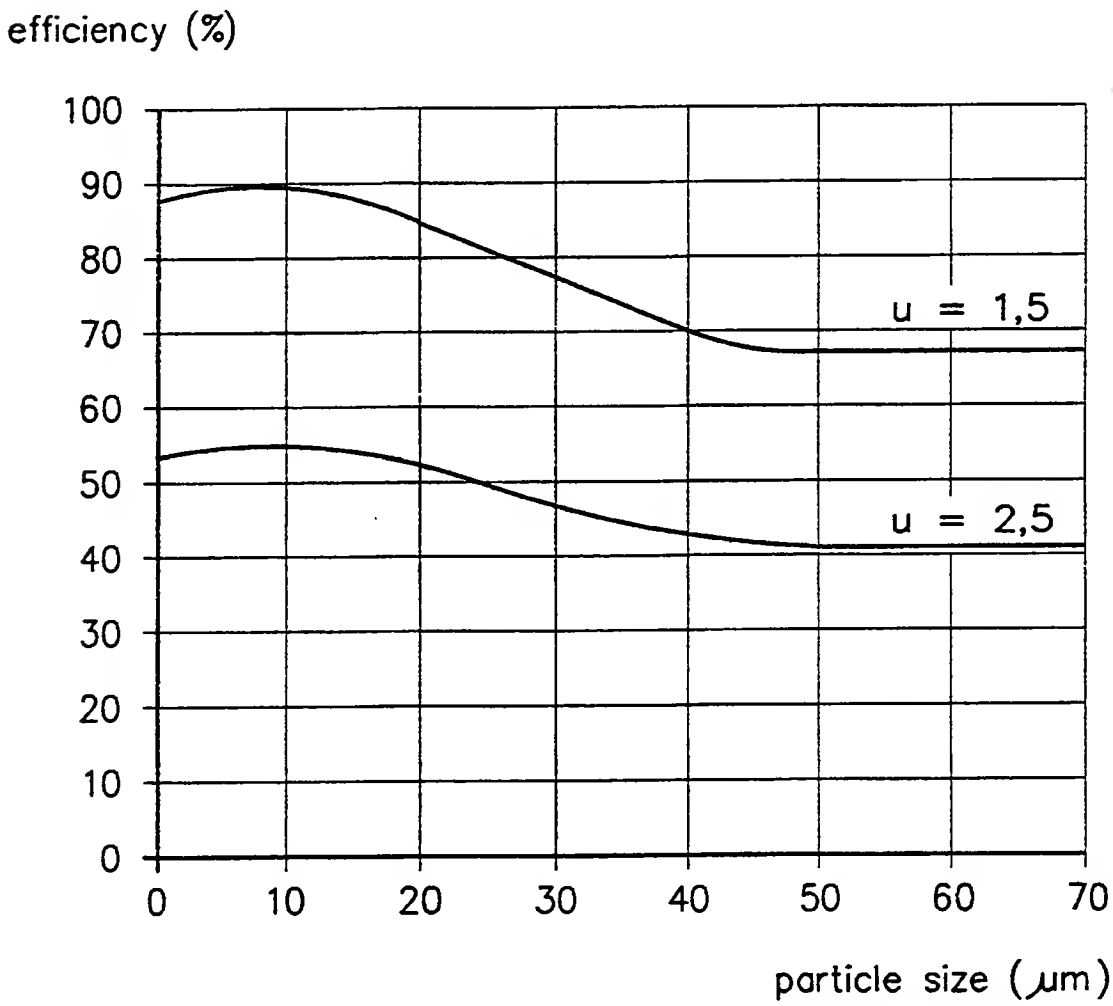
Circulating load

$$u = \frac{A}{F}$$

$$u = \frac{f - r}{a - r}$$

Separator efficiency

$$\eta \frac{f}{u \cdot a}$$



2.5 Classification (Tromp) Curve

The information given by the efficiency curve is not detailed enough because the defined fractions 0 to x [µm] are wide.

If we ask for the recovery of a feed size class with lower limit x1 and upper limit x2 (x1 greater zero) into the coarse stream, we arrive at the Tromp value for the coarse stream.

$$t_r = \frac{R \cdot \Delta r}{A \cdot \Delta a} 100[\%] \qquad \text{eq. (8)}$$

or

$$t_r = \frac{\Delta r}{\Delta a}\left(1 - \frac{1}{u}\right) 100[\%] \qquad \text{eq. (9)}$$

If this ratio is obtained for a number of size classes and plotted against x, there results the classification curve (**figure 7**). It is also called the Tromp curve after the name of the man to whom it is assigned.

2.6 Features of the Classification Curve

2.6.1 Cut Point

The cut point d_{50} corresponds to 50 % of the feed passing to the coarse stream as seen in **figure 8.** It is therefore that size which has equal probability of passing to either coarse or fine streams.

2.6.2 Sharpness of Separation

The sharpness of separation is defined as follows:

$$k = \frac{d_{75}}{d_{25}} \qquad \text{eq. (10)}$$

where d_{75} and d_{25} denote the sizes with Tromp values of 75 % and 25 % **(figure 8).**

For an ideal separation k would be 1.

2.6.3 By-Pass Effect

Ideally, the Tromp curve is asymptotic to the abscissa at ordinate values of zero and unity.

In practice, it is often the case that the lower asymptote occurs at ordinate values a ' greater than zero (**figure 9**), i.e. a portion of each size fraction bypasses the classifying action. Expressed in an other way, part of the feed reports to the coarse stream independently of its particle size.

Experience has shown that the bypass parameter a' varies with classifier feed rate, and hence it is difficult to describe a single Tromp curve which is representative of the classifier.

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Figures 7, 8 & 9 Tromp Curves

Basic equations

$$t_r = \frac{\Delta r}{\Delta a}\left(1-\frac{1}{u}\right)100$$

Figure 7 Tromp Curves (continued)

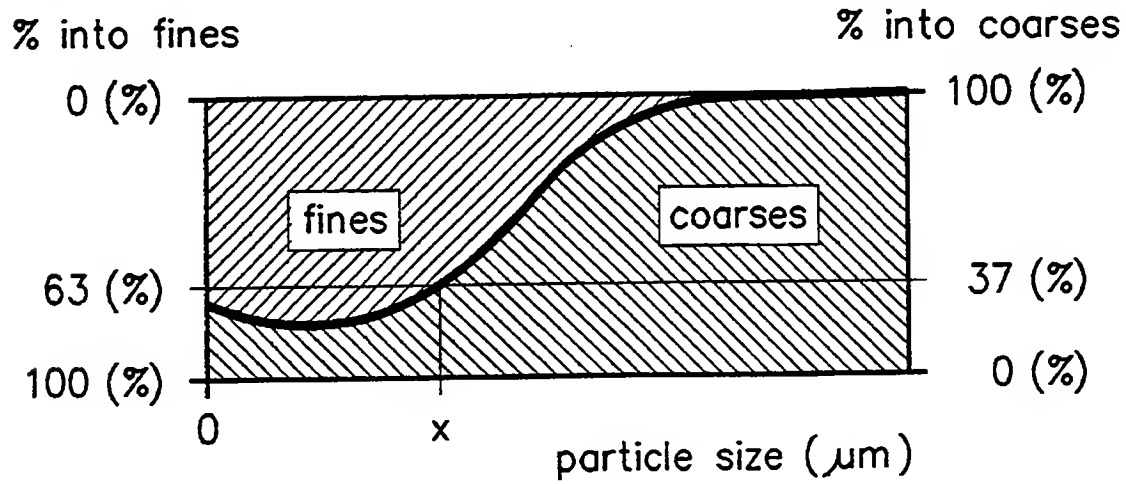

Figure 8 Tromp Curves (continued)

sharpness of cut k = $\frac{d_{75}}{d_{25}}$

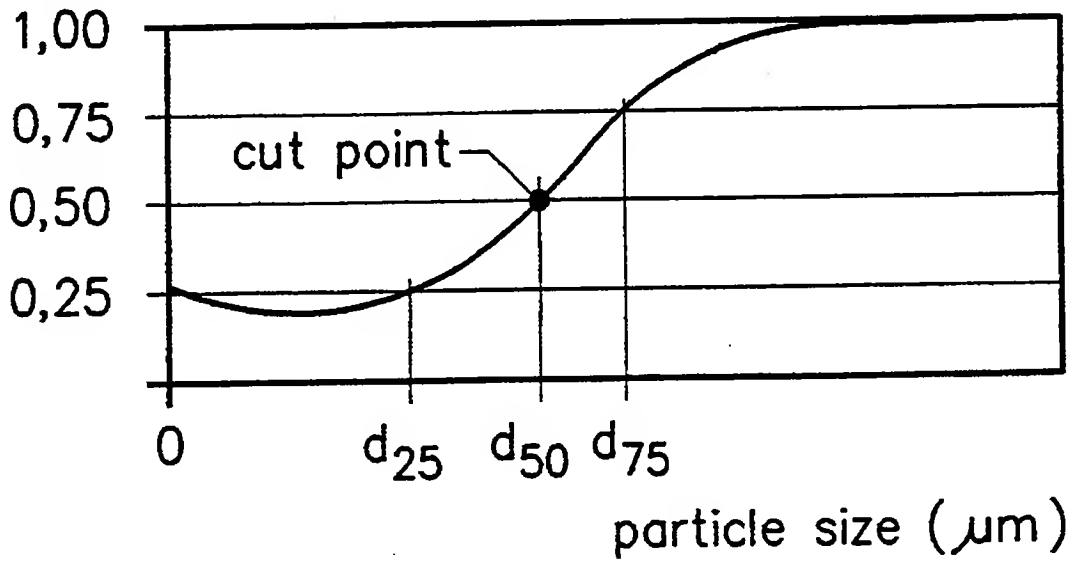

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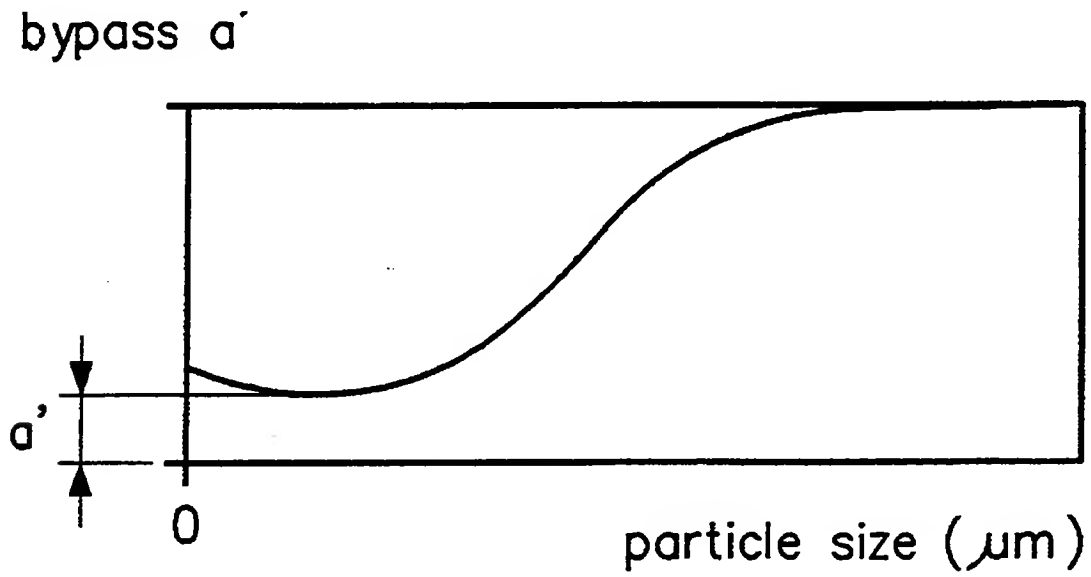
Figure 9 Tromp Curves (continued)



With mechanical air separators of Heyd or Sturtevant type it frequently happens that the bypass value exceeds 50 %, so that the definitions given above for d_{50} and k are no more applicable.

It is then common practice to introduce a separator model which explicitly includes the bypass effect, shown in **figure 10.**

The 'reduced' Tromp curve with

$$t_r^* = \frac{(tr - a')}{(100 - a')} 100[\%]$$

allows for determination of d_{50} and k*.

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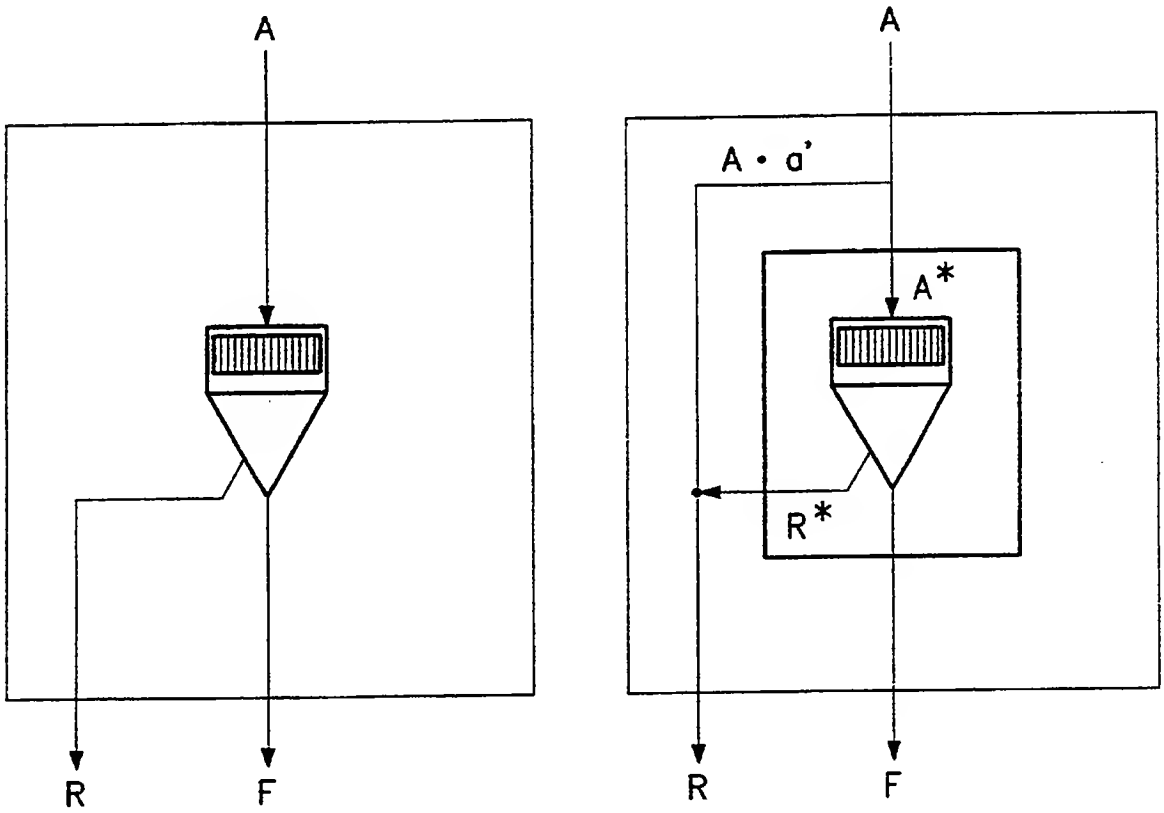
Figure 10 Separator Model



$$t_r = \frac{R \cdot \Delta r}{A \cdot \Delta A} \qquad t_r^* = \frac{R^* \cdot \Delta r^*}{A^* \cdot \Delta a^*}$$

$$R^* \cdot \Delta r^* = R \cdot \Delta r - a' \cdot A \cdot \Delta a$$

$$A^* = A(100 - a')$$

$$t_r^* = \frac{R \cdot \Delta r - a' \cdot A \cdot \Delta a}{A \cdot A(100 - a') \cdot \Delta a}$$

$$t_r^* = \frac{(t_r - a')}{(100 - a')}$$

3. TEST PROCEDURE

3.1 Target of Test and Conditions

The separation result is characterized by weight and fineness of coarse and fine streams. These values and the operating variables which affect separator performance (e.g. separator adjustment, feed, separating air, etc.) must be recorded during the test.

Since separator performance is strongly dependent on feed rate and feed size distribution, it is of utmost importance that stable operation of the mill is maintained during the test, which in this context means constant classifier feed rate.

Variables correlated with this figure are e.g. power consumption of the bucket elevator or separator rejects mass flow (provided a scale is installed). If no automatic control system is available, the fresh feed rate to the mill must be regulated such that the power consumption of the bucket elevator is constant.

3.2 Sampling and Duration of Test (figures 11, 12, 13)

The most accurate determination of A, F and R can be done if two of the values can be measured by scales and the third value is calculated by means of equation (1). In this case, the duration of the test is about 10 minutes, and samples should be taken in intervals of 1 to 2 minutes.

It is very often that only the fresh feed M to the mill can be measured. It is assumed that the weight of the fine fraction F is equal to the mill feed M. Separator feed rate A and coarse stream R can only be determined by means of the circulating load u, which in turn is calculated applying formula (5) to the particle size distributions of A, F and R. Duration of test is about 10 minutes, and samples should be taken every 1 to 2 minutes.

Then the individual samples of each sampling point can be homogenized to one composite sample for A, F and R. Before doing this, it is recommended to check for outliers due to e.g. instable mill operation by checking the residue on a sieve of medium size, e.g. 30 µm. Samples with extreme values should be eliminated.

Two separators operating in parallel usually do not have equal feed rates, therefore it is recommended to carry out a separate test for each unit.

3.3 Sieve Analysis

As mentioned earlier, it is necessary to determine the particle size distribution (PSD) of the three samples A, F and R.

In many plants it is only possible to carry out dry sieving tests down to 30 µm. For an accurate separator judgment the PSD must be known also in the finer range. Below 30 µm, wet sieving is applied.

A more advanced method of PSD determination would be the application of a laser diffraction analyzer.

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Figure 11 Sampling Points - Single Pass Separator

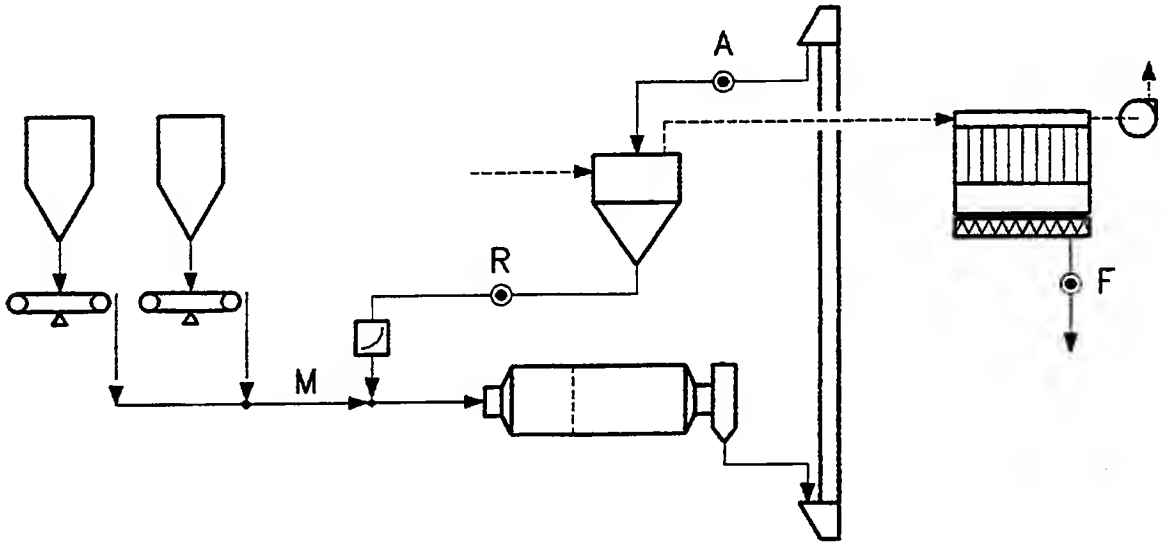

Figure 12 Sampling Points - Cyclone Air Separator

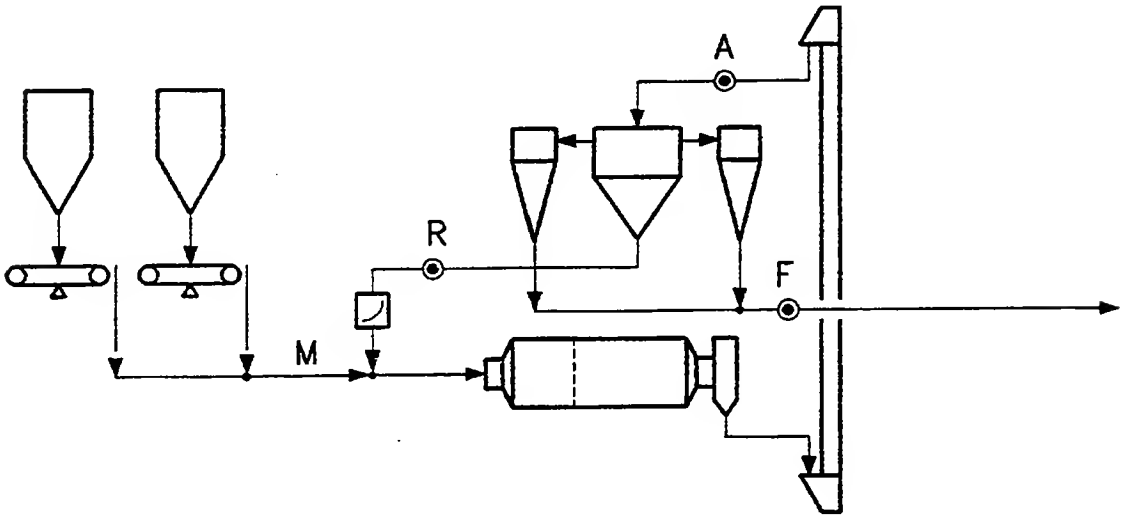

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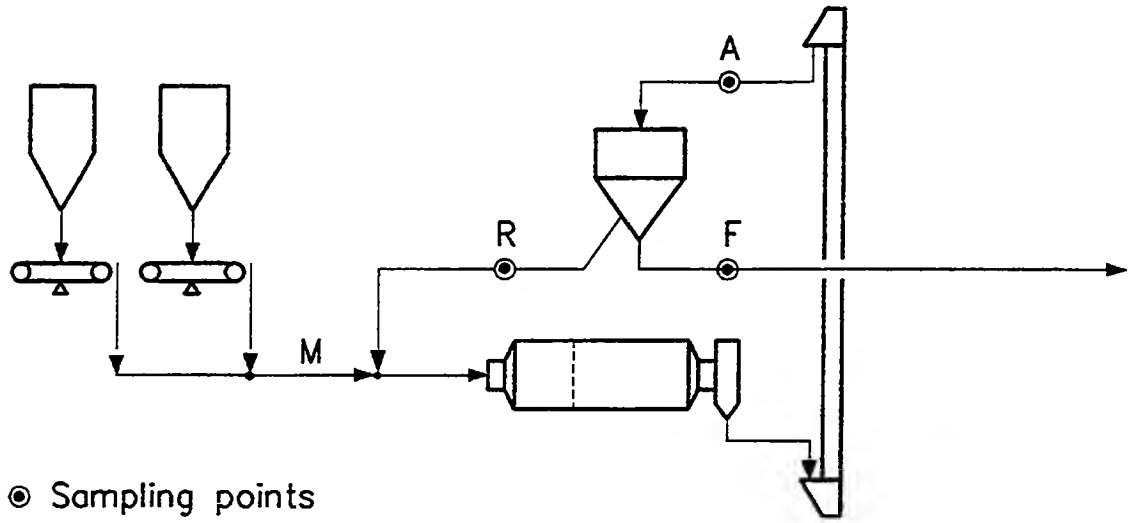
Figure 13 Sampling Points - Mechanical Air Separator



3.4 Evaluation of Test Results

The further procedure comprises calculation of:

- Mass balance
- Circulating load
- Efficiency, including graph
- Tromp values, including graph
- Cut point, bypass and sharpness of separation

The report at hand explains the basic methods how to carry out a separator performance test. These methods enable to define and evaluate the main operational characteristics of a closed grinding circuit, to compare them with other cases and with ideal values.

The optimum solution for a particular case must be found in several steps, the evaluation of each single step indicating a better choice of operating variables for the next one. No universal method is available to find the ideal solution at the writing desk without experiments.

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4. PRACTICAL CALCULATION AND EXAMPLE

4.1 Data of Separator and Mill

Plant	A	
Date of test	12.06.1991	
Separator:		
Make, type	Q&K QS 225	(HES)
Rotor diameter	2.25	[m]
Rotor height	2.25	[m]
Rotor speed	120	[rpm]
Installed motor power	220	[kW]
Separation air amount	170'000	[m^3/h]
Fineness regulation	rotor speed variation	
Year of start-up	1991	
Mill:	Cement mill, No. 4	
Type	Two compartment, end discharge	
Diameter, length	4.2 / 15.4	[m]
Installed motor power	3'600	[kW]
Year of start-up	1980	
Mill feed	Weigh feeder	
Control	automatic mill control system	
Flowsheet	**figure 14**	

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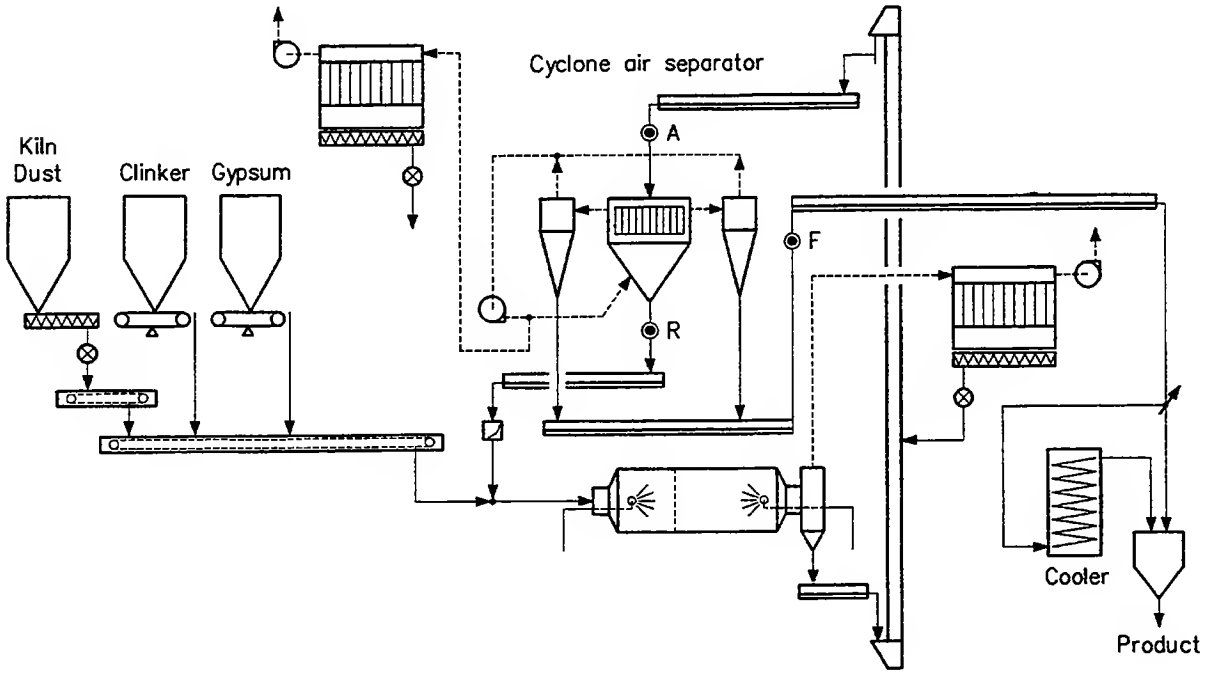
Figure 14 Flowsheet - Operation of Separators



4.2 Test

During the test a cement has been produced with 93.1 % clinker, 4.6 % gypsum and 2.3 % kiln dust. Since the system was equipped with a scale for the returns, test duration was only 30 minutes.

Average mill feed rate was 120 t/h, average returns rate 126 t/h.

4.3 Sampling and Sieve Analysis

Sampling points:	shown in figure 14, marked
Number of samples:	16 per sampling point
Sieve analysis:	Particles > 200 µm dry screening Particles < 200 µm CILAS laser analyzer

The particle size distributions of feed, coarses and fines are listed in **figure 15**, the corresponding graphs are shown in **figure 16.**

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4.4 Evaluation

4.4.1 Circulating Load u

1) Calculation by means of measured mill feed rate and rejects rate

$$u=\frac{A}{F}=\frac{(R+F)}{F}=\frac{(126+120)}{120}=2.05$$

2) Calculation by means of particle size distributions

$$u=\frac{(f-r)}{(a-r)}(equation5)$$

The circulating load for each particle size (e.g. 1,2, 4 µm etc.) can now be calculated and then the mean value of the circulating load can be determined.

A simpler form is to calculate the sum of a, f, and r and then to calculate u by means of the formula:

$$u=\frac{(\sum f-\sum r)}{(\sum a-\sum r)}$$

$$u=\frac{(607.7-285.1)}{(463.1-285.1)}$$

$$u=1.81$$

Figure 15 Size Distribution

size (µm)	feed	coarses (% passing)	fines
1	3.6	1.9	4.9
2	6.1	3.5	8.3
4	11.0	6.3	14.7
8	18.5	10.1	25.4
16	31.2	14.4	44.9
24	39.7	16.1	58.8
32	47.8	19.1	71.1
48	60.6	28.1	87.0
64	67.7	37.1	92.6
96	81.8	59.5	100.0
200	95.1	89.0	100.0
Sum	463.1	285.1	607.7

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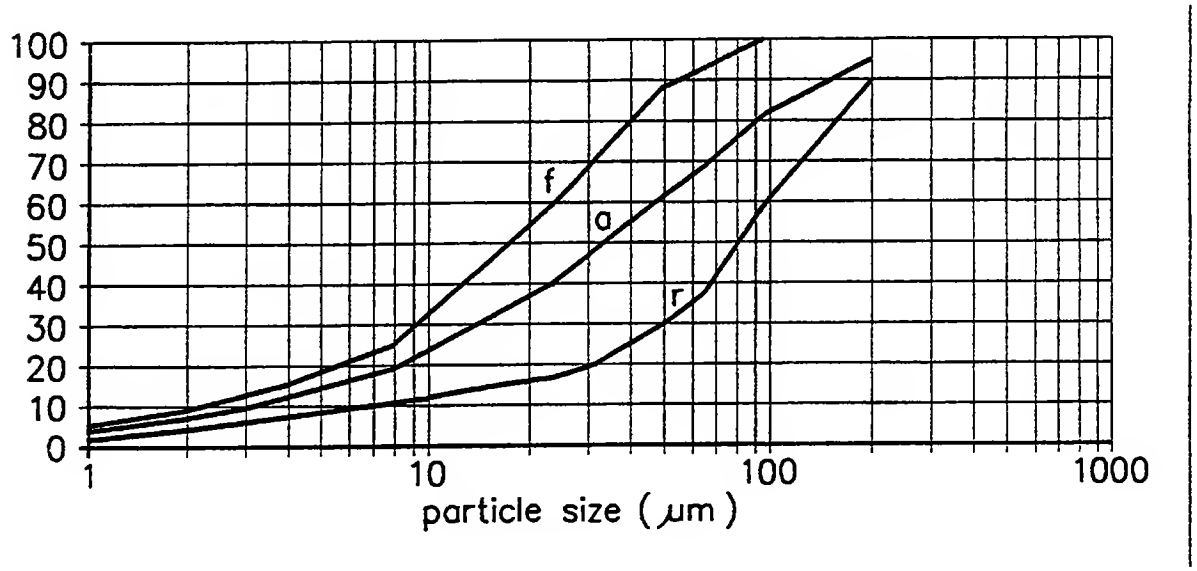
Figure 16 Particle Size Distribution



4.4.2 Separator Feed A

The mill feed M is known and is equal to F. The separator feed A can now be calculated by means of equation 4:

$$u=\frac{A}{F}\rightarrow A=u\cdot F=u\cdot M$$

$$A=1.81*120=217[t/h]$$

4.4.3 Coarse Fraction R

R = A - F = 97 t/h

4.4.4 Efficiency

Equation 7 is used for calculating the efficiency.

$$\eta = \frac{f}{a\cdot u}\cdot 100[\%]$$

The calculated values are listed in **figure 17**, and the corresponding graph is shown in **figure 18**. It is to state that for the calculation the value u = 1.81 has been used.

4.4.5 Tromp Values

Equation 9 is used for calculating the Tromp values.

$$t_r = \frac{\Delta r}{\Delta a}\left(1-\frac{1}{u}\right)100[\%]$$

The calculated values are listed in **figure 17**, and the corresponding graph is shown in **figure 18**.

4.4.6 Cut Point

The cut point was determined graphically, and its value is

d_{50} = 52 [µm]

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4.4.7 Sharpness of Separation

Equation 10 was used for the calculation

$$k=\frac{d_{75}}{d_{25}}=3.4$$

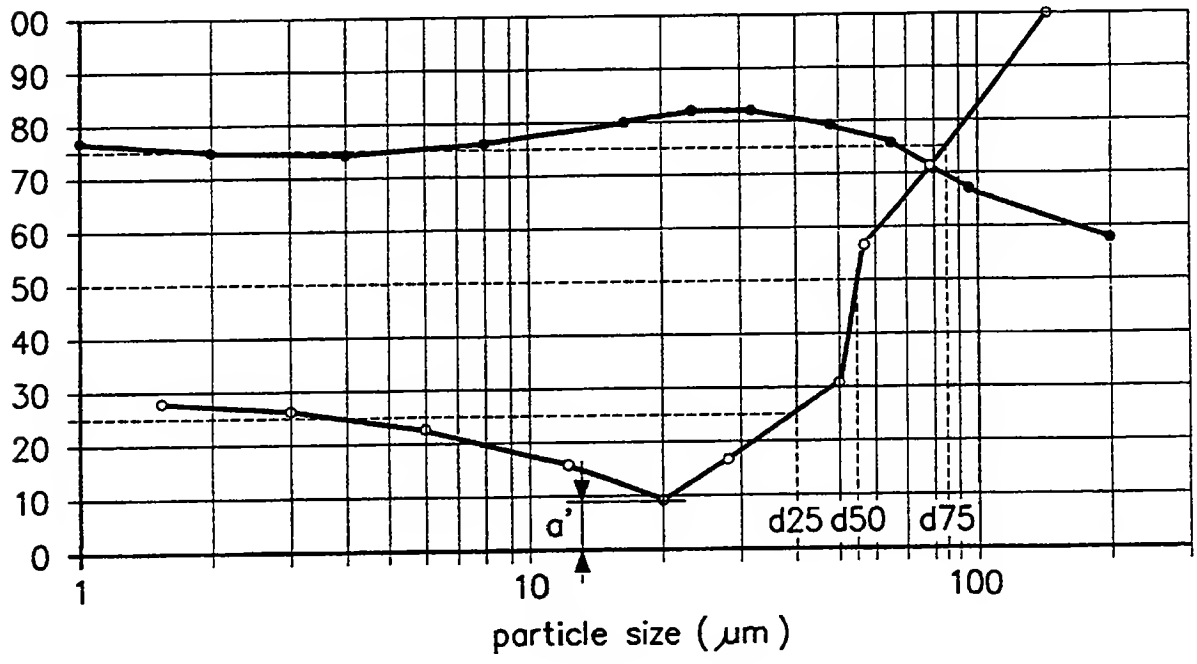
4.4.8 Bypass

The minimum of the Tromp curve is located at 20 µm, the bypass value is 9 %.

Figure 17 Efficiency, Tromp Curve - Calculation

size (µm)	efficiency (%)	Tromp (%)
1	76.4	23.7
2	74.7	27.9
4	74.2	26.6
8	75.8	22.2
16	79.4	15.4
24	81.9	9.0
32	82.2	16.7
48	79.3	31.5
64	75.6	56.6
96	67.5	71.1
200	58.1	100.0

Figure 18 Efficiency, Tromp Curve



4.4.9 Specific Air Loads

$$\frac{feed}{airamount} = \frac{217'000[kg/h]}{170'000[m^3/h]} = 1.28[kg/m^3]$$

$$guidevalue \leq 2.5[kg/m^3]$$

$$\frac{fines}{airamount} = \frac{120'000[kg/h]}{170'000[m^3/h]} = 0.71[kg/m^3]$$

$$guidevalue \leq 0.8[kg/m^3]$$

4.4.10 Specific Rotor Load

D [m] rotor diameter
H [m] rotor height

$$\frac{fines[t/h]}{D \cdot 3.14 \cdot H} = \frac{120}{2.5 \cdot 3.14 \cdot 2.5} = 6.1[t/m^2/h]$$

$$guidevalue \leq 10[t/hm^2]$$

4.4.11 Conclusion

The evaluation of the separator performance test shows that separator efficiency is good, bypass is low and sharpness of separation is sufficient.

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Chapter 10

Raw Grinding Systems

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Raw Grinding Systems

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1. PURPOSE OF RAW GRINDING

The raw grinding is the size reduction stage for raw material between crushing and burning process (figure 1) and has to fulfill the requirements:

- Produce a raw meal fineness (particle size and particle size distribution) adequate for the production of the required clinker quality
- Provide the proper raw mix by proportioning the components in the correct percentages
- Mixing the components
- Drying of the raw material to enable a good handling of the raw meal

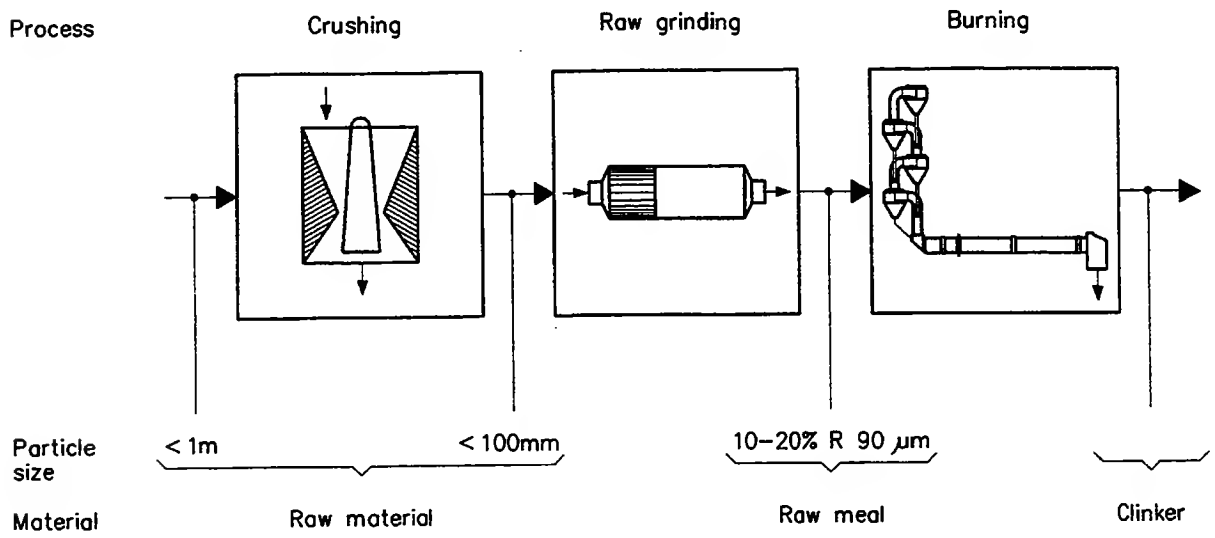
The raw grinding stage cannot homogenize the raw meal due to the short retention time in this process.

The raw grinding can be divided in

- Dry grinding process
- Wet grinding process

All new installations are, with few exceptions, dry grinding processes.

Figure 1 Purpose of Raw Grinding



2. RAW MATERIAL PROPERTIES AND PRODUCT QUALITY

- **Raw materials** and correctives used for the raw mix:
  - Limestone
  - Marl
  - Clay
  - Iron correctives (e.g. iron ore, slag)
  - Silica correctives (e.g. quartz sand, fly ash)
  - etc.
- Max. feed size of raw material components to the raw mill depend on the mill type and mill size (figure 2):

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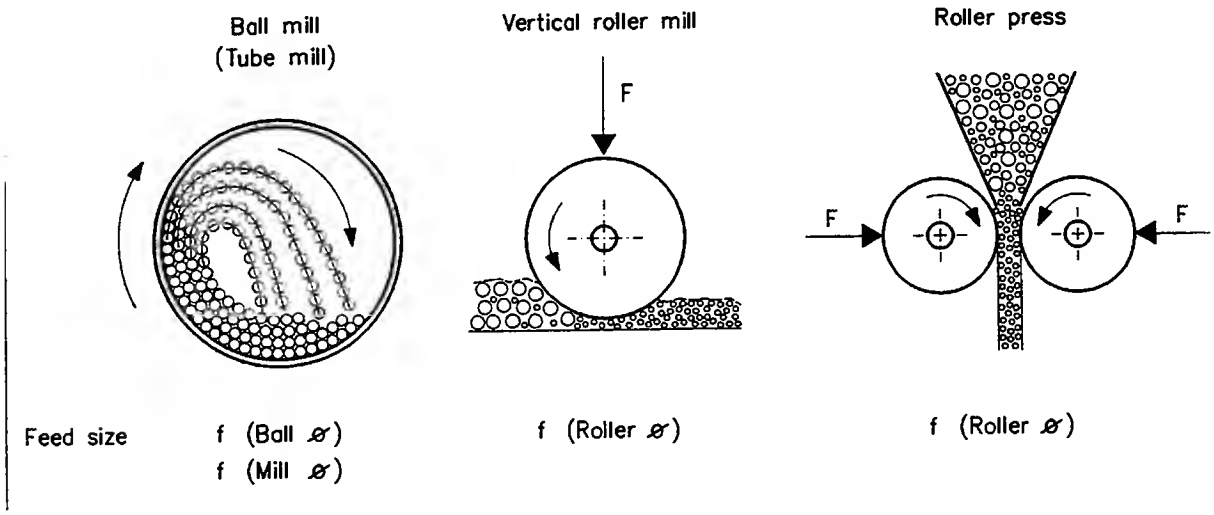
Tube mill	< 25 - 50 [mm] (max. ball size!)
Roller mill	< 100 [mm] (max. 5 - 7 [%] of roller diameter)
Roller press	< 50 [mm] (max. 3 - 5 [%] of roller diameter)
Autogenous mill	< 300 [mm]
Hammer/impact Crusher	< 200 [mm]

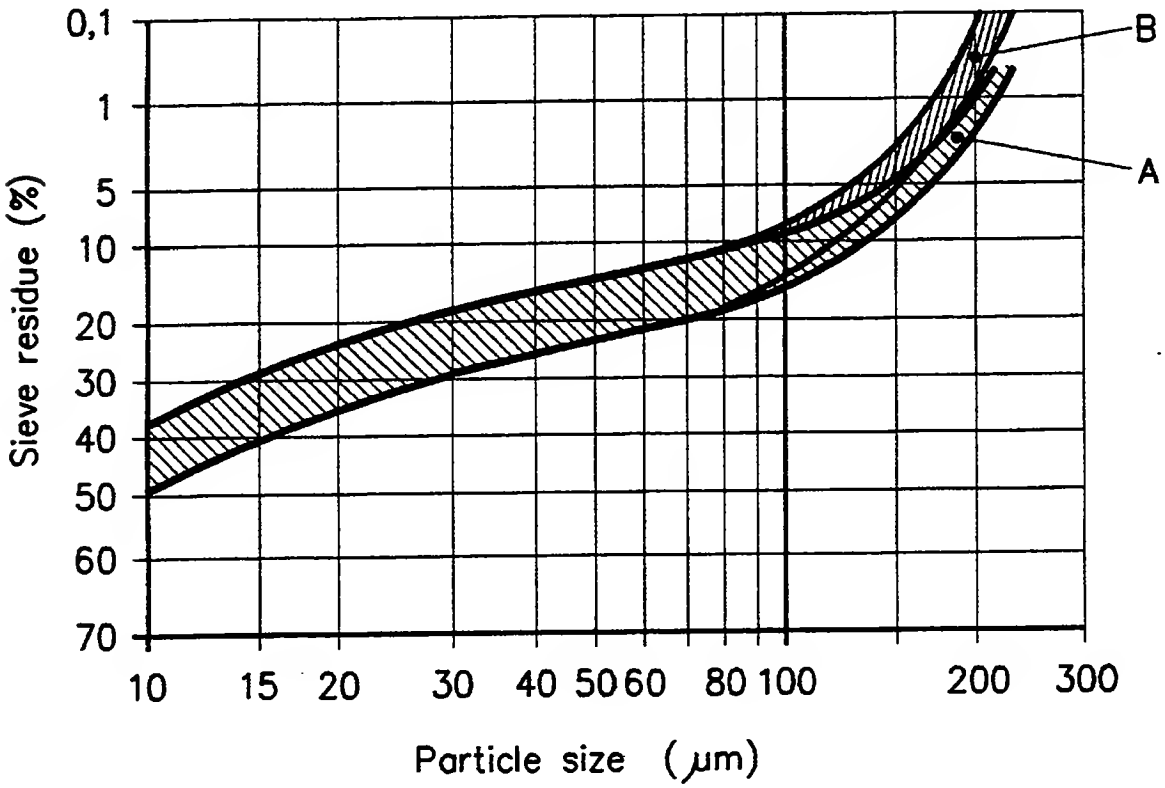
- Besides the max. feed size also the particle size distribution must be observed, and eventually adapted to the needs of the mill type (roller mill, roller press) to enable a stable grinding process.
- The level of raw material moisture limits the application and calls for a certain grinding/drying equipment.
- For high moisture contents (8 - 20 [%] H_20), the drying capacity is the critical item for dimensioning the grinding/drying system.
- For low moisture contents (< 8 [%] H_20), normally the grinding capacity is the predominant factor.
- Indications of moisture contents are always on **wet basis**.
- The stickiness of moist feed components require certain precautional measures for a safe handling such as
  - Bin design
  - Bin extraction facilities
  - Conveyors and chutes
  - Air locks (ev. heated)

Figure 2 Mill Feed Particle Size



- **Product (raw meal)**
  - Raw meal fineness depends on the requirements of burning process and is normally in the range of
    10 - 20 [%] R 90 [µm]
  - The fineness should be determined by the rule
    'Grind only as fine as necessary'
  - Even more important than the residue on 90 [µm] is the one on 200 [µm], as this fraction contains the poor burnable quartz particles. The aim is, to reduce the residue on 200 [µm], while maintaining the same residue on 90 [µm]. The general aim is
    < 0.5 [%] R 200 [µm]
  - A typical range for particle size distribution is shown in figure 3.
- **The raw meal** moisture is usually reduced to < 1.0 [%] H_2O, basically to achieve a good flowability and to allow an easy and save handling of the raw meal.

Figure 3 Typical Particle Size Distribution Raw Meal



Ranges of raw meal particle size distribution

A - Conventional
B - Improved (mainly by separator)

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Main influence of particle size distribution

⇐ Fine range	Coarse range ⇒
⇐ Grinding process	Design and operation of separator ⇒

3. BASIC RAW GRINDING EQUIPMENT (DRY GRINDING)

3.1 Available Raw Grinding Equipment (figure 4)

Main types:

- Tube mills
  - End-discharge mill
  - Center discharge mill
  - Air swept mill
- Vertical roller mill
- Roller press

Various types:

- Autogenous mill
- Hammer/impact mill

The tube mill is the most used raw grinding equipment up today.

Modern raw grinding equipments are the vertical roller mill for high production and high drying capacities and the roller press for low moisture contents and moderate capacities.

Autogenous and hammer/impact mills are only used for special applications. Autogenous mills are often fitted with a few balls for better efficiency.

Hammer/impact mills are mainly used for pregrinding/predrying purpose, ahead of a conventional tube mill system. They are often used with a kind of flash drier.

In the past, many complicated connections between various grinding/drying equipment have been installed. The trend today goes towards simple and reliable plants.

3.2 Selection Criteria for Raw Grinding Equipment

- Capacity of installation
- New installation or extension
- Moisture content of raw materials
- Raw material hardness (grindability of components)
- Raw material properties (stickiness, abrasiveness, etc.)
- Feed size of raw material (depending on crushing installation)

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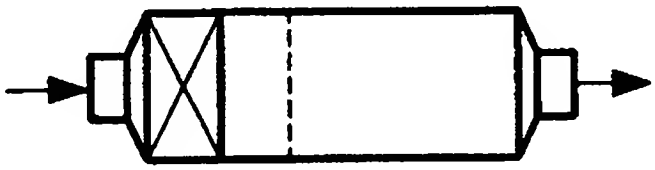
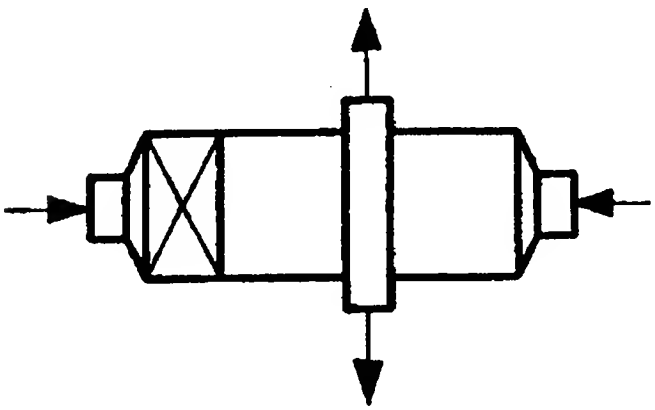
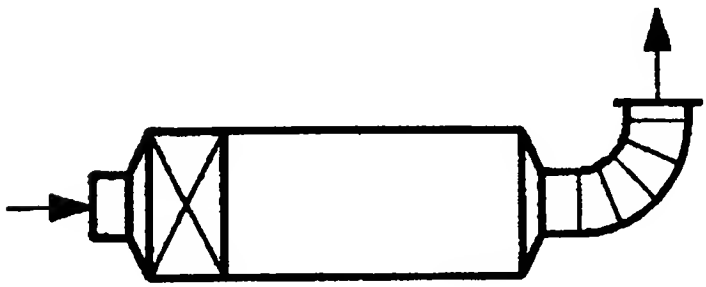
Figure 4a **Raw Grinding Equipments - Main Types**
Tube Mills



End-discharge



Center-discharge



Air-swept

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Figure 4b **Raw Grinding Equipments - Main Types**
Vertical Roller Mill

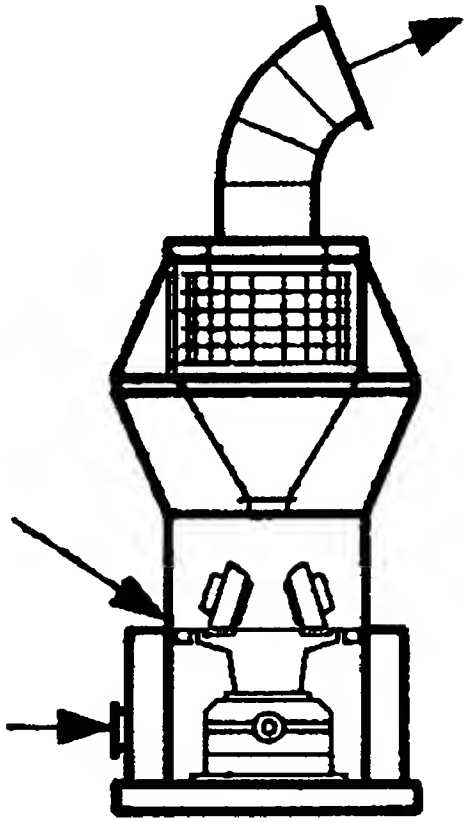

Figure 4c **Raw Grinding Equipments - Main Types**
Roller Press

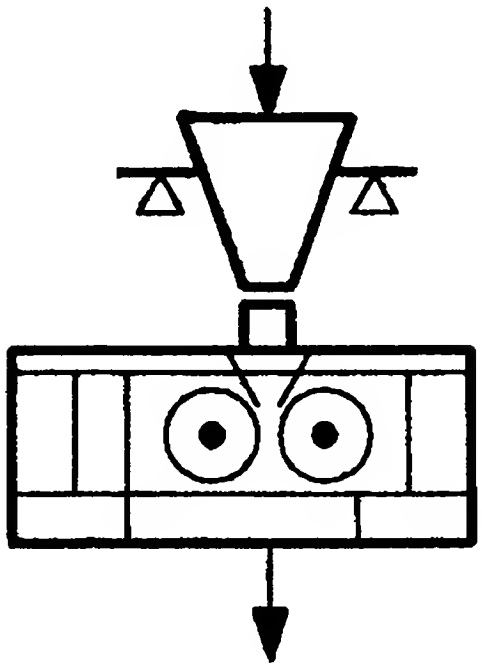

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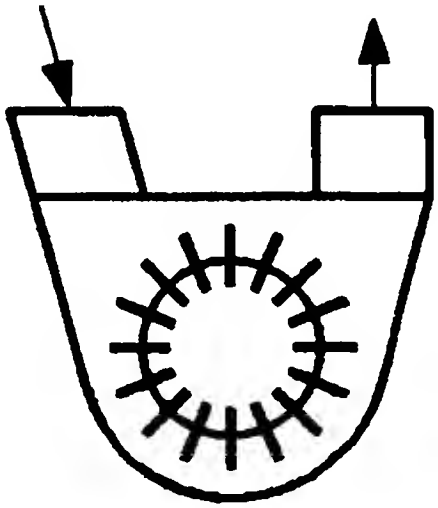
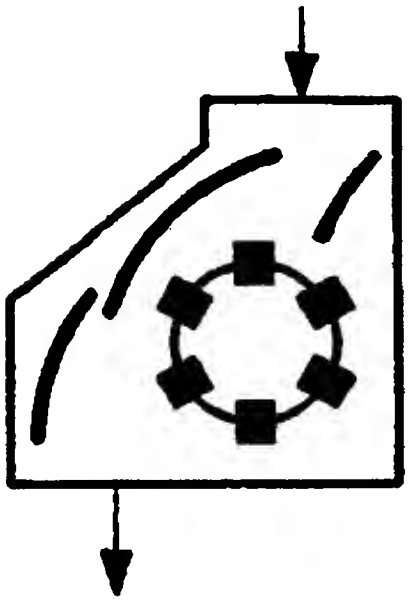
Figure 4d **Raw Grinding Equipments - Various Hammer/Impact Mill**



Hammer mill



Impact mill

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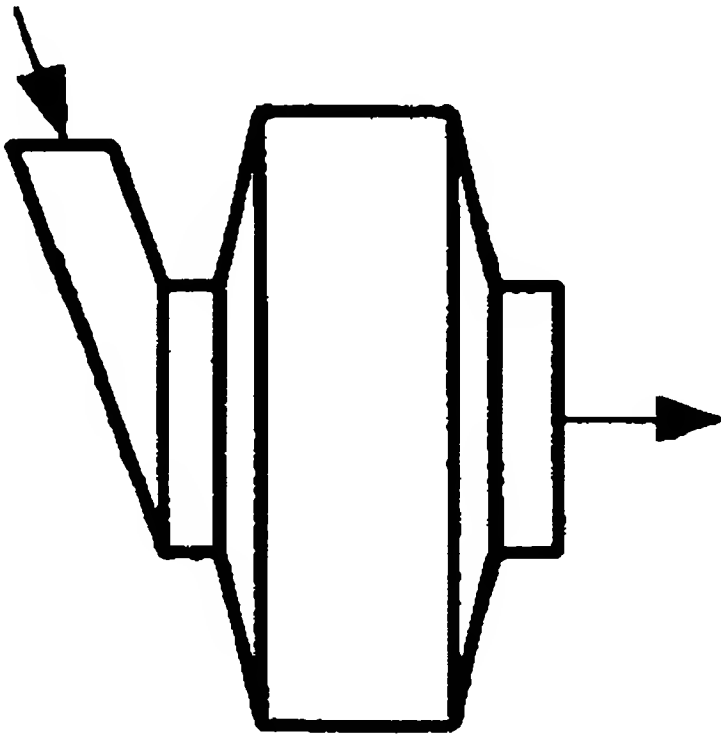
Figure 4e **Raw Grinding Equipments - Various**
Autogenous Mill



4. DRYING POSSIBILITIES AND PERFORMANCE

4.1 Heat Sources

Kiln exhaust gases (depending on kiln system and number of preheater stages)	< 350 [C]
Cooler exhaust air	< 350 [C]
Hot gas generator	< 600 [C]

Utilizing all kiln exhaust gases, in general, raw material can be dried up to 7 - 8 [%] H_20. For higher moisture contents, a hot gas generator can be installed. The limitation of drying with kiln exhaust gases is mainly given by the maximum available amount of gases at a production equilibrium between raw mill and kiln. All parts of equipment in contact with hot gases of 450 - 600 [C] have to be lined with refractories and bearings insulated to prevent from overheating.

Max. hot gas temperature for vertical roller mills is < 450 [C] at the mill inlet. The same applies for tube mills without drying compartment.

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4.2 Drying Possibilities (figure 5)

- Separate drying (predrying) in
  - Rotary dryers (drum dryers)
  - Dispersion dryers
  - Flash dryers, with or without hammer/impact crusher
- Drying in mills
  - Drying compartments within tube mills or ahead of the tube mills in overhung drying compartments
  - Drying in grinding compartments of tube mills (low drying capacities)
  - Drying in vertical roller mill casing
  - Drying in separator (e.g. for roller presses)

4.3 Drying Performance

The drying performance depends basically on:

- Type of drying equipment
- Volume and design of drying facility
- Hot gas temperature (energy level)
- Hot gas flow rate (restricted by max. gas speed in drying equipment)
- Raw material properties
- Particle size of material, when drying
- Retention time in drying equipment

Many times, the drying capacity of a given system is limited through the high false air intake in various places of the system, and the installed fan cannot match the entire gas flow rate.

Any drying capacity of mill can be increased by a preceding, separate drying process (e.g. flash drier etc.).

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Figure 5a Drying Possibilities - Predrying

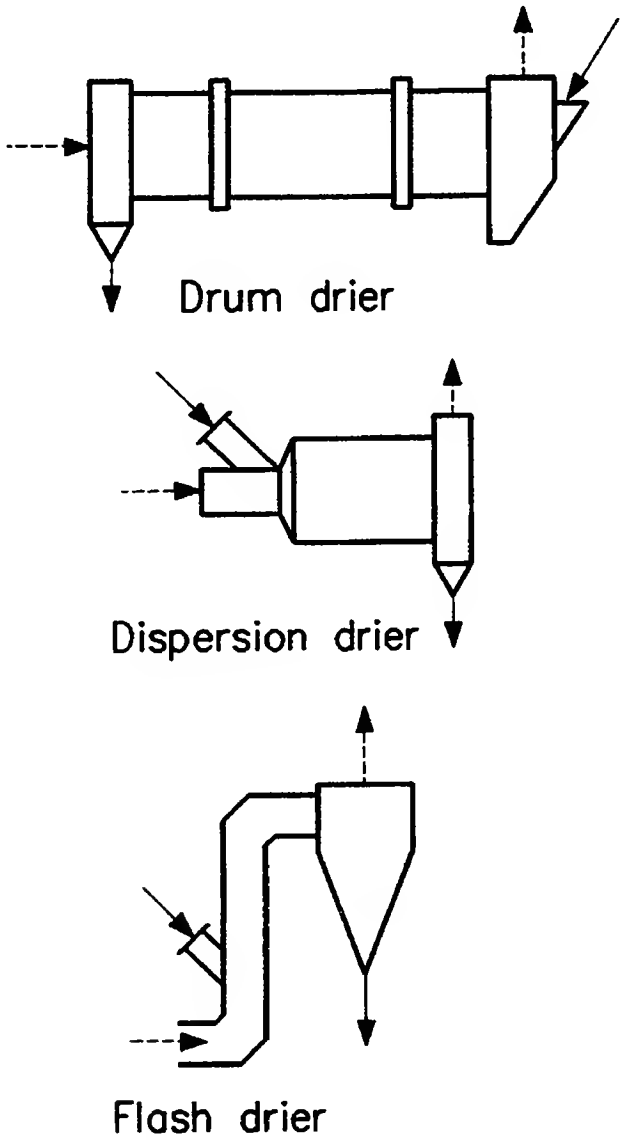

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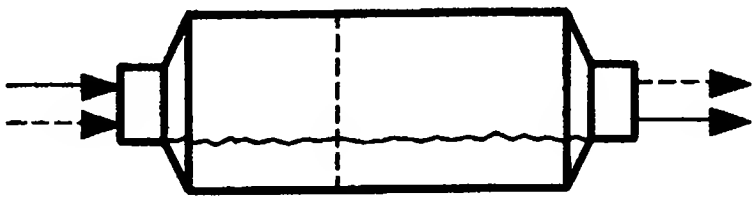
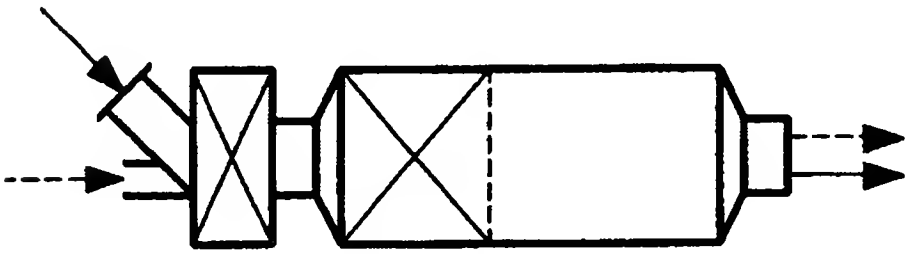
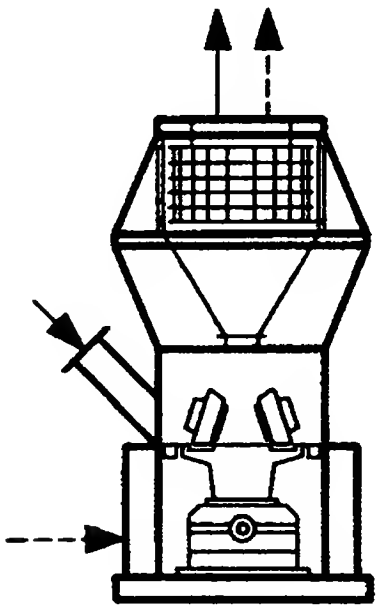
Figure 5b Drying Possibilities - Drying in Mill



in Grinding compartment



in Drying compartment
(integrated/overhung)



in Roller mill

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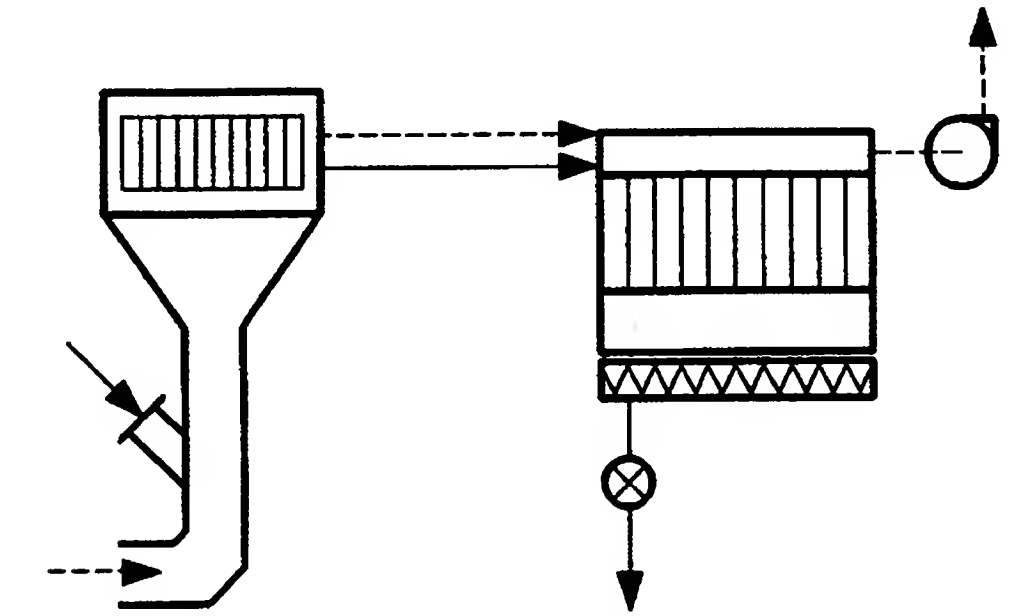
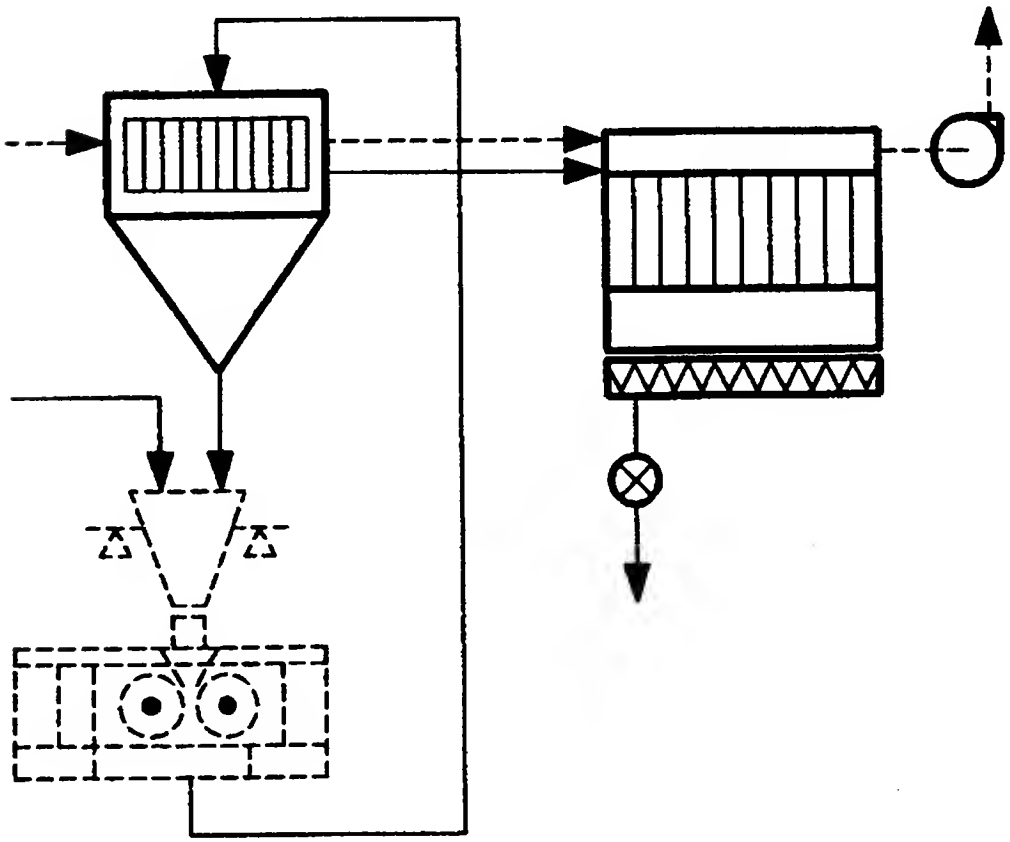
Figure 5c Drying Possibilities - Drying in Separator



with Raiser duct



without Raiser duct

5. RAW GRINDING SYSTEMS (DRY GRINDING)

5.1 Raw Grinding System with End-Discharge Mill (Figures 6, 7)

This mill is used for grinding raw materials with low moisture contents or predried materials. The drying capacity is limited by the admissible gas speed through the mill. The drying compartment is either incorporated into the mill shell or as drum installed ahead of the inlet trunnion. The drying gases or the mill ventilating air passes generally through a static separator (4) and is dedusted in a dust collector (5).

The system utilizes a mechanical conveyor (bucket elevator) to lift the mill discharge product to the dynamic separator (3). The mechanical transport is very economical with a minimum of electrical energy consumption.

In some cases, raw material with low moisture content is even dried within the grinding compartment. Moisture contents of 1 - 1.5 [%] can be dried without hot gases. Moisture content s up to 5 [%] can be dried with hot gases. The critical point is to avoid clogging of the intermediate diaphragm with too moist a material.

When drying in the grinding compartment, special care has to be paid to avoid overheating of the grinding media and intermediate diaphragm.

Sufficient cooling and/or insulation of the bearing of tube mills with trunnion bearings has to be provided.

The tube mill may be fitted with one or two grinding compartments, according to the material hardness and feed size.

The mill system is of a conventional, simple and easy operable set-up.

5.1.1 Features

- Low drying capacity (limited by allowable gas speed within the mill < 2 [m/s] over ball charge)
- With or without drying compartment
- With one or two grinding compartments (according to material hardness and consequently adaptation of ball sizes)
- Conventional, simple mill

Figure 6 End Discharge Mill

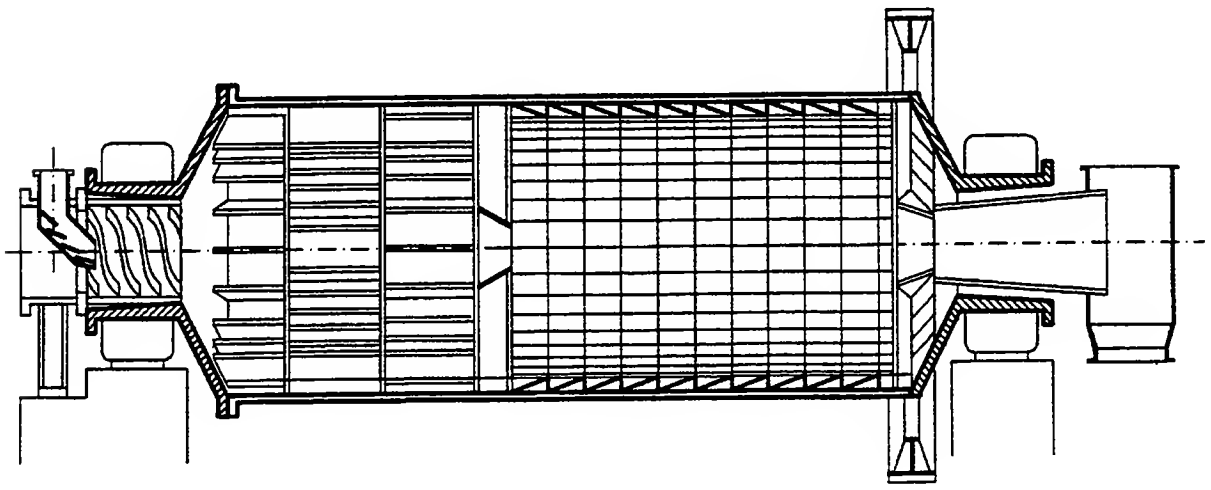


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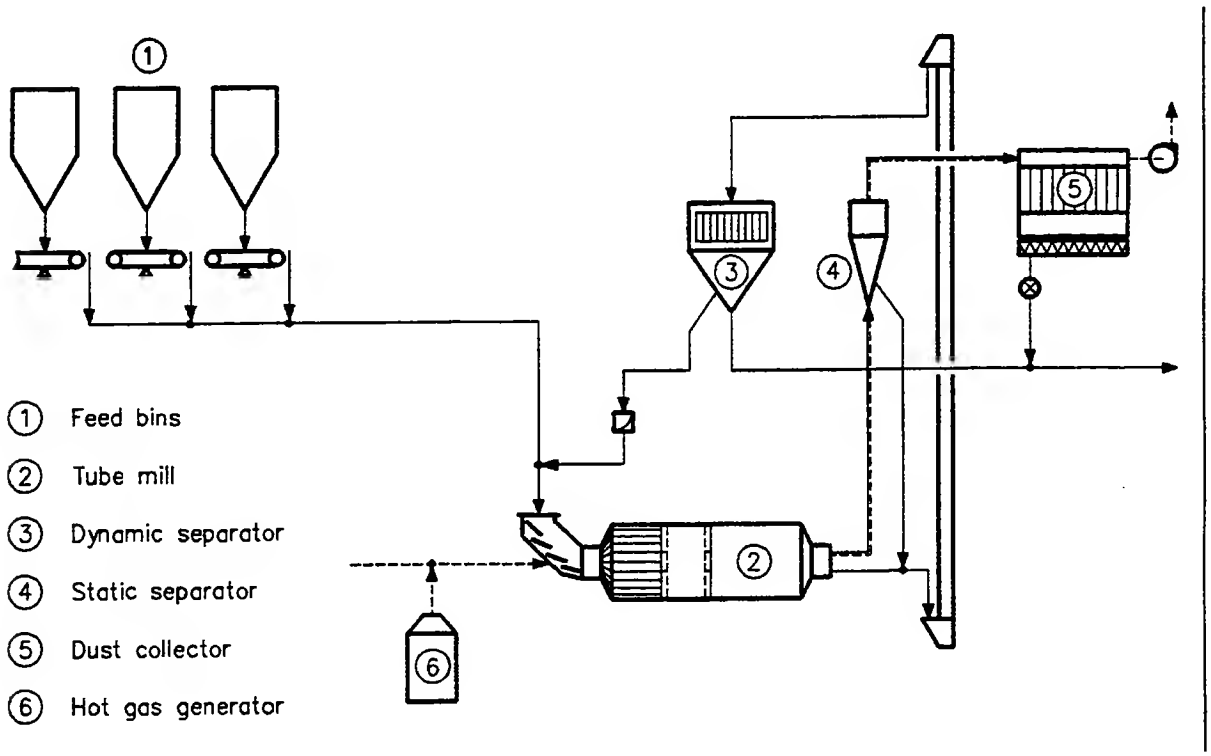
Figure 7 Raw Grinding System with End Discharge Mill



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5.2 Raw Grinding System with Center Discharge Mill (Figures 8, 9)

The drying is done in an incorporated drying compartment ahead of the first grinding compartment. For higher moisture contents, also a second drying compartment (drum) ahead of the inlet trunnion is often installed. The drying gases pass the drying and the first grinding compartment before leaving the mill through the central openings around the mill shell. If needed, hot gases can be drawn through the second grinding compartment for a complementary drying or fresh air can be introduced for a normal mill ventilation. The dust in the mill exhaust gases is separated in a static separator (4). The tailings are sent to the dynamic separator whereas the final dedusting of the gases is done in the mill/kiln gas dust collector (5).

The separator tailings from the dynamic separator are fed to the second grinding compartment for fine grinding. Usually approx. 30 [%] of the tailings return to the first grinding compartment.

The center discharge mill is applied for hard grindable raw materials or materials with components of significant different grindabilities. The advantage is, that particles, which are already sufficiently ground after the first compartment, can be classified by the separator and only the harder particles pass through the second compartment.

False air rates are seldom below [%] for mill and mill fan.

As the mills are preferred for harder materials, generally the specific energy consumption of center discharge mill system is higher than for other tube mill systems.

The experience shows, that the center discharge mill system, due to its complexity, is quite difficult to handle and to operate.

5.2.1 Features

- Medium drying capacity up to 15 [%] H_20 with a heat source additional to the kiln exhaust gases
- Suitable for materials of high hardness, and materials with hard and soft components
- Sensitive to false air leakage (sealing around entire mill shell)
- Complex process (difficult distribution of gases and tailings)

Figure 8 Center Discharge Mill

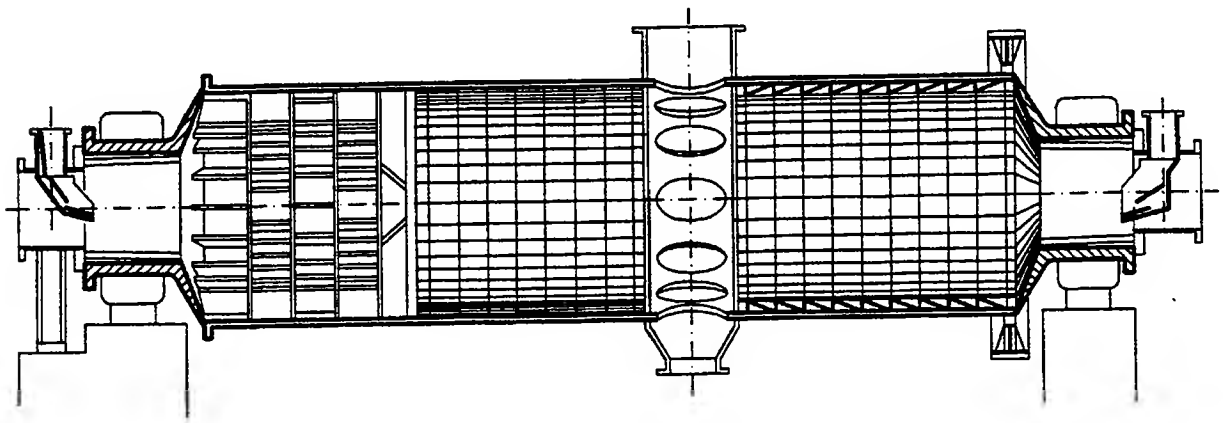

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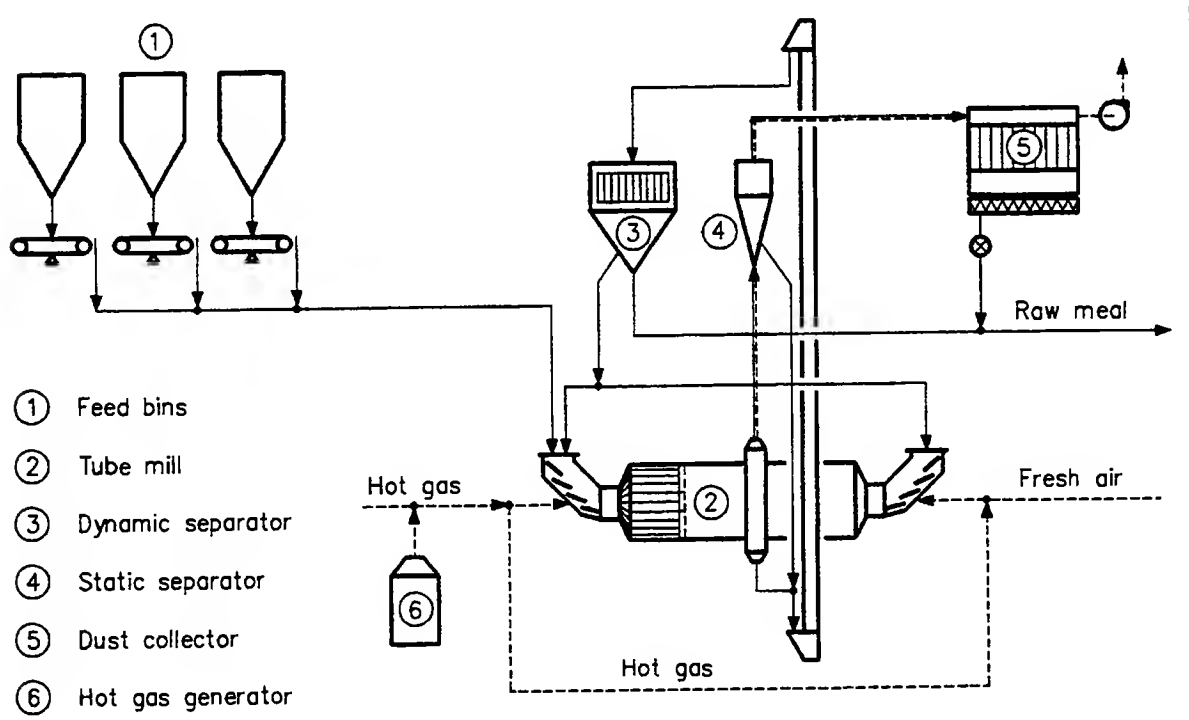
Figure 9 Raw Grinding System with Center Discharge Mill



5.3 Raw Grinding System with Air Swept Mill (Figures 10, 11)

Drying, grinding and transport are close connected in an air swept mill system. The drying is done in an incorporated drying compartment, the grinding in a subsequent single grinding compartment.

The single grinding compartment requires a corresponding pattern of liner plates, as the materials have to be ground from feed size down to the final fineness in this compartment. Often lifting or combi liners are used over the first few meters of the compartment and classifying or combi liners over the rest of the compartment.

One of the weak points of the air swept mill is the diaphragm between drying and grinding compartment, which has to provide low pressure drop for the gas flow and a good passage for the feed materials.

The drying gases discharge the ground materials from the grinding compartment and convey the materials through a static or dynamic separator (3) to a dust collection facility (4) (5). The grits from the separator return to the mill inlet.

One part of the mill exhaust gases are recirculated form the mill fan outlet to the mill discharge to ensure a sufficient gas speed for the material transport. An other part of the gases are recirculated to the mill inlet to maintain a proper gas speed and constant conditions within the mill. Excessive false air presents often a capacity problem of the mill fan and makes a proper control difficult. False air rates of more than 25 [%] over mill and fan are common.

The close connection between drying and transport with the hot gases make the system quite difficult for a proper adjustment and optimization.

The pneumatical transport leads to higher electrical energy consumption for the transport part.

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5.3.1 Features

- Medium drying capacity (limited by max. gas speed within the mill < 6 [m/s])
- Low to medium hardness of materials (single grinding compartment)
- Small feed size to mill required (< 30 [mm])
- Difficult adjustment of the ball charge distribution
- Higher gas flow rates cause higher circulating loads

Figure 10 Air Swept Mill

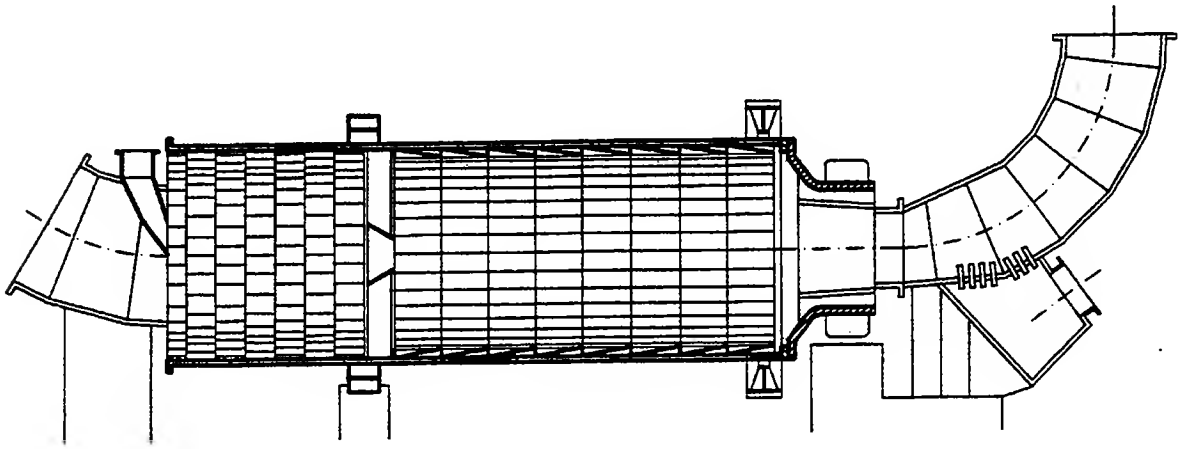
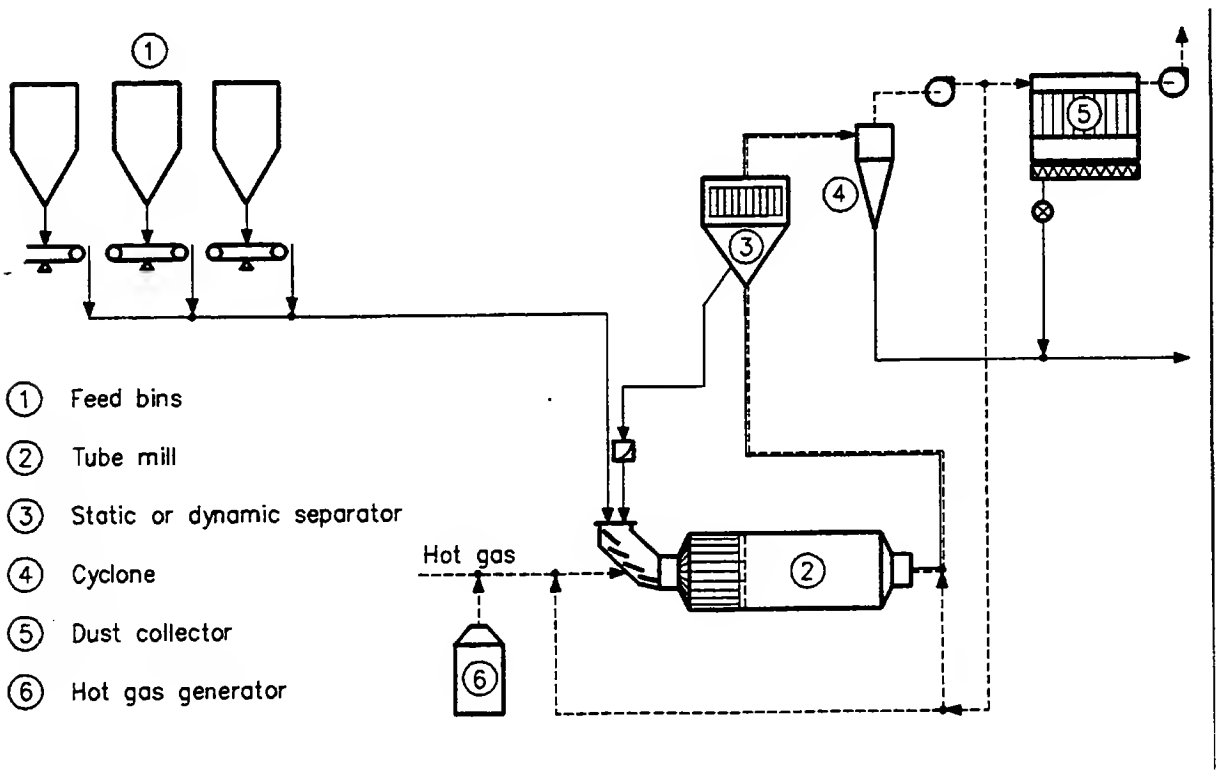


Figure 11 Raw Grinding System with Air Swept Mill



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5.4 Raw Grinding System with Vertical Roller Mill (Figures 12, 13)

Drying, grinding and separating are done in one single machine.

The drying gases are used to dry, to provide the internal material circulation, the transport to and from the separator as well as acting as separating air. The system can only be kept in equilibrium and in stable conditions by controlling and maintaining a constant gas flow through the roller mill. Normally recirculating gases are used to maintain the gas flow constant and a negative pressure at the mill inlet. The drop of the gas pressure across the mill is usually kept constant by adjustment of mill feed rate.

Some systems are setup with cyclones and three fans, whereas others operate only with a filter (3) and two fans. The duct for the gas recirculation and the duct for direct operation of kiln gases to the filter should always be separate to avoid operating problems.

This roller mill system works with high negative gas pressures and needs therefore special care of the sealings to avoid excess false air intake. False air rates of more than 20 [%] over mill and fan are common.

The external material circulation with a bucket elevator (4) is used to recirculate coarse material, which falls through the nozzle ring back to the mill inlet. The bucket elevator can easily be used with a surge hopper in connection with a metal detector by-pass or when emptying the mill for maintenance purpose.

For low moisture the mill is sized according to the grinding capacity, and for high moisture according to the necessary gas flow rate (casing diameter).

5.4.1 Features

- High drying capacities, up to 20 [%] with an additional hot gas generator
- Max. mill inlet temperature of < 450 [C] admitted
- Efficient grinding process in a compact machine
- More specialized maintenance staff required
- Coarse material feed size possible < 100 [mm]
- Operation at partial load 70 - 100 [%] possible
- Large grinding capacities up to approx. 550 [t/h] in one machine

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Figure 12 Vertical Roller Mill

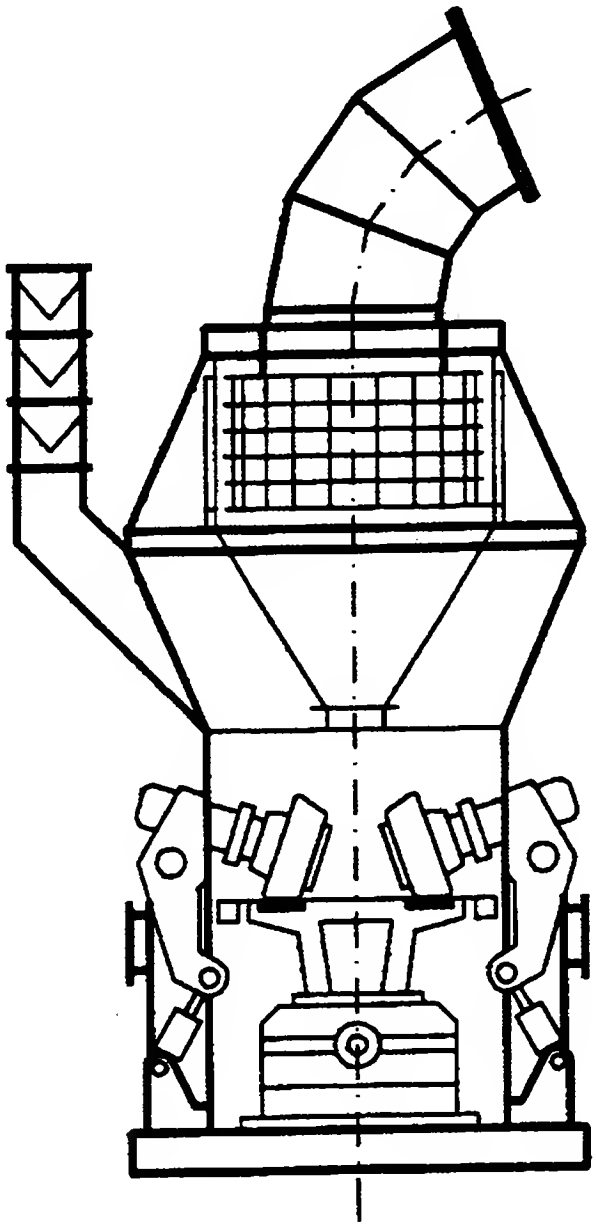

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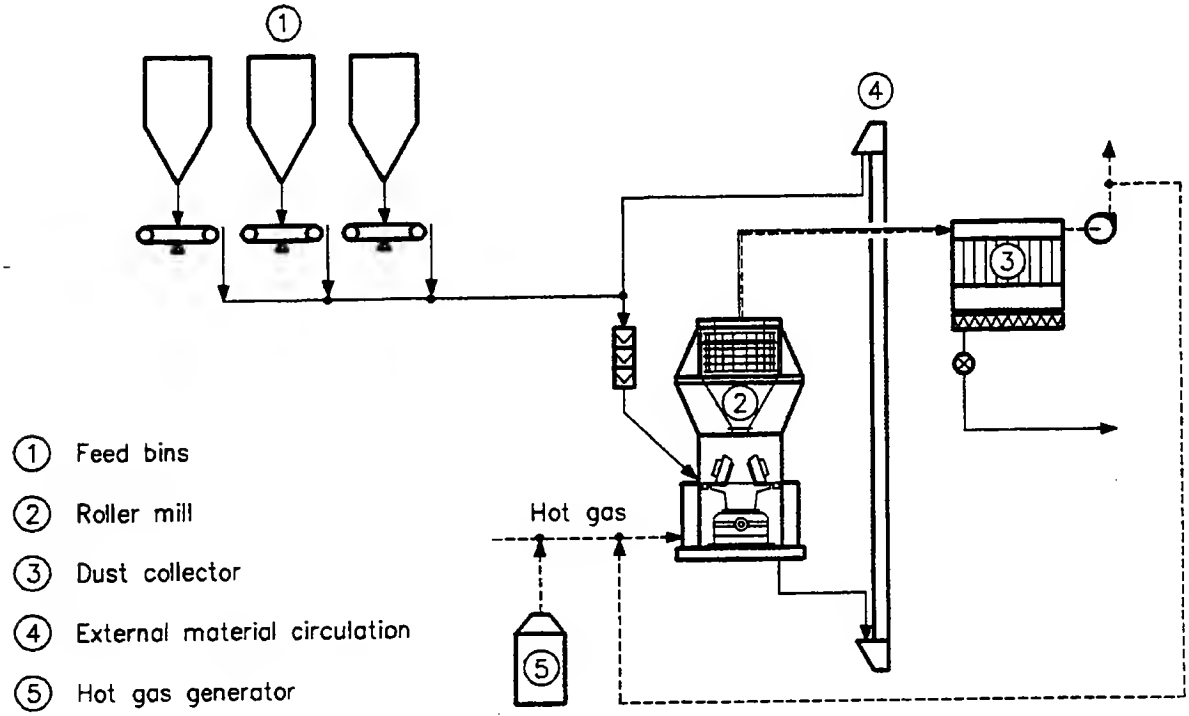
Figure 13 Raw Grinding System with Vertical Roller Mill



5.5 Raw Grinding System with Roller Press (Figures 14, 15)

The materials are proportioned to a press feed hopper and are dosed to the gap between the rolls of the press (2). The discharge material from the press consists of slabs, a type of compacted flakes. The slabs contain a portion of fine material which has to be broken up in a desagglomerator (3) before it can be fed to the separator (4). The fines from the separator are collected in a dust collector (5) and leave the system as raw meal. The separator tailings are sent back to the roller press.

The entire drying takes place in the separator where hot gases are utilized for drying and separating. The maximum hot gas temperature at the separator inlet depends on the placement of the rotor bearing and its maximum admissible operating temperature and will generally be around 250 - 300 [C].

For higher drying capacities a single pass separator (4) (5), where up to 100 [%] of the separating air can consist of hot gases, may be used. For lower drying needs a cyclone air separator (4a) with a steady use of high amounts of recirculating gases is recommended.

A possibility for drying higher moisture contents is the use of a vertical duct under the separator, which acts as a flash drier (figure 5). In this case, the bucket elevator is replaced by the raiser duct, feeding the separator with the material laden flow of hot gas from below. Fine and moist correctives can be fed directly to the raiser duct.

Material moisture is limited to 3 - 5 [%] H_20, above this, the grinding efficiency is reduced and a predrying facility will be necessary. The drying will be completed in the separator.

Fine and dry feed components call for a reduction of the roller speed for vibration free operation, resulting in lower production rates.

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5.5.1 Features

- Suited for low moisture of feed material or even predried materials
- Moist components reduce the grinding efficiency and increase the wear rates of the rolls surface
- Feed components with high quartz contents also provoke high wear rates, which may cause excessive stops for rolls surface recharching
- Moist feed components may present flow problems in the press hopper
- Feed size limited to < 50 [mm] (draw-in action to roller gap!)
- Size of a single machine system up to 200 [t/h]

Figure 14 Roller Press

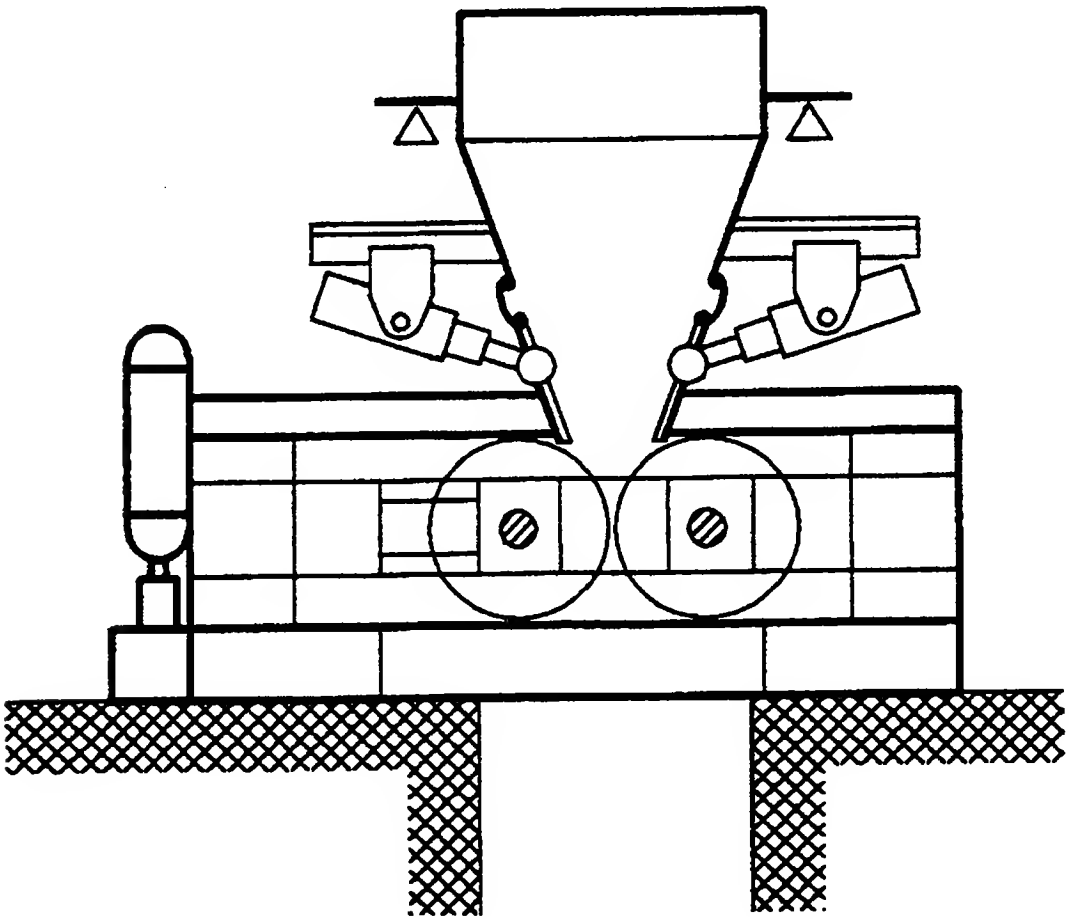

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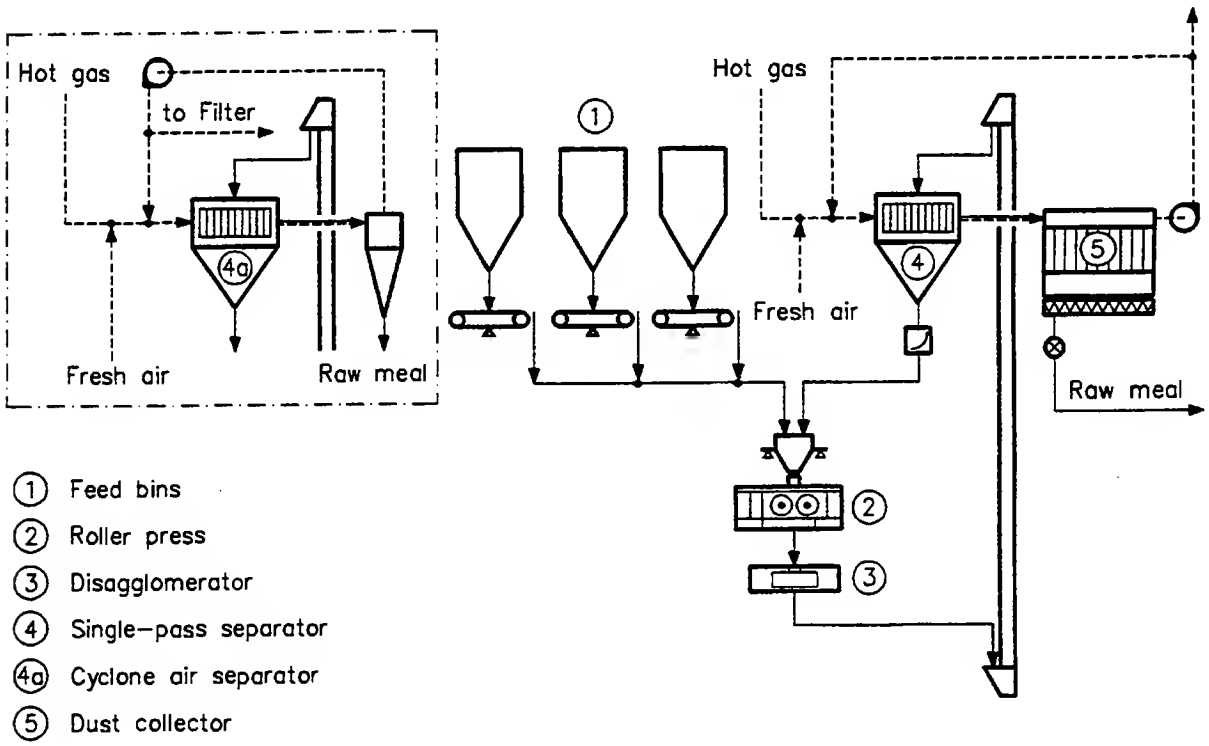
Figure 15 Raw Grinding System with Roller Press



6. WET GRINDING (FIGURES 16, 17)

The wet grinding is an older process which, today and in the future will no more be installed, provided very special conditions are prevailing.

Typical mills for wet grinding are tube mills with balls or rods.

The raw material is fed through a spout feeder to the tube mill with addition of water. The product leaving the mill is a slurry which is pumped (3) to a slurry basin (5). The slurry is further conveyed to a wet kiln or treated by a filter press. In case of a filter press, the filter cakes are either granulated and fed onto a traveling grate or dried in a drier before being fed to a semi-wet process kiln.

Slurries have typical moisture contents of 28 + 42 [%] H_20.

A typical water content of cakes after the filter press is approx. 15 - 18 [%] H_20.

Due to the high water content in the mill, high wear rates (mainly corrosion) at liners and grinding media are resulting. The mills must frequently be recharged with grinding media.

The slurry mills are either operated as open circuit or as closed circuit mill systems. In the latter case, DSM screens (4), centrifugal or vibrating screens, or hydrocyclones are utilized for separation of the fine and the coarse fraction.

The electrical energy consumption of a slurry tube mill is quite low as the water addition to the mill favors the grinding. However more thermal energy is necessary for drying of the high water contents of the slurry.

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6.1.1 Features

- Suited for wet raw materials, escavated from under water
- Low electrical grinding energy needed
- Heavy wear of mill internals and pumps
- Today only used in special cases

Figure 16 Slurry Mill

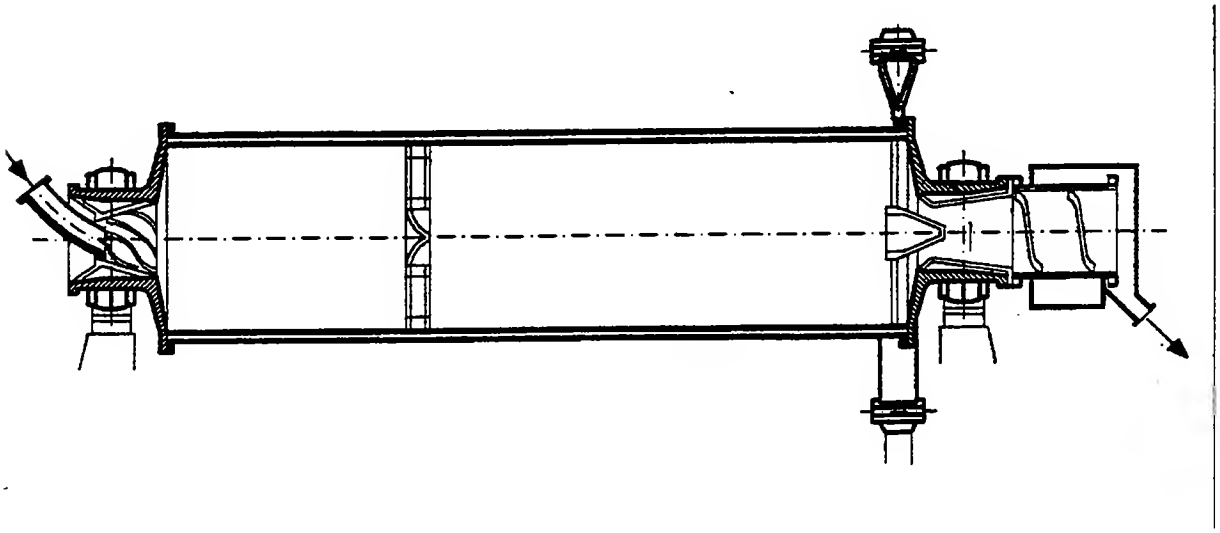
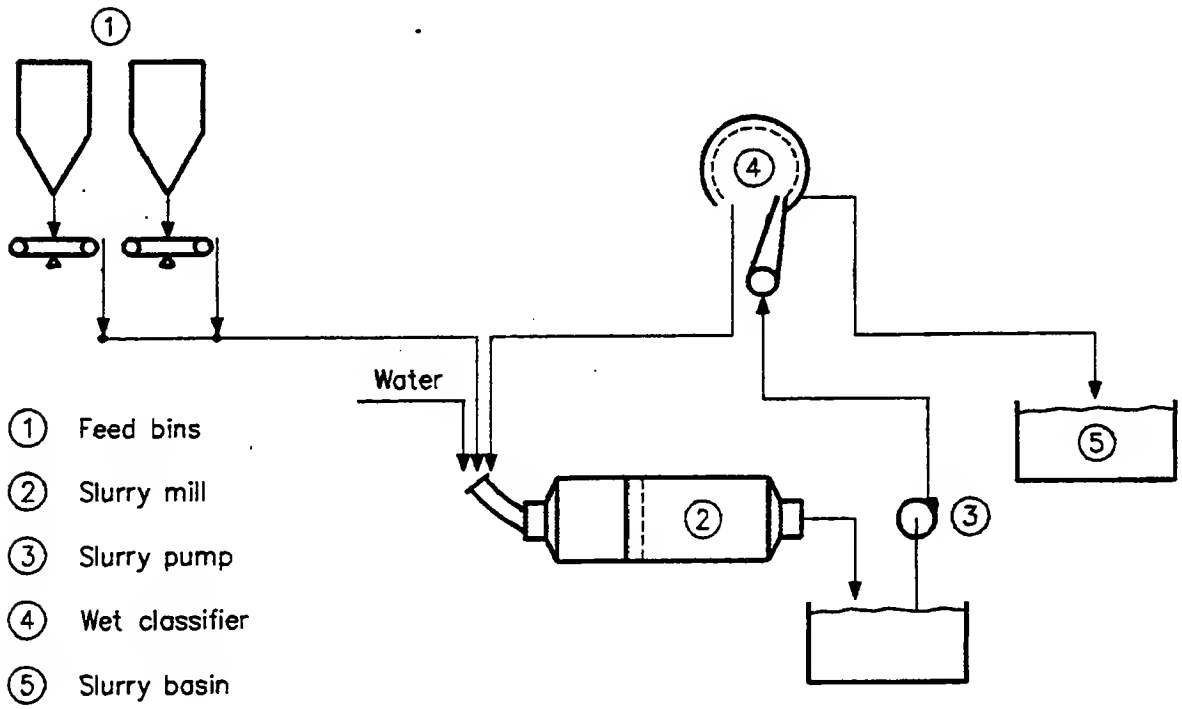


Figure 17 Wet Grinding System with Wet Classifier



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7. COMPARISON OF RAW GRINDING SYSTEMS

7.1 Performance (Figure 18)

- **Moisture content:** The moisture content of raw material is the predominant criteria for the performance of a raw grinding/drying system. The drying capacity is limited by the max. allowable gas speed within the drier/mill, the volume of drying compartment and the available amount and temperature of hot gases.
- **Material feed size**: Materials fed to a specific grinding facility have to be crushed to the suitable max. particle size. In tube mills, the max. ball diameter can be adapted in a very small range to cope better with a specific feed size.
- **Grinding capacity**: The grinding capacity of a tube mill system increases with smaller material feed size. The finer the raw meal, the lower the production rate.
- **Stickiness**: Sticky components are difficult to handle and have to be put into a drying compartment before grinding (roller press is not suitable).
- **Abrasivity:** Suitable equipments are those which have easy replaceable and cheap wear parts (tube mills).

7.2 Energy Consumption (Figures 19, 20)

- **Grindability**: Raw material grindabilities vary in wide range according to the specific properties of the components. A comparison between various raw grinding systems can only be done on basis of grindability tests.

  Among the tube mill systems, the slurry mills are generally the lowest (due to the water addition) and the center discharge mills the highest (suitable system for hard materials) in electrical energy consumption.
- **Specific energy consumption**: Generally can be said, that the lowest specific energy consumption can be achieved with roller mill and roller press systems based on the more efficient grinding principle.
- The experience shows, that three fan systems have about the same specific energy consumption as two fan systems.

7.3 Grinding Installation

- **Complexity**: Complex systems like center discharge and air swept mill systems are often difficult to optimize, as there are many interactions between various parameters. The same applies to systems with integrated pregrinding/predrying facilities.
- **Extensions:** Often the grinding capacity of an existing tube mill system can be increased by adding a pregrinder, but the drying capacity can only be increased with higher efforts in costs and considerable space requirements.

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Figure 18 Performance of Raw Grinding Systems

FIG.18 PERFORMANCE OF RAW GRINDING SYSTEMS

Type of Mill	Drying Capacity [%] H2O (5 / 10 / 15 / 20)	Max. Mill Feed Size [mm]	Max. Mill Gas Speed [m/s]	Max. Mill Capacities [t/h]	Energy Consumption	Stickyness of Materials	Maintenance
End-Discharge Mill		25-30	2	350	medium	no	simple
Center-Discharge Mill		30-50	2	350	high	suited	simple
Air-Swept Mill		25-30	6	350	medium	suited	simple
Vertical Roller Mill		100		550	low	suited	specialized
Roller Press		50		200	low	no	specialized
Hammer/Impact Mill		200		100	medium	no	medium
Autogenous Mill		300		400	high	very suited	simple

Legend:
- without drying compartment
- with gases of max. 350 [C] (kiln exhaust gases)
- with gases of more than 350 [C] (hot gas generator)

--> Higher drying capacities can be achieved with predrying facilities (e.g. flash drier etc.)

Raw Grinding Systems

"HOLDERBANK" Process Technology Division

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Figure 19 Spec. Energy Tube Mills

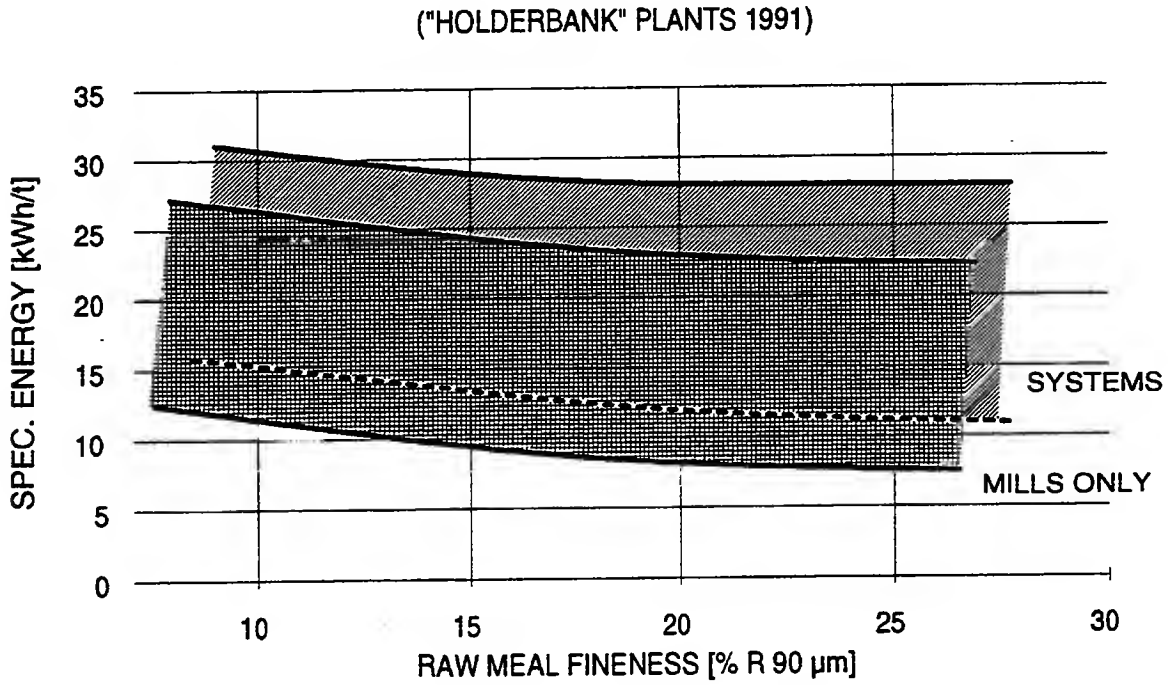

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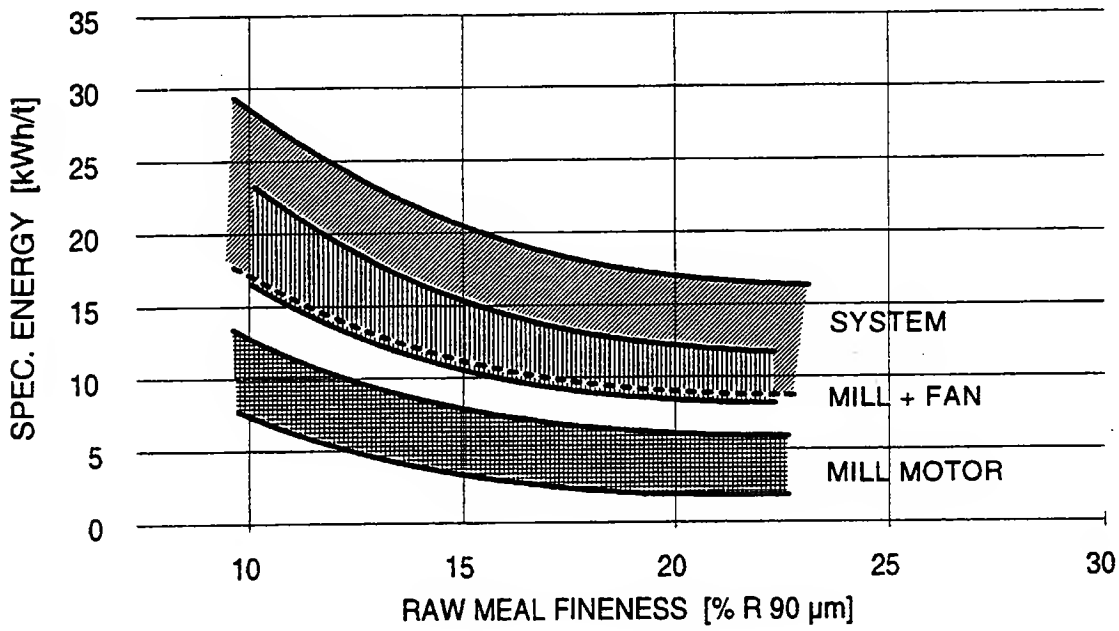
Figure 20 Spec. Energy Roller Mills



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Chapter 11

Drying Technology

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Drying Technology

H.P. Fisch
PT 96/14027/E

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1. BASICS OF DRYING

Drying means in general the removal of water from a material. This might be a mechanical or a thermal "dewatering".

Materials to be dried in the cement industry are normally thermally dewatered, that means the removal of moisture by evaporation under utilization of various types of hot gases.

Types of moisture are:

- Surface moisture
  water film on particle surface causing agglomerations of fine particles
- **Outer Moisture** (provokes handling problems)
  - cavity moisture
    droplets captured in voids between fine particles
  - capillary moisture
    water drawn in thin gaps between surfaces leads to adhesion and agglomerations of particles
- **Inner moisture** (no influence on handling)
  - pore moisturewater
    contained in pores
  - inherent moisturenatural moisture
    inhibited in solid materials

The drying process is mainly influenced by the

- characteristics of material
- percentage of moisture
- type of moisture
- particle size of material
- time of exposure to heat
- drying system

When drying a material, first the surface then the outer and at last the inner moisture is evaporated due to the heat transfer and moisture diffusion. The outer and surface moisture can evaporate easily. The inner moisture has to diffuse first to the particle surface, from where it can be evaporated. This diffusion of moisture is facilitated by heating up the particle.

The water evaporation rate of a moist material increases with higher hot gas temperature and is different for each material.

The residence time during which a particle is exposed to drying gases as well as the particle sizes are of utmost importance for a sufficient drying.

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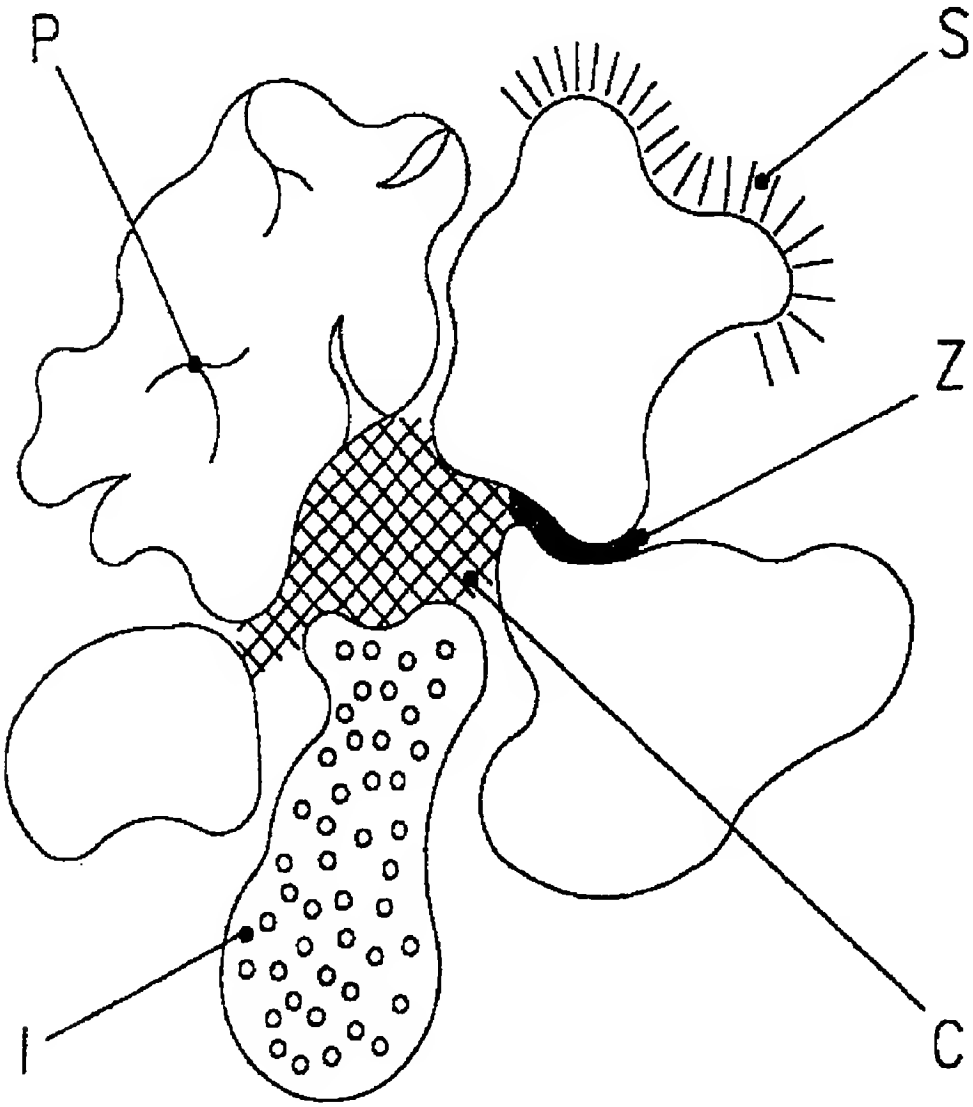
Figure 1: Moisture and drying speed



C cavity moisture
Z capillary moisture
S surface moisture
P pore moisture
I inherent moisture

poor handling

good handling

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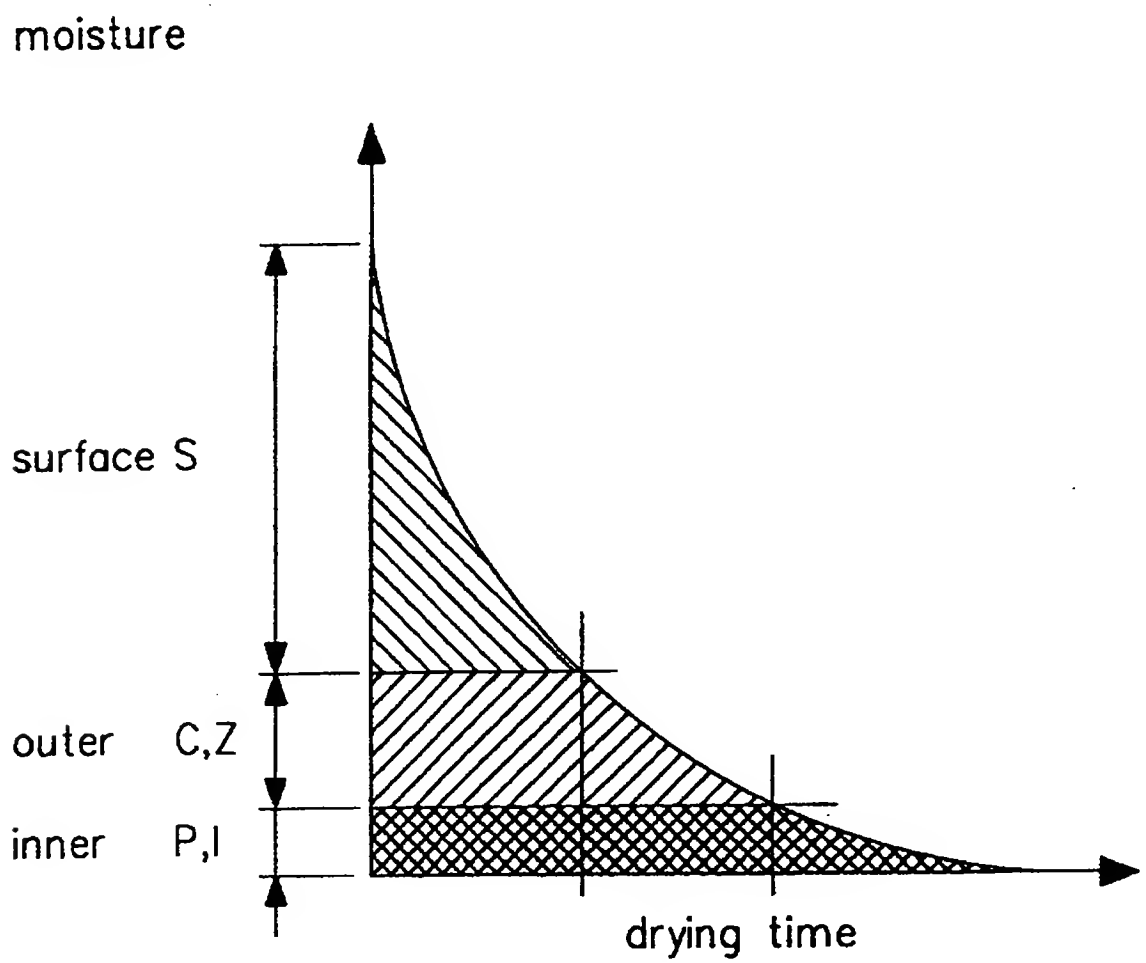

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2. ANALYSIS AND CALCULATION

Remember: The values of moisture content are always indicated on wet basis and weight percent. This means that the weight of the wet material is always 100 %. This rule corresponds to the analysing method in the laboratory.

In thermal drying processes, the heat is transferred mainly by convection from the gas to the material and moisture.

Major possible heat sources:

- kiln exhaust gases < 350 [°C] (depending on system)
- clinker cooler exhaust air < 350 [°C] (depending on system)
- hot gases (hot gas generator) < 1300 [°C]
- grinding heat (electrical/mechanical energy)
- clinker heat (for cement grinding)

The necessary energy of a drying gas is expressed by

$$Q = V \cdot t \cdot cp$$

Q	=	required energy	[kJ/h]
V	=	gas flow rate	[Nm^3/h]
t	=	gas temperature	[°C]
cp	=	spec. gas heat	[$kJ/Nm^3 \cdot °C$]

and has to be found through a heat balance.

The residual moisture is mainly a function of the exhaust gas temperature and the dry product temperature after the drying process. For raw material drying, the residual moisture is generally < 1 [%] H_20 and for cement close to zero.

Some types of coal are only dried to the level of the hygroscopic moisture, as the coal will reabsorb moisture to the equilibrium point. The hygroscopic moisture has no negative effect on the flowability and handling ability of the coal.

Often, drying of material is done in connection with a grinding process, so called drying/grinding, e.g. raw mill, coal mill, cement mill. General drying is done in drying equipments (dryers).

Special attention has to be paid to the dedusting equipment of drying installations. Drying exhaust temperatures have preferably to be kept about 30 [°C] above the dew point temperature of the exhaust gas to avoid condensation of the moisture in contact with ambient air or cold spots of the equipment. Sometimes filters and ducts need insulations.

Problems may arise with the handling of material which is partially dried and the dedusting equipment. Partially dried and warm material is difficult to handle and develops vapors which often clog the filterbags.

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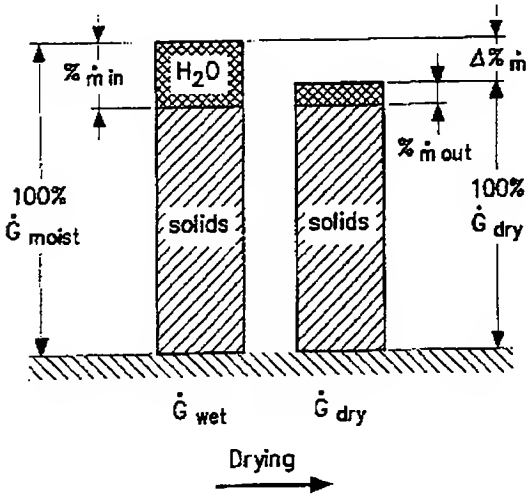
Figure 2: Analysis and calculation

Analysis : Moisture content ⟶ Wet basis (weight %)



$\dot{G}_{wet}$	moist material (feed)	(t/h)
$\dot{G}_{dry}$	dryed material (product)	(t/h)
$\%\,\dot{m}_{in}$	initial moisture	(%)
$\%\,\dot{m}_{out}$	residual moisture	(%)
$\dot{m}$	evaporated moisture	(t/h)

$$\text{Product } \dot{G}_{dry} = \dot{G}_{wet} \cdot \frac{100 - \%\,\dot{m}_{in}}{100 - \%\,\dot{m}_{out}} \quad (t/h)$$

$$\text{Evaporated moisture } \dot{m} = \dot{G}_{wet} \cdot \frac{\%\,\dot{m}_{in} - \%\,\dot{m}_{out}}{100 - \%\,\dot{m}_{out}} \quad (t/h)$$

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3. DRYERS

3.1 Drum dryers

Design

The drum or rotary dryer consists of an inclined steel tube on two tyres driven by a girth gear drive. The dryer volume is essential for the amount of water to be evaporated. Usually the L/D ratio is kept < 8 [-].

Drying

The heat exchange in a drum dryer depends mainly on:

- drum speed
- inlet gas temperature
- gas speed within the drum
- type, size and surface of internal fittings

Drum dryers are specially suited for moist and sticky material (clay components) and allow a larger material feed size than other dryers.

- Parallel flow
  Material and gas flow in parallel from the inlet to the discharge. Moist feed material is already exposed to the hot gas at the dryer inlet. This prevents from clogging the inlet and improves the material flow through the dryer.
  High hot gas inlet temperatures are possible and advantageous (high temperature difference between moist material and hot gas) for a more efficient heat exchange. Parallel flow operation is always preferred.
- Counter flow
  The hot gases enter at the drum discharge, where the material moisture is low. A main disadvantage of this principle is that the gases at already low temperatures enter in contact with the moist feed material. An increase in hot gas temperature leads mainly to unnecessary heating-up of the discharge material. On the other hand, the dust emission is relatively low. Counterflow operation is generally not recommended.

Features

♦ retention time depends on:		
• dryer rotating speed	2 - 5 [min^{-1}]	
• dryer inclination	3 - 6 [°]	
• internal fittings		
• gas speed	< 6 [m/s]	
♦ dust load	< 150 [g/m^3]	in exhaust
♦ pressure drop	2 - 5 [mbar]	accross drum
♦ exhaust gas temperature	100 - 120 [°C]	Recommende
♦ specific electrical energy consumption	~ 3 [kWh/t]	for drum, transport and fan

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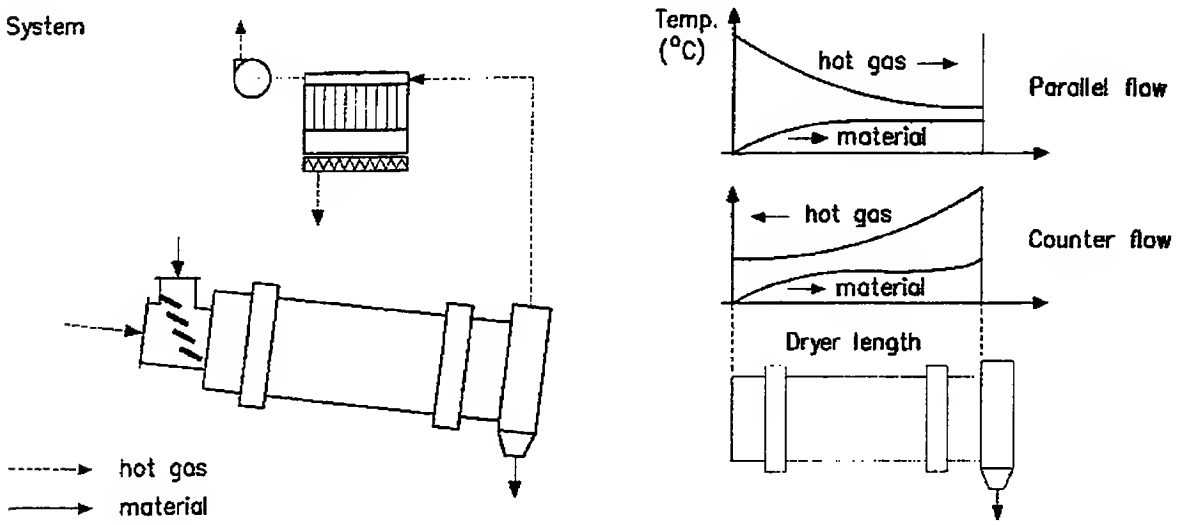
Figure 3: Drum (rotary) Dryer



Feed size	(mm)	200–300
Drying capacity	(%H_2O)	< 25
Spec. load	(kg $H_2O/m^3 \cdot h$)	< 150 depending on internal fittings
Temperature max.	(°C)	< 1200

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3.2 Dispersion Dryer

Design

Dispersion dryers are comparable with a drying compartment of a tube mill. The drums are short (L/D ~ 2.5 [-]) and contain lifter plates for the material dispersion across the entire drum length.

The inlet is generally fitted with a trunnion bearing and the support at the discharge end is combined with the central drive shaft.

A retention ring at the outlet retains the material to a certain level. As the dryer drum is horizontally without inclination, the material movement towards the outlet is strongly influenced by the material flowability and the gas speed.

Drying

Material and hot gases are introduced through the trunnion bearing into the drum. Radial openings at the outlet provide the material discharge and the exhaust gases leave the dryer through the discharge hood to a dust collecting facility.

The hot gas temperature at the inlet is limited by the admissible temperature for the trunnion bearing.

The lifter plates disperse the material over the dryer cross section for a good contact with the hot gases.

The evaporation rate depends very much on:

- gas inlet temperature
- material dispersion
- material granulometry
- retention time

Features

- rotating speed ~70 [%] or critical speed, which is almost the same as for a tube mills
- higher material filling degrees than drum dryers, up to 15 [%] according to the retention ring at the outlet

♦ gas speeds	< 3 - 4 [m/s]	in drum
♦ dust load	< 300 [g/m^3]	exhaust gas
♦ pressure loss	< 30 [mbar]	depends also on internal fittings, inlet, outlet arrangement
♦ spec. energy consumption	< 1 [kWh/t]	drum
	< 3.5 [kWh/t]	system

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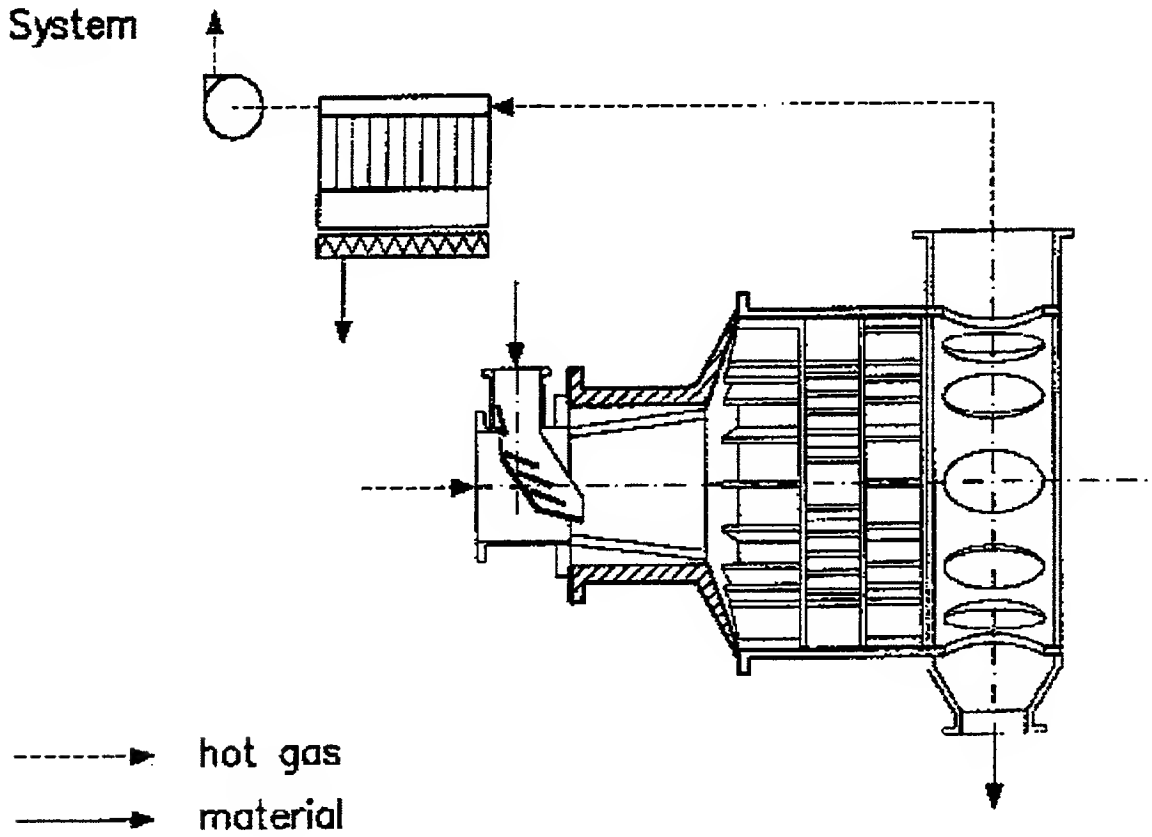
Figure 4: Dispersion Dryer



Feed size	(mm)	20–30
Drying capacity	(%H_2O)	< 15
Spec. load	(kg $H_2O/m^3 \cdot h$)	< 200
Temperature max.	(°C)	< 850

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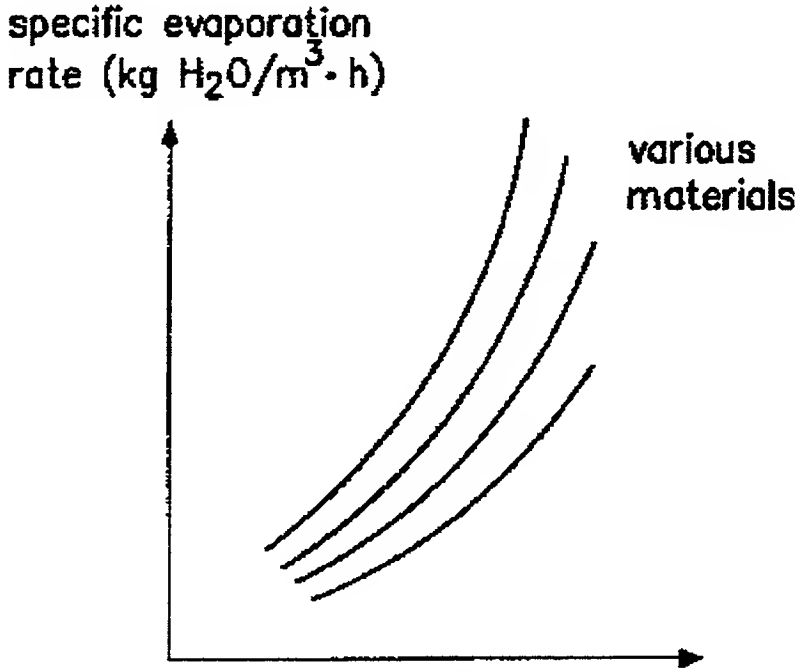

Influence of gas temperature on evaporation rate

3.3 Rapid Dryer

Design

The rapid dryer is stationary and consists of an air-tight closed casing. The lower part is fitted with one or two rotating shafts which are fitted with paddles. The paddles throw the material up and disperse it in the gas stream.

The material transport is based on the paddle action and the gas flow through the dryer.

As the casing is stationary, it can very well be sealed with an inlet and outlet airlock.

The casing inside is lined with wear resistant plates for abrasive materials.

Drying

Rapid dryers are operated in parallel flow mode. High hot gas temperature may be used if necessary, which leads to sufficient casing wall temperature to prevent build-ups in case of moist feed components.

The mechanically forced dispersion of material in the gas stream enables an intense heat transfer efficiency and therefore a quick water evaporation.

Features

- air speeds in dryer < 1.0 [m/s]
- suppliers indicate that slurries with extreme high moisture content can easily be dried (attention to dew point and filter operation!)
- one-shaft dryers are available up to an evaporation rate of 2'500 [kg H_2O/h]
- the largest dryers available can dry up to 25'000 [kg H_2O/h]
- this dryer occupies little space
- special attention has to be paid to wear on the paddles when processing abrasive materials (slag etc.)
- the energy consumption of rapid dryers is in general higher than for drum dryers due to the mechanical dispersing by the paddles.

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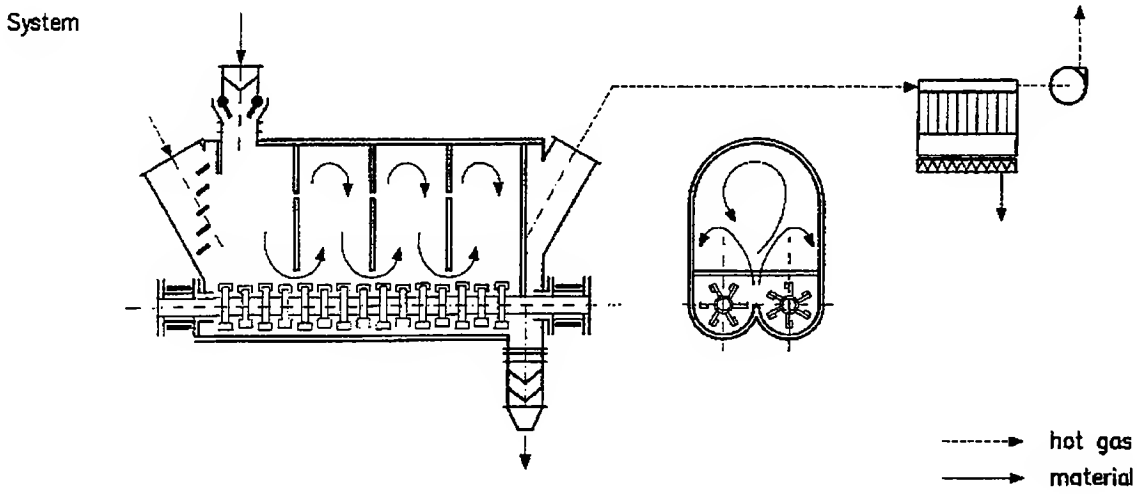
Figure 5: Rapid Dryer



Feed size	(mm)	< 500 soft material
Drying capacity	(%H_2O)	< 20
Spec. load	(kg $H_2O/m^3 \cdot h$)	< 200
Temperature max.	(°C)	< 800

3.4 Dryer-Crusher

Design

As the name reveals, the dryer-crusher is suited for the comminution and the simultaneous drying of soft and low abrasive materials. Moist feed materials pass through an impact or a hammer rotor and are most of the time extracted by the proper drying gases. The discharged, ground or partly ground and dried material may pass a classifier (static or dynamic) for separation of already fine enough particles. The return material from the separator reenters the crusher, the fines are swept to a dust collector.

Dryer-crushers are used in many different arrangements and set-ups according to the needs and possibilities.

Drying

The drying process takes place in the crusher and during the subsequent air transport. The comminution of the feed material improves tremendously the drying effect, as smaller particles provide much better evaporation of moisture.

The hot gases through the crusher rotor avoid clogging and blockage of the casing. The hot gas temperature is limited by the heat based stresses and dilatation of the rotor parts.

Crushers with gravity discharge have a much lower drying efficiency than the air-swept types. The effect is a result of smaller dryer volume and thus lower retention time.

Features

- circumferential rotor speed 25 - 35 [m/s]
- suited for partial predrying ahead of a mill
- single and double rotor crushers available
- spec. energy consumption of crusher depends on:
  - material hardness
  - particle size reduction ratio
  - produced fineness
  - feed moisture

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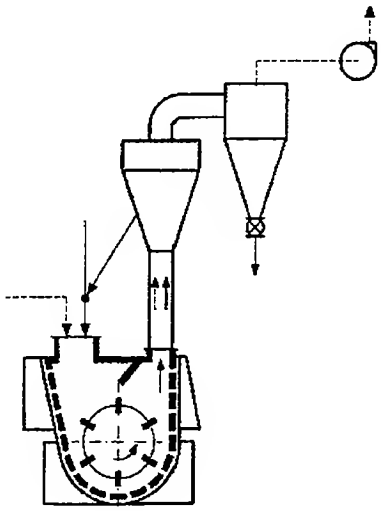
Figure 6: Dryer-crusher (with separator)

System



hot gas
material

Feed size	(mm)	< 150	(depends on crusher size)
Drying capacity	(%H_2O)	< 25	(air-swept with flash dryer)
Spec. load	(kg $H_2O/m^3 \cdot h$)	< 500	(air-swept with flash dryer)
Temperature max.	(°C)	< 800	

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3.5 Flash dryer

Design

Flash dryers have a wide field of applications. They usually are used ahead or in connection with an additional equipment (crusher, mill separator etc.).

Hot gases are drawn through a vertical duct with a fan, while moist material is fed to the upward flowing gas stream. Often a disagglomerator is installed ahead of the flash dryer to ensure a fine enough granulometry. The finer, dried material is collected in a dust collector where the large particles fall out of the dryer through an airlock.

L/D-ratios reach as far as 8 - 20 [-] according to application and fineness of material.

Drying

Drying takes place during the pneumatical transport with the drying gases. The vertical gas speed has to be superior to the terminal velocity of the particles to be lifted-up. The heat transfer from the gases to the particles and the evaporation of moisture is very efficient and intense.

The material to air ratio is limited by an acceptable maximum pressure drop over the system and the prevailing heat transfer for drying.

Features

- gas speed superior than 18 - 25 [m/s] according to the particle size
- material to air ratio < 1,8 [kg/m^3]
- typical pressure drop 35 - 45 [mbar] without dust collector
- pneumatical transport inceases the specific energy consumption

Ranges of conditions

- particles fall down below the terminal velocity
- fluidized bed zone (high pressure drop)
- critical transition zone with whirling of particles
- pneumatic transport for economical operation

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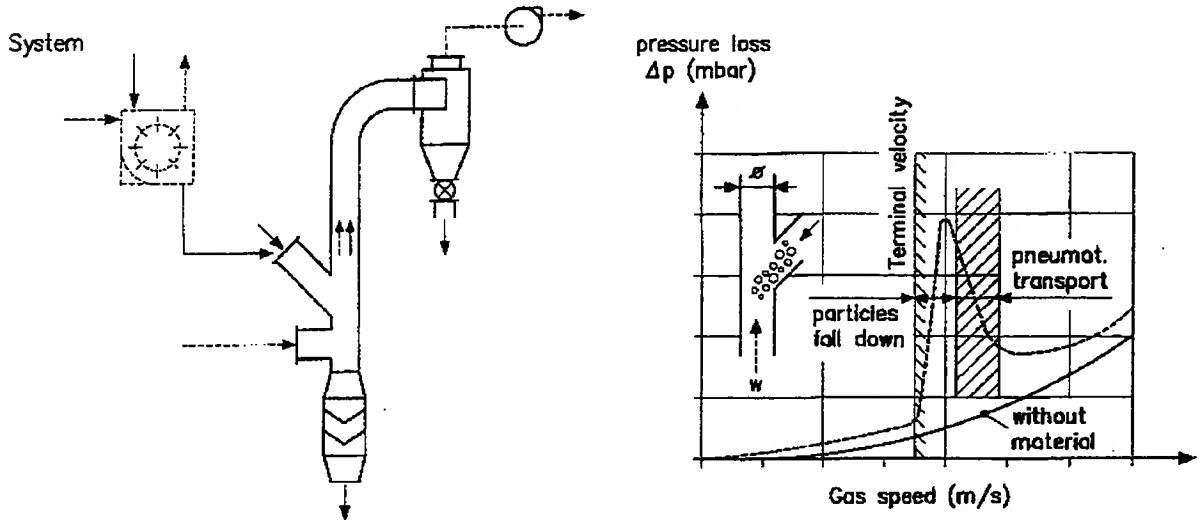
Figure 7: Flash Dryer (with disagglomerator)



Feed size	(mm)	function of terminal velocity (~ < 20 mm)
Drying capacity	($\%H_2O$)	< 35
Spec. load	(kg $H_2O/m^3 \cdot h$)	< 500
Temperature max.	(°C)	< 900

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3.6 Tube mill (without drying compartment)

Design

There are tube mills, mostly two-compartment mills, with end discharge which have no drying compartment. This is normally the case when materials with low moisture content are ground. Often cement mills, producing blended cement with wet additives, have to be adapted to a maximum drying capacity by

- utilization of a hot gas source
- modification of the inlet arrangement
- modification of intermediate diaphragm for maximum open area
- modification of mill outlet
- provide an appropriate dedusting facility

Drying

The drying must entirely take place in the first compartment. If the material is not sufficiently dry at the intermediate diaphragm

- the slot plates may be clogged or
- the material does not flow within the 2nd compartment (loss of efficiency and throughput)

Therefore, the maximum moisture content is limited by the drying ability of the 1st compartment. Beyond this limit

- a predrying facility must be utilized or
- the percentage of the wet component must be reduced or
- the mill production must be reduced.

Main heat sources are

- hot gases (hot gas generator, kiln or cooler exhaust gases)
- grinding heat generated by grinding media
- hot clinker (in case of cement grinding)

Features

- Max. gas/air speed (over ball charge) < 2 [m/s]
- Max. gas/air speed (inlet, outlet, diaphragms) < 25 [m/s]
- Fan and filter to be adapted to required gas volume and pressure drop
- Filter fabric to be selvted to operate with mois exhaust gas
- Operate filter at min. 30 [°C] above the point

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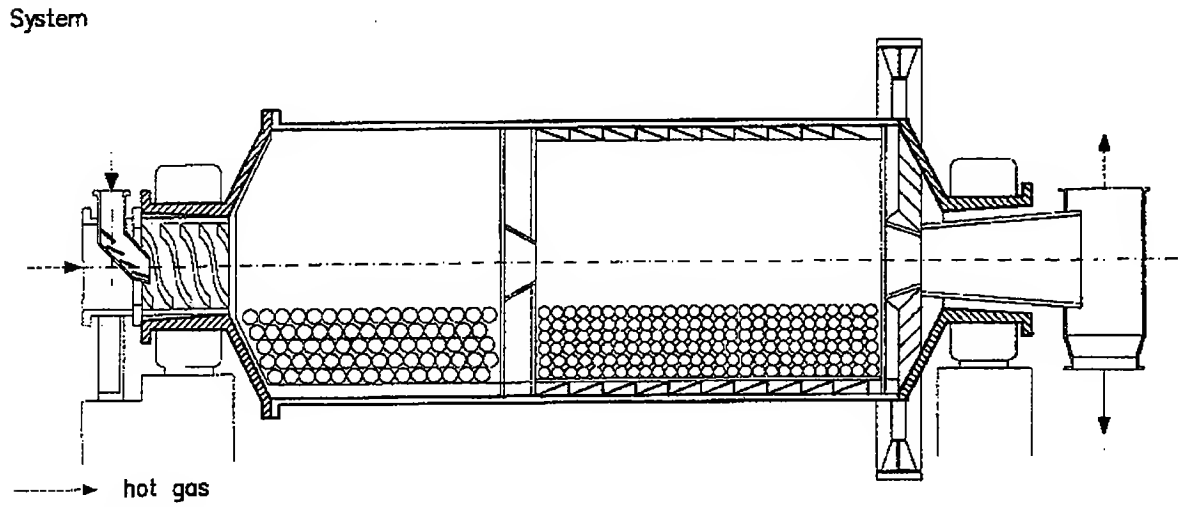
Figure 8: Tube Mill without drying compartment



Feed size	(mm)	< 30
Drying capacity	(%H_2O)	< 5
Spec. load	(kg $H_2O/m^3 \cdot h$)	< 150
Temperature max.	(°C)	< 450

3.7 Tube mills with drying compartment

Design

Typical tube mills for drying/grinding are of a design with

- end discharge
- center discharge
- air swept mills

Drying compartments may be incorporated (normal case) or of overhung type. The overhung compartments are added for a better drying (increase of dryer volume). They allow higher inlet gas temperatures as the mill bearings are placed at the outlet of these compartments.
Drying compartments have to be fitted with lifter plates for dispersion of material in the gas stream. Worn lifters reduce the dispersion effect and the drying efficiency.

Drying

The drying has to take place before the material enters into the grinding compartment. The heat source used is normally hot gas. The drying capacity depends on

- total volume of drying compartment(s)
- max. acceptable gas speed through mill
- hot gas temperatures available/acceptable
- type of mill discharge
- false air leakage

In case of air swept mills, the gas speed may be higher than in other mills and the gas flow rate to the separator must be adequate for the pneumatic transport of the ground material.

Features

Typical features are very different for all three mill types and are outlined in the paper "Raw grinding systems" VA 93/4013/E.

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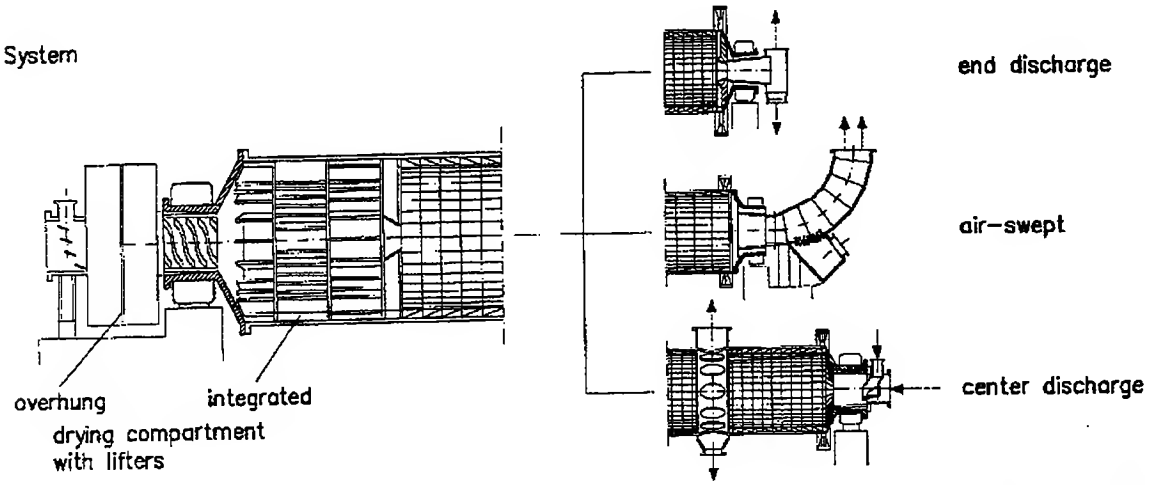
Figure 9: Tube with drying compartment



		end discharge	air-swept	center-discharge
Feed size	(mm)	25–30	25–30	30–50
Drying capacity	($\%H_2O$)	< 6	< 15	< 15
Spec. load	(kg $H_2O/m^3 \cdot h$)		< 200	
Temperature	max. (°C)		< 450	

3.8 Vertical roller mills

Design

The vertical roller mill is specially designed for grinding **and** drying.

The mill has to be dimensioned for the required grinding or the drying capacity, whichever is the critical item of the two. The mill table and the mill casing diameter have to be sized for this critical item.

The same applies to the gas flow rate, which has to be adapted either to the grinding needs (internal material circulation ≡ grindability) or to the drying needs.

For very moist and sticky materials, the inlet must be designed to avoid clogging and is preferably centralized (through the separator axis). For coal mills, often screw conveyors force the material to be fed to the grinding table.

Drying

The vertical roller mill is in principle an airswept mill. As the ground material is extracted by the drying gases, an excellent heat exchange between material and gases takes place up to the dust collecting facility.

The hot gases hit the moist material for the first time above the mill's louvre ring. Hot gases enter from the inlet gas channel and the material, first fed to the center of the grinding table, is spread over the nozzles. This first contact is the major drying action, where the gas temperature is already reduced to practically exhaust temperature.

Major material dried and ground in vertical roller mills are:

- raw materials
- coal
- slag
- pozzolana

Features

- For more technical details see also reports:
  - roller mills, VA 94/4167/E
  - raw grinding systems, VA 93/4013/E
- Recirculating gases are needed to ensure a steady gas flow rate through the mill (transport function) in case less hot gases are required for drying

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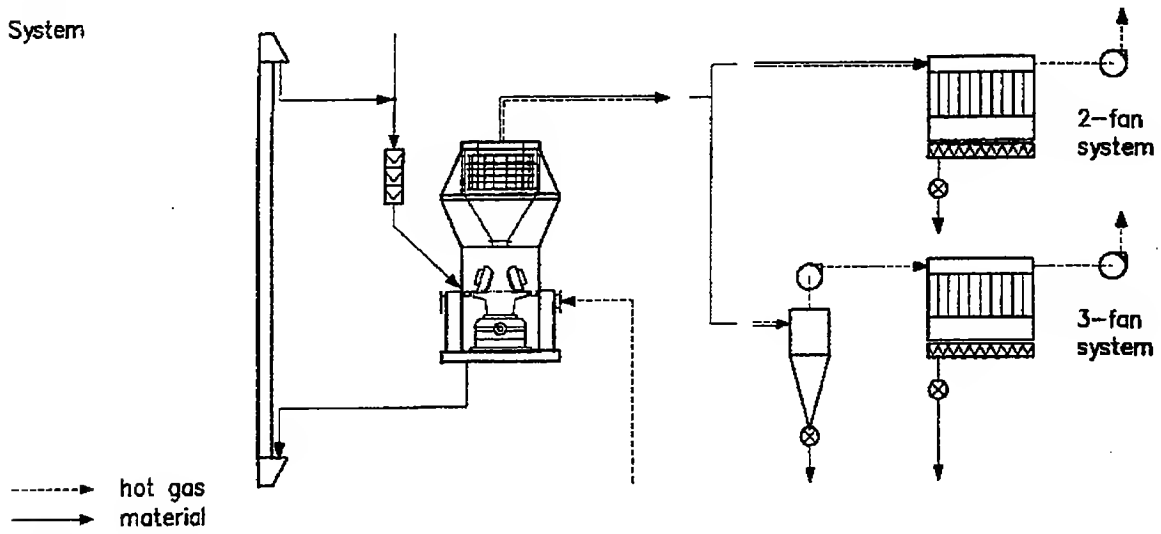
Figure 10: Drying in vertical roller mill



Feed size	(mm)	80–100 (depending on roller diameter)
Drying capacity	($\%H_2O$)	< 20
Spec. load	($kg\ H_2O/m^3 \cdot h$)	–
Temperature max.	(°C)	< 450

3.9 Separators

Design

Air separators are designed to classify partly fine or already ground particles by means of air.

Certain grinding equipments (e.g. roller press, Horomill) are not suited to dry in the mill and need therefore an additional facility. Up to a certain extent, drying in the separator is possible.

Two problems are obvious:

- these mills can handle only materials with low moisture content (efficiency)
- only materials with low moisture content can sufficiently be dispersed in the separator.

Drying

The drying capacity in the separator is very limited due to the small volume and the short retention time. This capacity can be increased by adding a flash dryer ahead of the separator to increase the dryer volume.

The maximum permissible hot gas temperature is limited by the lower separator rotor bearing to avoid the overheating of the bearing.

A gas recirculation is needed to keep the gas flow (speed) in flash dryer and separator constant, regardless of drying gases required.

Features

- Wet and fine materials can be added after the grinding equipment, preferably to the flash dryer
- Partly fine materials can be fed directly to the the separator

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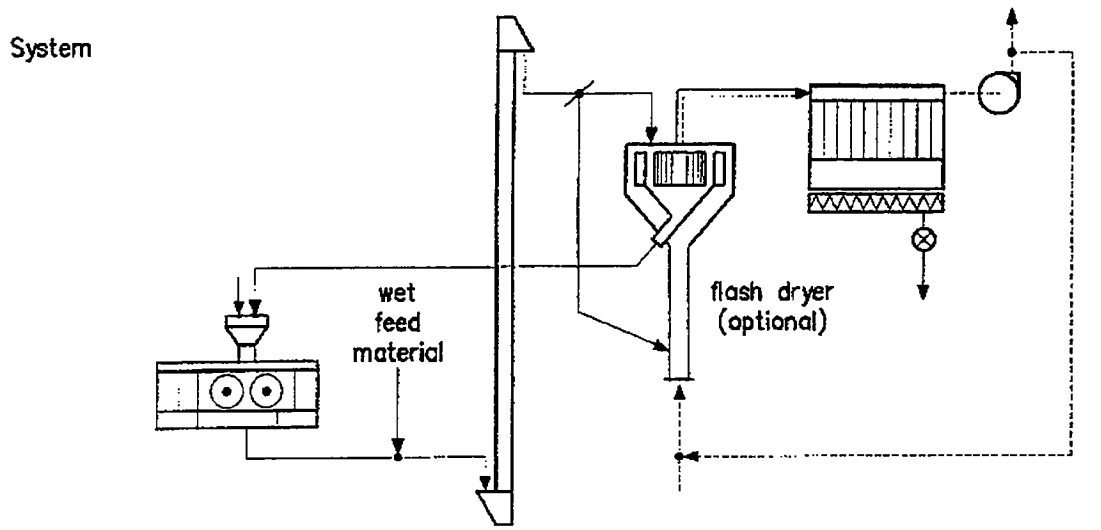
Figure 11: Separator (with flash dryer)



		without flash dryer	with flash dryer
Feed size	(mm)	< 5	< 5
Drying capacity	(%H_2O)	< 5	< 8
Spec. load	(kg $H_2O/m^3 \cdot h$)	–	–
Temperature max.	(°C)	< 450	< 900

4. ENERGY BALANCE (HEAT BALANCE)

The calculation or recalculation of a drying process is normally done through an energy balance. The aim of the heat balance is to find the necessary quantity of drying gases to cover the required drying energy.

In principle all energy inputs (indicated as heat) have to be equal to the energy outputs of the drying system.

Procedure:

1) Define borders of the system
2) Define inputs and outputs
3) Compute the balance based on

   Heat inputs = outputs [kJ/kg product]

   Gas flow inputs = outputs [Nm^3/h]

   (including false air)
4) Observe the restrictions
   - max. gas speed
   - max. gas temperature (available and equipment related)
   - dew points
   - plant altitude (➔ ait/gas density)

Hints

- The heat is always related to 1 [kg] of system output product dry
- The heat balance is based on a reference temperature of 20 [°C], therefore the Δt is always (t - 20)
- Dust refers to the amount of dust contained in the kiln or cooler exhaust gases/air
- In the grinding heat all major electrical consumers are included (mill, separator, fan) as energy measured at the equipment drive shaft (if not, to be converted with drive efficiency)
- The evaporation heat consists of heating up the water to operating temperature and the subsequent portion of evaporation energy
- The radiation/convection losses are often assumed as a spec. figure related to [kg] water evaporated for simplicity reasons.

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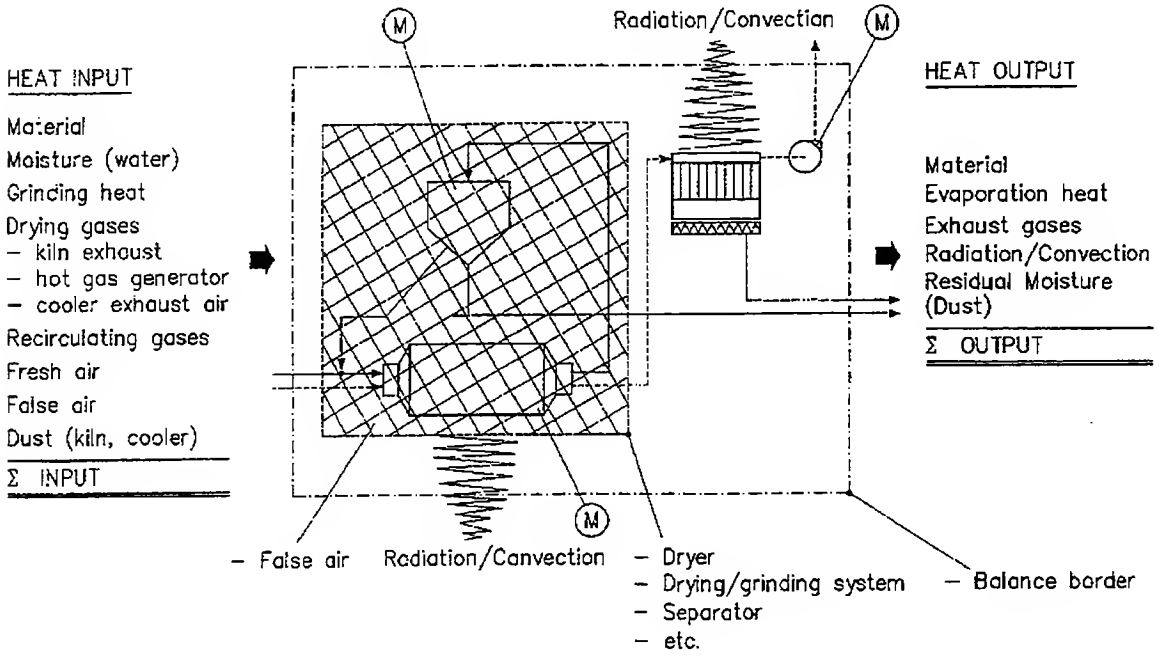
Figure 12: Heat Balance



Heat Balance (Example)

Raw grinding system:

Data	Vertical roller mill with filter and fan			
	Production		145 [t/h]	raw meal dry
	Moisture content		9 [%]	initial
	Moisture content		0.5 [%]	residual
	Electrical energie	mill	1050 [kW]	counter
		separator	35 [kW]	counter
		fan	1100 [kW]	counver
	Drive efficiencies		93 [%]	
	Drying gases	kiln exhaus gas	340 [°C]	
	Mill exhaust gases	(after fan)	90 [°C]	
	False air rate	system	12 [%]	
	Recirculating air		42 [%]	
	Dust in kiln exhaustet gas		70 [g/Nm^3]	

Index	G	[t/h]	Mill production	
	P_{abs}	[kW]	Absorbed power at counter	
	η	[-]	Drive efficiency	
	V	[Nm^3/h]	Gas/air flow rate	
		Index	k	kiln exhaus
			h	hot gas (from auxiliary furnace)
			c	cooler exhaust air
			f	false air
			r	recirculating air
			e	mill exhaust gases
	cp	[kJ/Nm^3]	Spec. heat value (related to preference temperature	
		[kJ/kg]		
	W	[kg/h]	Amount of Water	
		Index	in	moisture in feed material
			ev	moisture evaporated
			out	moisture in product
	q	[kJ/kj H_2O]	Evaporation / heat	
	r	[kJ/kg H_2O]	Radiation / convection factor	
	t	[°C]	Temperature	
	Δt	[°C]	Temperature difference (operating temp. - reference temp.)	
	d	[kg/Nm^3]	Dust in drying gas	

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Figure 13: Heat Balance

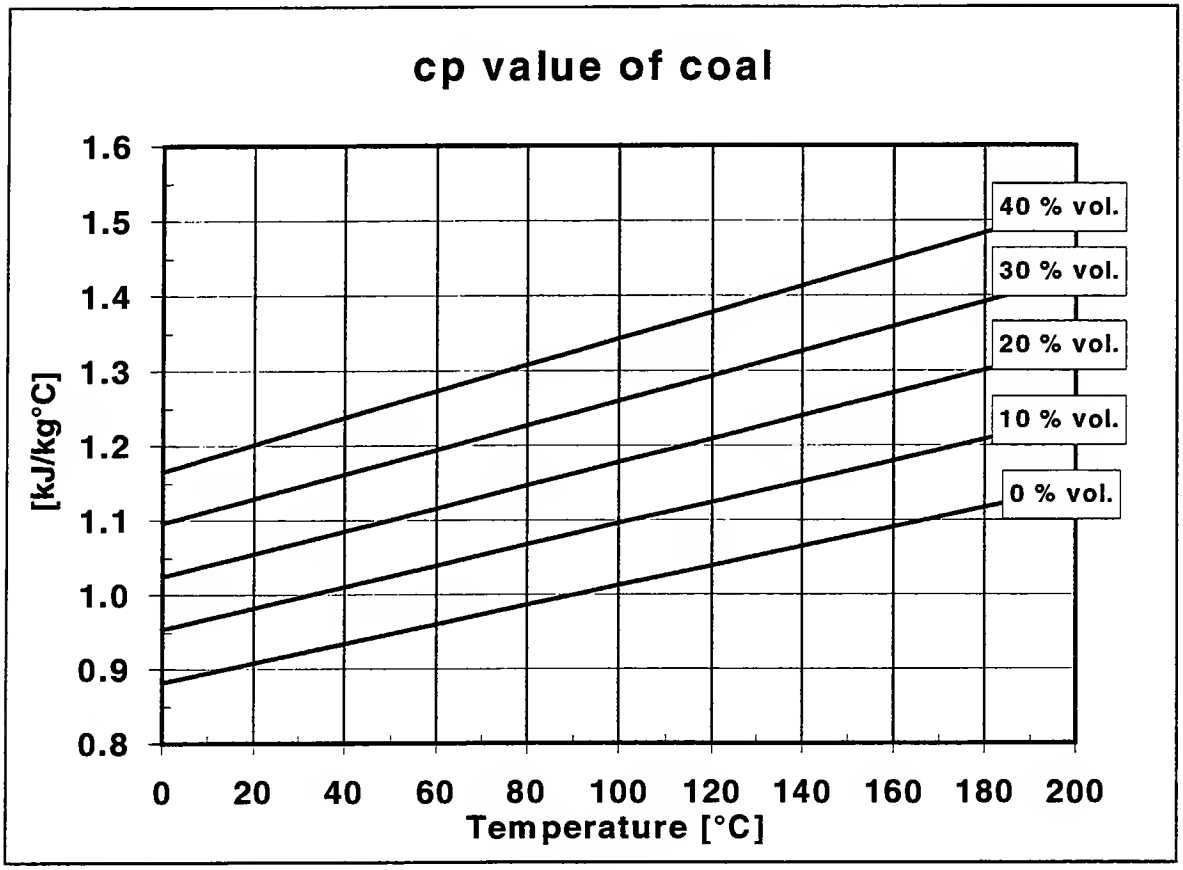
Heat Balance Vertical Roller Mill Example "HOLDERBANK"

INPUT					[kJ/kg]	[%]
Raw material (sensible heat)	G * 1000 *cp * Δt	/(G*1000)	145 * 1000 * 0.83 *(25-20)	/(145* 1000)	4.2	0.9
Grinding heat (mill, separator,fan)	$P_{abs} * \eta * 3600$	/(G*1000)	(1050 + 35 + 1100) * 0.93 * 3600	/(145* 1000)	50.4	11.1
Kiln exhaust gases	$V_k * cp * \Delta t$	/(G*1000)	101423 * 1.48 * (340-20)	/(145* 1000)	331.3	72.9
Hot Gases	$V_h * cp * \Delta t$	/(G*1000)		/(145* 1000)		
Cooler exhaust air	$V_c * cp * \Delta t$	/(G*1000)		/(145* 1000)		
Fresh air	$V_f * cp * \Delta t$	/(G*1000)		/(145* 1000)		
False air	$V_{fa} * cp * \Delta t$	/(G*1000)	20984 * 1.30 * (25-20)	/(145* 1000)	0.9	0.2
Recirculating gases	$V_r * cp * \Delta t$	/(G*1000)	73445 * 1.40 * (90-20)	/(145* 1000)	49.6	10.9
Water in feed material	$W_{in} * cp * \Delta t$	/(G*1000)	14338 * 4.19 * (25-20)	/(145* 1000)	2.1	0.5
Dust from kiln gases	$V_k * d * cp * \Delta t$	/(G*1000)	101423 * 0.07 * 1.03 * (340-20)	/(145* 1000)	16.1	3.6
SUM Input					**454.6**	**100.0**

OUTPUT						
Raw meal (sensible heat)	G * 1000 *cp * Δt	/(G*1000)	145 * 1000 * 0.89 *(90-20)	/(145* 1000)	62.3	13.7
Water evaporated	$W_{ev} * q$	/(G*1000)	13609 * (2501 + (1.88 * 90))	/(145* 1000)	250.6	55.1
Radiation/convection	$W_{ev} * r$	/(G*1000)	13609 * 50	/(145* 1000)	4.7	1.0
Exhaust Gases	$V_c * cp * \Delta t$	/(G*1000)	195852 * 1.40 * (90-20)	/(145* 1000)	132.4	29.1
Kiln Dust	$V_k * d * cp * \Delta t$	/(G*1000)	101423 * 0.07 * 0.89 * (90-20)	/(145* 1000)	3.1	0.7
Water in Raw meal	$W_{out} * cp * \Delta t$	/(G*1000)	729 * 4.19 * (90-20)	/(145* 1000)	1.5	0.3
SUM Output					**454.6**	**100.0**

Heat of evaporation q = 2501 + (1.88 * t) [kJ/kg H2O] heat of evaporation at 90 °C related to : 1 [kg] of raw meal
Radiation/convection factor r = 50 [kJ/kg H2O] Temperature 20 [°C]

Figure 14:



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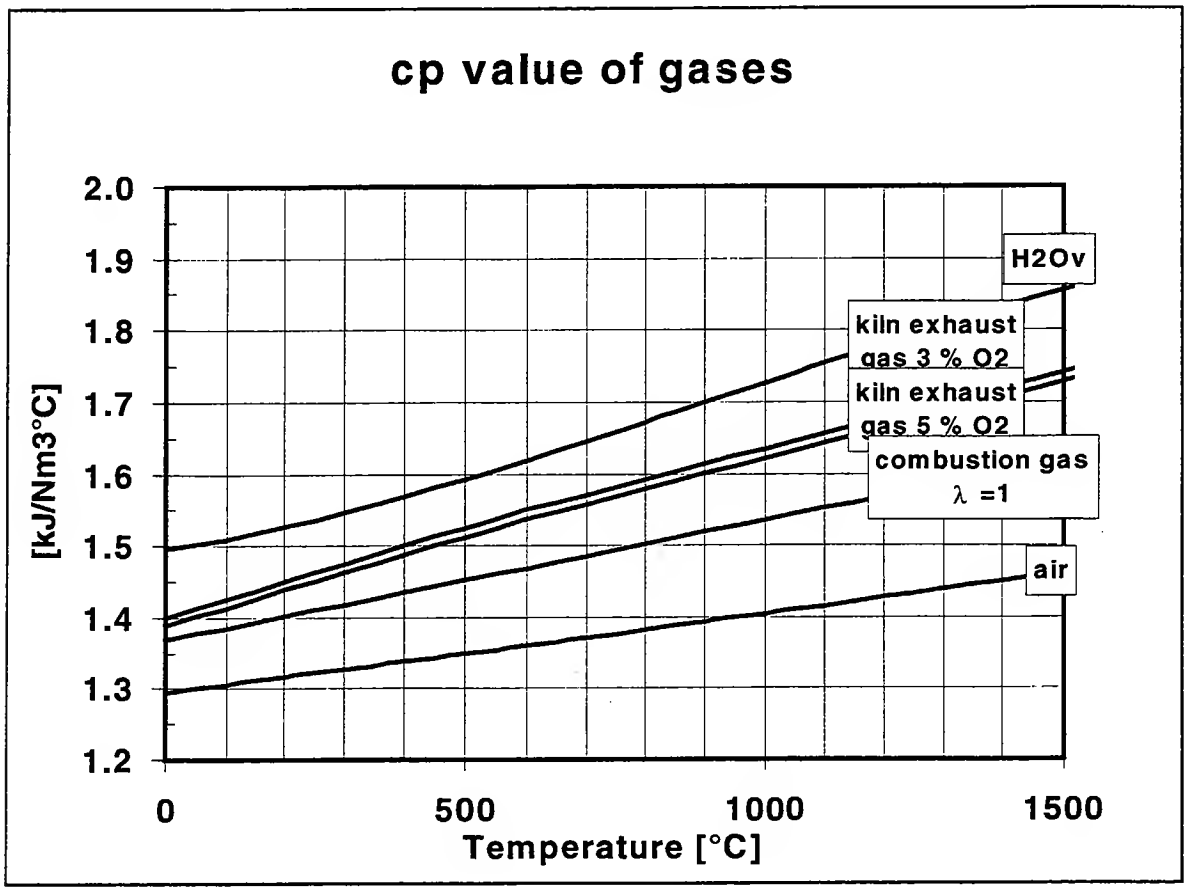
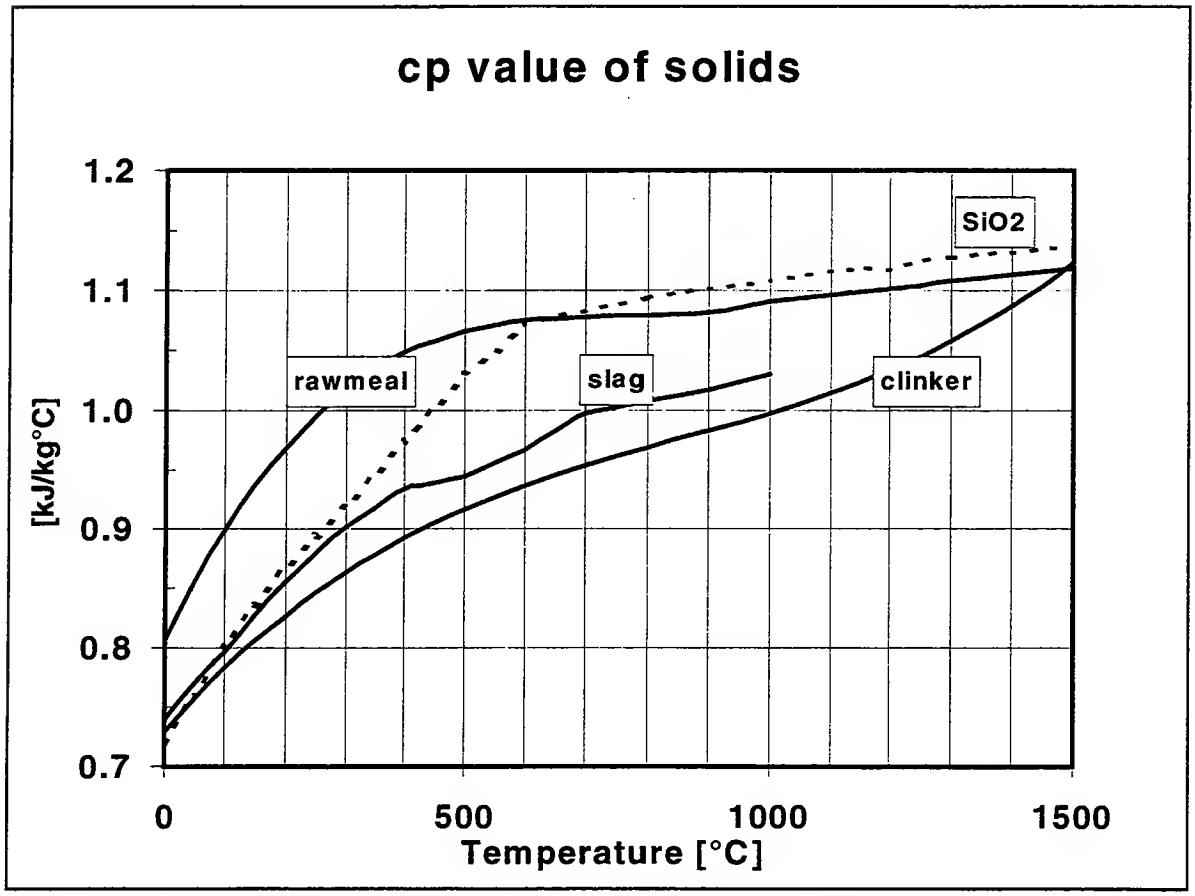

cp value of gases
2.0
1.9
1.8
1.7
1.6
1.5
1.4
1.3
1.2
[kJ/Nm3°C]
0
500
1000
1500
Temperature [°C]
H2Ov
kiln exhaust gas 3 % O2
kiln exhaust gas 5 % O2
combustion gas λ =1
air



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Chapter 12

Cement Grinding Systems

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Cement Grinding Systems

K. Breitschmid
VA 93/4014/E (Revised Version 02/98)

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1. OVERVIEW GRINDING SYSTEMS

The common cement grinding systems are summarized in Figure 1 and can be divided up into three main groups:

Tube Mill only

All grinding work is done in the tube mill and the set-up is:

- Tube mill in open circuit
- Tube mill in closed circuit with
  - Mechanical air separator
  - Rotor type separator

Tube Mill with Pregrinding Unit

A part of the grinding work is done in a pregrinding unit which can be:

Roller press

Vertical shaft impact crusher

Vertical roller mill (without separator, only grinding tool is used or with separator)

Depending on the selected system, the tube mill is in open or closed circuit (in most applications the tube mill is in closed circuit).

Finish Grinding System

The tube mill is omitted and all grinding work is done in a more efficient comminution machine. The possible systems are:

Roller press with desagglomerator and rotor type separator

Vertical roller mill with integrated rotor type separator

Horizontal roller mill with rotor type separator

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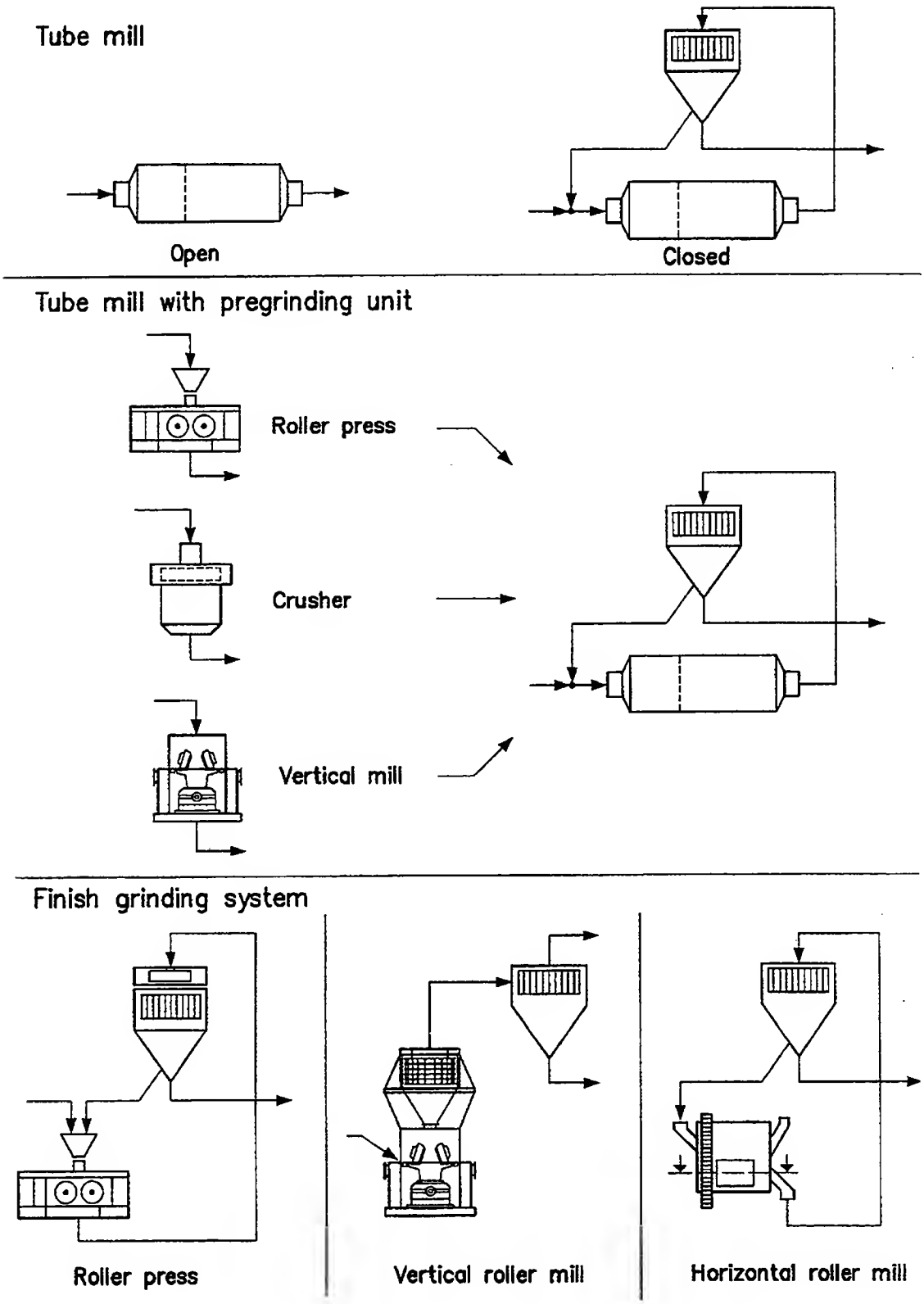
Figure 1: Overview



2. REQUIREMENTS

The requirements on a cement grinding system are manifold and a profound investigation and valuation of all criteria is necessary in order to find the appropriate solution (technically and economically) for:

Procuring of a new grinding system

Expanding and optimizing an existing system

The common criteria are summarized in Figure 2 and the most important will be further discussed.

Production - Cement Types

From a process point of view the grouping is:

- Portland cement 2'500 - 6'000 [cm^2/g]
- Portland cement with minor additive ≤ 5 [%] 2'500 - 6'000 [cm^2/g]
- Composite or blended cement, additive > 5 [%] 2'500 - 9'000 [cm^2/g]

  Possible additives for composite cements are:
  Fly ash, pozzolana, blast furnace slag, limestone,
  silica fume, calcined clay etc.

A system must be able to grind one or several of above cement types to the required fineness at the lowest possible energy consumption.

In case of Portland cements cooling must be performed within the system.

In case of composite cements drying within the system is often a need and components with different grindabilities must be interground.

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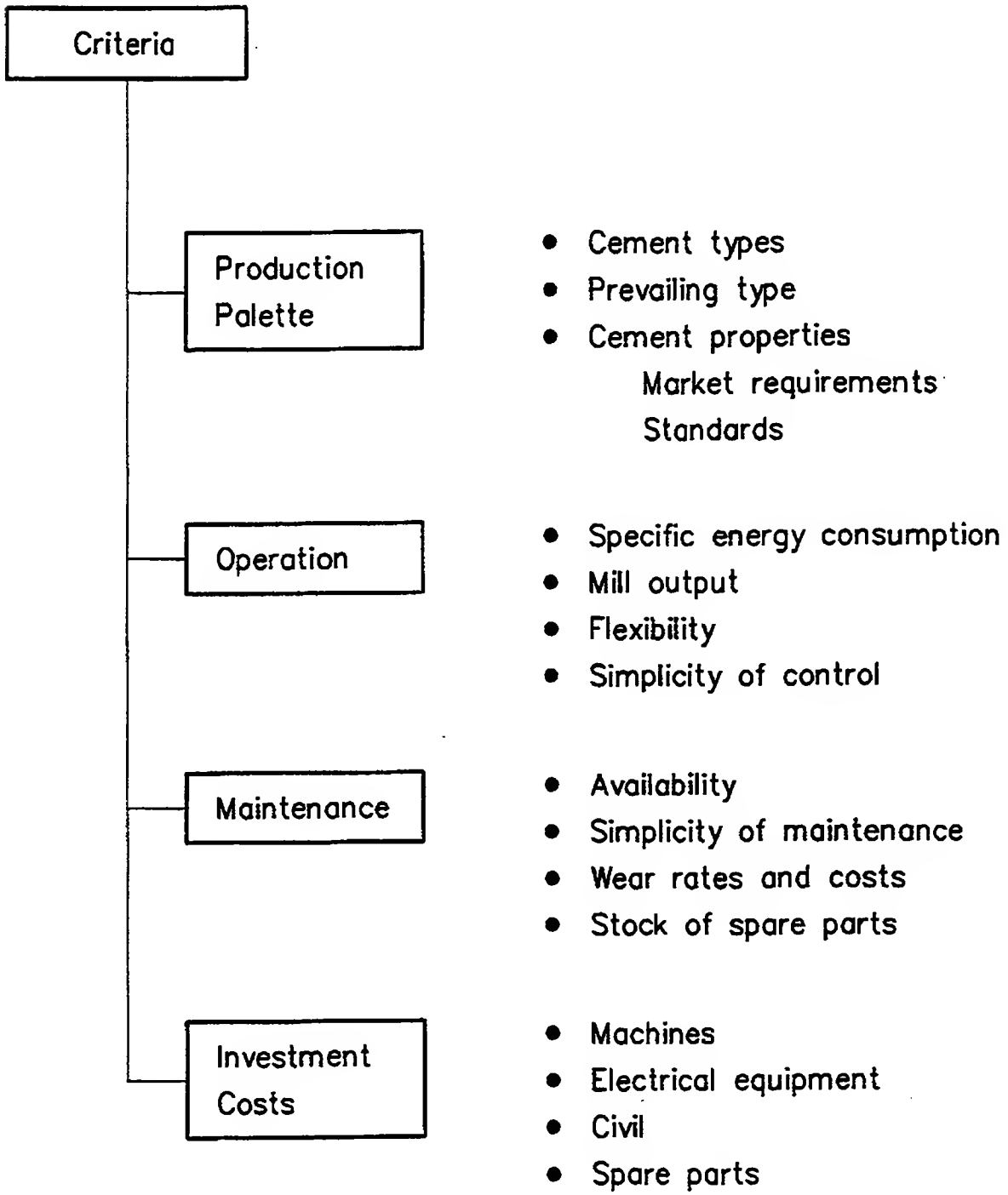
Figure 2: Requirements



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Production - Cement Properties

The cement properties are influencing the:

- Workability of the concrete
- Early and late strength

The necessary values are given by the market requirements and the standards.

The cement properties are mainly influenced by:

- Composition of the cement
- Chemical and mineralogical composition of components
- Reactivity of clinker
- Particle size distribution, particle shape
- Sulphate agent:
  - Distribution and fineness
  - Modification and its solubility

The grinding system has influence on:

Particle Size Distribution and Shape

The particle size distribution is presented in the RRSB-diagram as shown in Figure 3 and characterized by:

- Steepness of distribution n
- Limiting particle d' (36,8 [%] R)

The steepness is depending on the type of grinding system, as shown in Figure 3, and can be adjusted within a certain range.

Without considering the chemical-mineralogical influences often the tendency is, going from wide to narrow distribution:

Early strength → decrease

Late strength → increase

Workability → worsening

The particle shape can be characterized (simplified):

Round shape → Tube mill system (grinding by friction)

Angular shape → Finish grinding system (grinding by compression)

Angular shape could result in a higher water requirement.

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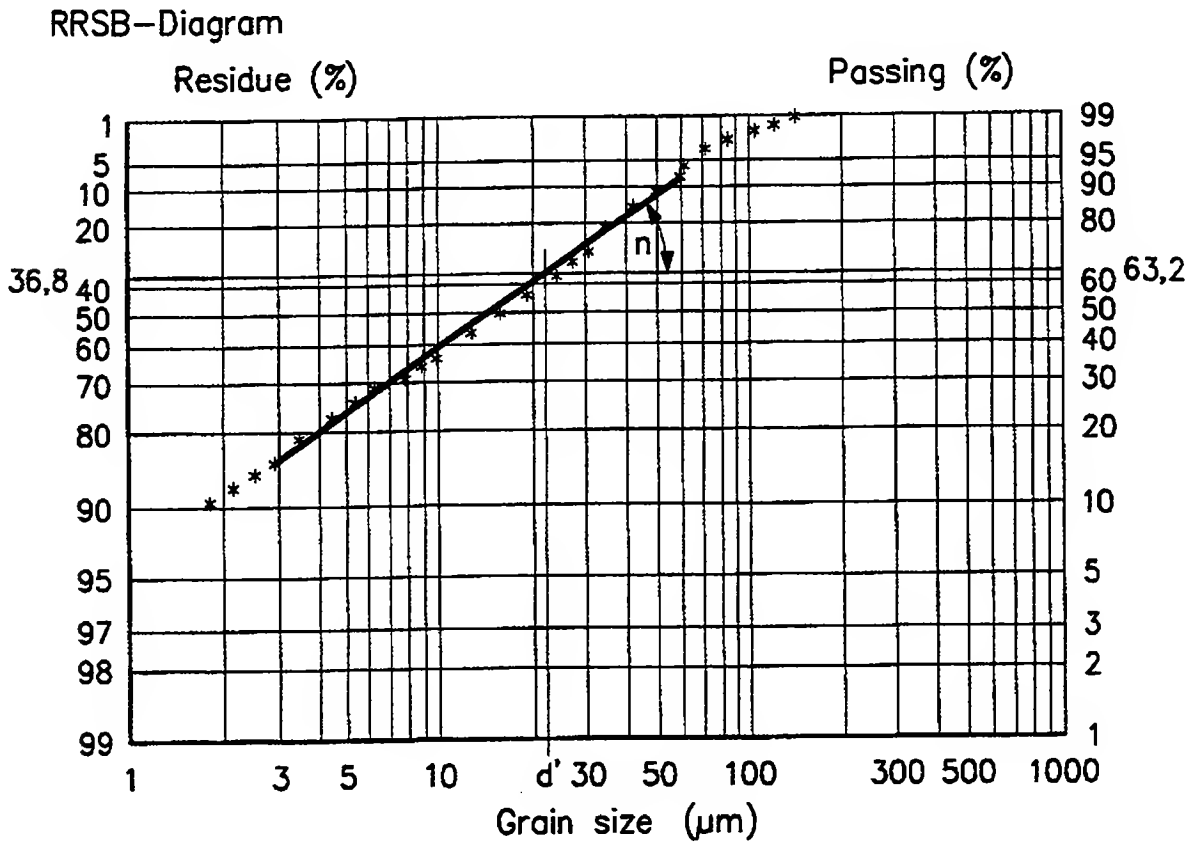
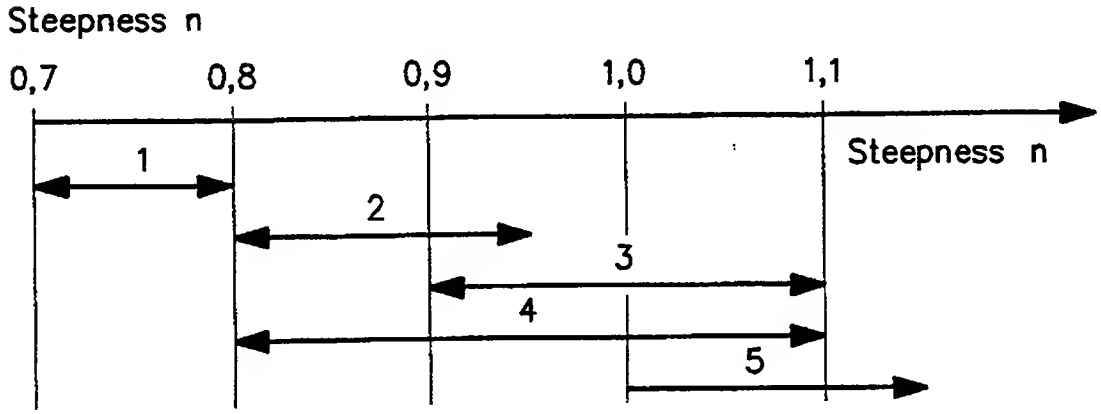
Figure 3: Particle Size Distribution





1 Open circuit
2 Closed circuit mechanical air separator
3 Closed circuit rotor type separator
4 Tube mill with pregrinding unit (depending separator)
5 Finish grinding system

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Sulphate Agent

In order to achieve the optimal workability of concrete an adaptation of clinker reactivity, particle size distribution and soluble calcium sulphate is a need. The relation can be simplified as follows:

Clinker reactivity	Particle size distribution	Easily soluble $CaSO_4$
Low	Wide	Low
Low	Narrow	Medium
High	Wide	High
High	Narrow	Very high

Clinker reactivity low $C_3A < 8$ [%], $Na_2O_{eqv.} < 0,6$, SR > 2,7

Clinker reactivity high $C_3A > 11$ [%], $Na_2O_{eqv.} > 0,9$, SR < 2,3

Particle size distribution wide $n < 0,9$

Particle size distribution narrow $n > 1,0$

Solubility of sulphate agent:

The modification of the sulphate agent is

- Dihydrate — fed to the mill as "gypsum"
- Hemihydrate } produced in the mill under high
- } temperature from dihydrate
- Soluble anhydrite }
- Natural anhydrite — fed to the mill as "gypsum"

The solubility is increasing in the sequence:

- Natural anhydrite
- Dihydrate
- Soluble anhydrite / hemihydrate

If supply of easily soluble sulphate is short

→ flash set can occur

If oversupply exists

→ false set can occur

If high amount of easily soluble sulphate is required, cement must be ground at high temperature.

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The following grinding systems are producing a narrow particle size distribution at a low grinding temperature:

- Tube mill in closed circuit with rotor type separator with product collection in bag filter (cooling in separator)
- Finish grinding system with roller press
- Finish grinding system with vertical roller mill and horizontal roller mill (less critical)

If the clinker has in addition a high reactivity the system must be laid out in such a way that:

- Steepness of particle size distribution can be adjusted
- Cement can be ground at high temperature

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Operation - Specific Energy Consumption

The total specific energy consumption of a grinding system is depending on a multitude of factors:

- **Type of grinding system**

  The highest influence on the energy consumption has the type of system. Figure 4 gives a comparison between tube mill open circuit, closed circuit, tube mill with pregrinding unit and finish grinding system. The given [kWh/t] in relation to fineness [cm^2/g] should only indicate at which level the individual systems are and are not absolute values.

Further factors are:

- **Mill feed**
  - Composition
  - Grindability of components
  - Grain size distribution
  - Moisture content of additive
- **Technical condition**
  - Main machines
  - Wear parts in general
  - Mill internals, ball charge etc.
- **Mill control**
  - Manual
  - Automatic (High Level Control)
- **Use of grinding aid**

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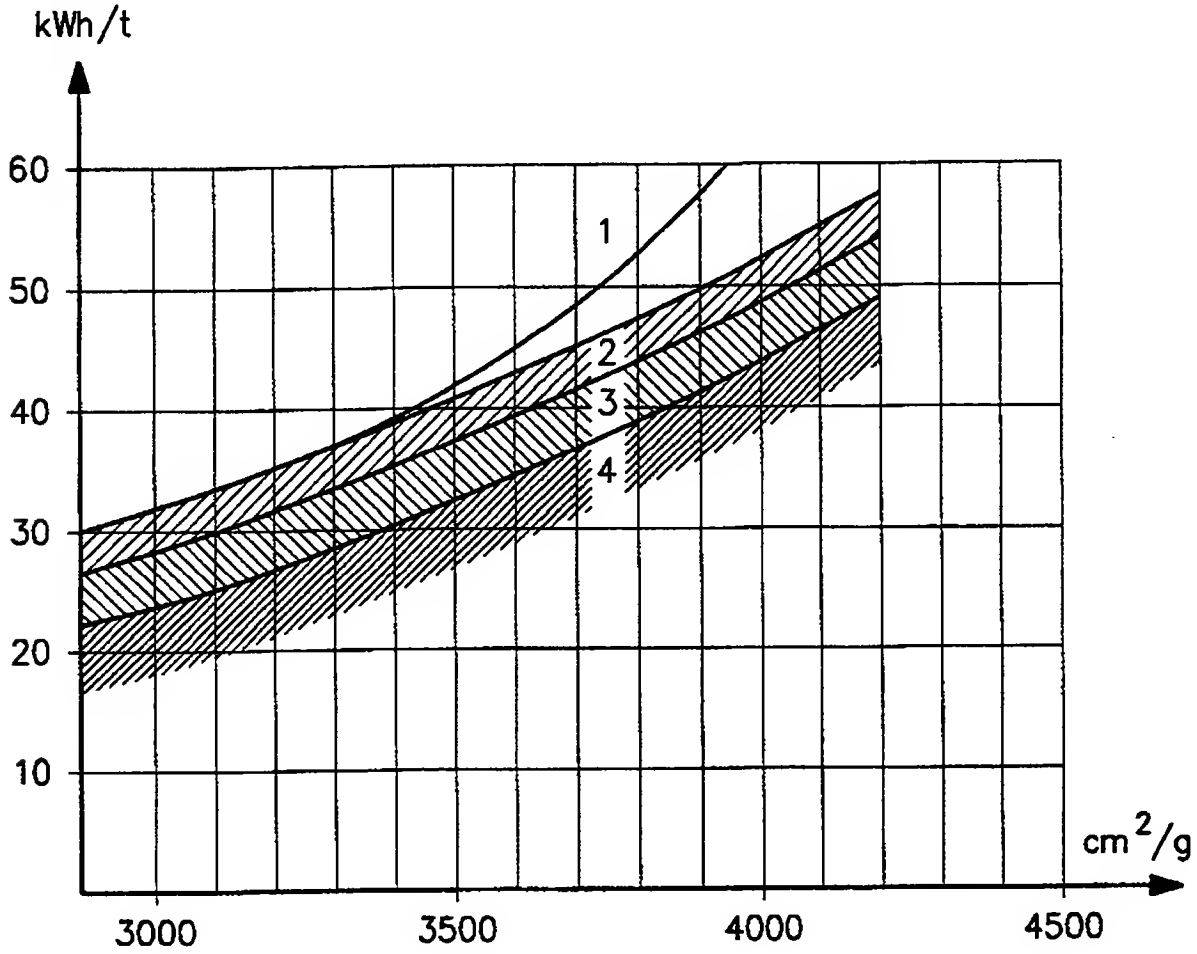
Figure 4: Specific Energy Consumption



1 Tube mill open circuit
2 Tube mill closed circuit
3 Tube mill with pregrinding unit
4 Finish grinding system

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3. GRINDING SYSTEMS

3.1 Tube Mill Systems

3.1.1 Tube Mill in Open Circuit (Figure 5)

Main Elements of System

1) Feed bins with weigh feeders
2) Tube mill with two or three compartments
3) Mill filter: Electrostatic precipitator or bag filter
4) Metal trap: Vibratory screen
5) Cement cooler: Not standard outfit

Judgement

- **Production**
  Suited for Portland cement with low fineness.
  Not recommended if several types must be ground.
  Product characterized by wide particle size distribution.
- **Operation**
  Specific energy consumption high for fineness > 3'000 [cm^2/g].
  Mill output up to 200 [t/h].
  Product temperature high. Cooling by mill ventilation and water injection: Due to limited ventilation often cement coolers are required.
  Sensitive to variation in mill feed grindability and granulometry.
- **Maintenance**
  High availability of system.
  Simple to maintain and proven wear parts.

Comment

The tube mill in open circuit can not be recommended for new installations (exception combination with pregrinding unit).

Conversion to closed circuit often recommendable due to more flexibility and reduced energy consumption.

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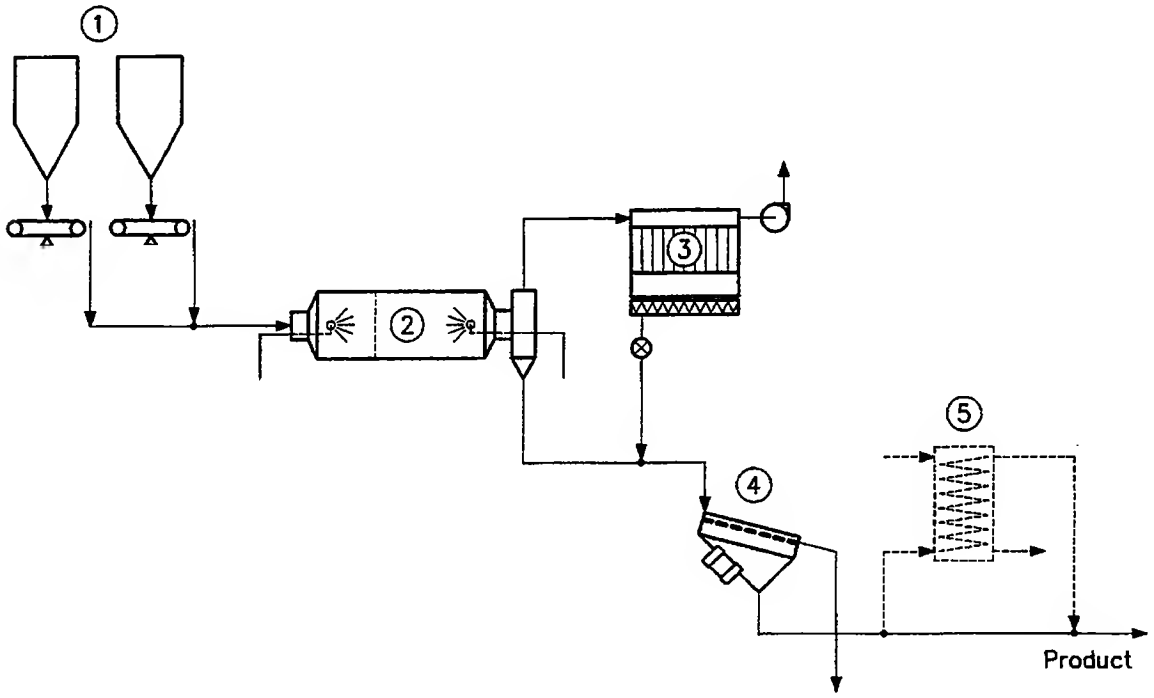
Figure 5: Tube Mill in open circuit



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3.1.2 Tube Mill in Closed Circuit

System with mechanical air separator (Figure 6)

Main Elements

1) Feed bins with weigh feeders
2) Tube mill with two compartments
3) Mechanical air separator
   Configurations: 1 or 2 mechanical air separators
   Separator with/without dedusting
4) Mill filter: Electrostatic precipitator or bag filter with/ without prededusting in static separator or cyclone 4a
5) Cement cooler: Not standard outfit

Judgement

- **Production**
  Suited for Portland cement, Portland cement with minor additives and composite cements (if equipped accordingly e.g. hot gas generator).
  Product characterized by wide to medium particle size distribution.
- **Operation**
  Specific energy consumption medium to high depending on product fineness.
  Mill output up to 200 [t/h].
- Product temperature medium. Cooling by mill ventilation and water injection: Optionally by fresh air in separator and/or cement cooler.
  Variations in mill feed grindability can be equalized.
- **Maintenance**
  High availability of system.
  Simple to maintain and proven wear parts.

Comment

Most commonly used system for cement grinding.

Not recommended for new installations due to separator design.

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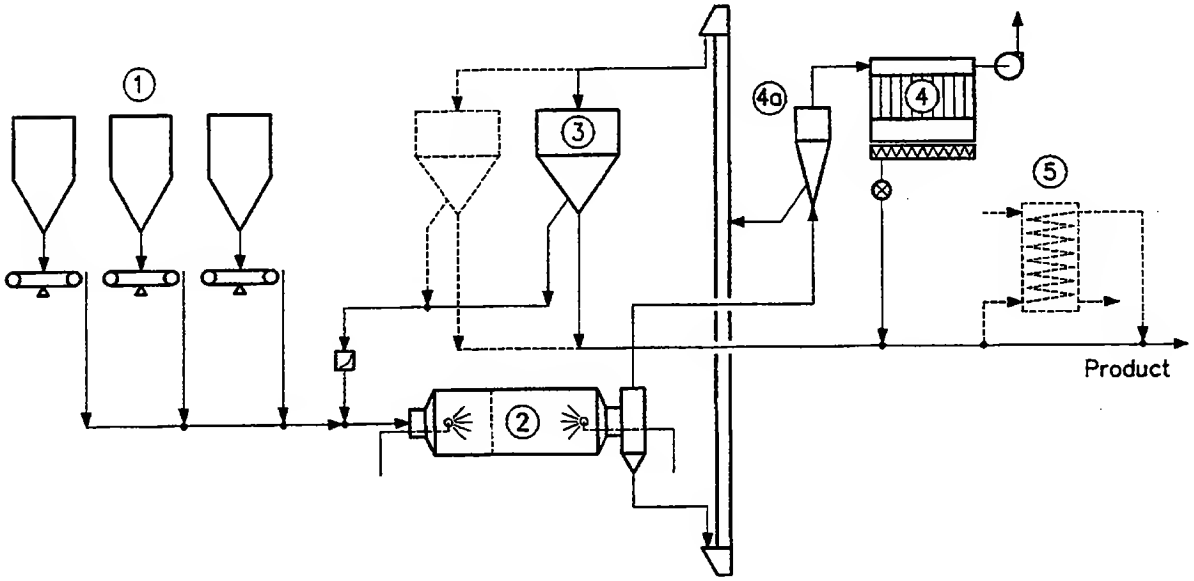
Figure 6: Tube Mill in closed circuit, Mechanical Air Separator



System with rotor type separator (Figure 7)

Main Elements

1) Feed bins with weigh feeders
2) Tube mill with two compartments
3) Rotor type separator
   Configurations: 3 rotor type separator with bag filter
   3a rotor type separator with cyclones and small dedusting bag filter
4) Mill filter: Bag type
   Cement cooler: Not standard outfit and mainly in combination with rotor type separator with cyclones

Judgement

- **Production**
  Suited for Portland cement, Portland cement with minor additives and composite cements (if equipped accordingly).
  Product characterized by narrow particle size distribution.
- **Operation**
  Specific energy consumption medium to low.
  Mill output > 200 [t/h].
  Product temperature medium:
  - Configuration 3 - Product respectively grinding temperature low. Cooling by mill ventilation and fresh air supply to separator.
  - Configuration 3a - Product temperature medium. Cooling by mill ventilation and water injection and optionally by cement cooler.
- **Maintenance**
  High availability of system.
  Simple to maintain and proven wear parts.

Comment

Suitable for new installations.

Possibility for conversion of old systems from open to closed or for replacement of mechanical air separators.

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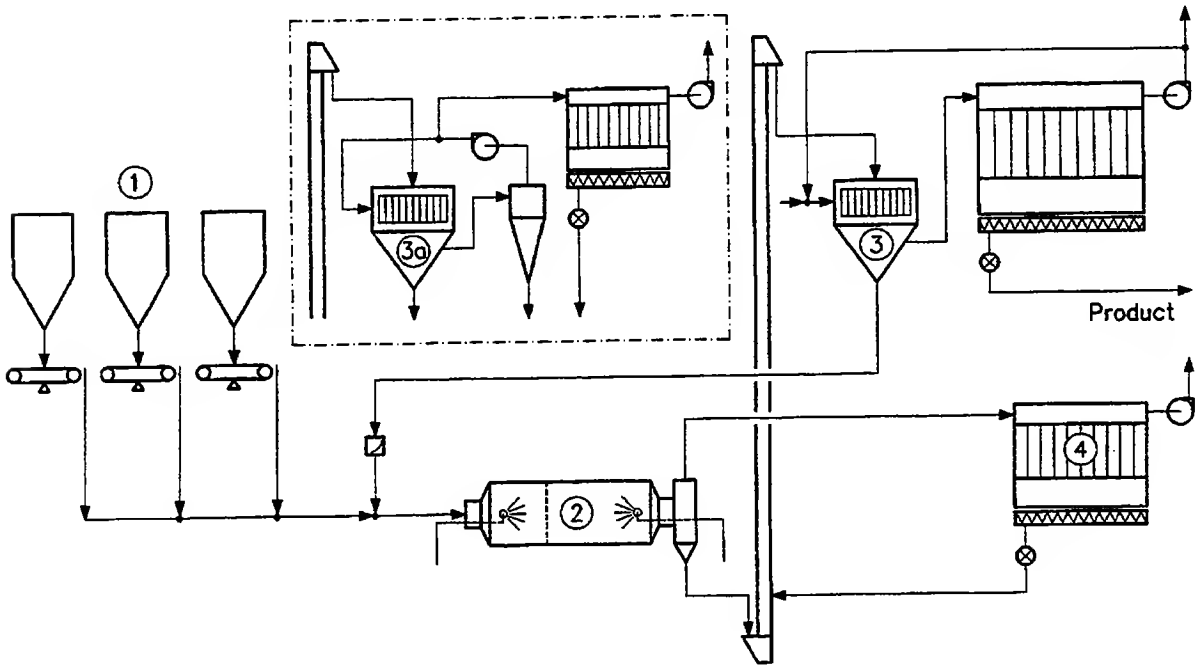
Figure 7: Tube Mill in closed circuit, Rotor Type Separator



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3.2 Tube Mill Systems with Pregrinding Unit

3.2.1 Roller Press

The roller press in front of the tube mill takes over a part of the grinding work of the mill. Size reduction in a press is more efficient than in a tube mill.

Basic configurations are (Figure 8):

- **Roller press in pregrinding mode**
  Press independent of tube mill, can serve several mills.
  Type of tube mill: Closed circuit with mechanical air separator or rotor type separator.
  Power relation: Press : mill ~ 20 : 80 [%].
- **Hybrid system**
  Press linked together with tube mill, can serve only one mill.
  Type of tube mill: Closed circuit with mechanical air separator or rotor type separator.
  Power relation: Press : mill ~ 30 : 70 [%].
- **Two stage grinding system**
  Press equipped with separator and desagglomerator, with/without intermediate storage, press could serve several mills.
  Type of tube: Closed or open circuit.
  Power relation: Press : tube mill ~ 50 : 50 [%].
- **Finish grinding system**
  Press equipped with separator and desagglomerator.
  Cement is ground in roller press only.

Rule of Thumb

1 [kW] absorbed in the press replaces 2 - 2,5 [kW] in the tube mill.

If roller press is used as a retrofit for existing systems, capacity increases of 25 up to 100 [%] can be achieved.

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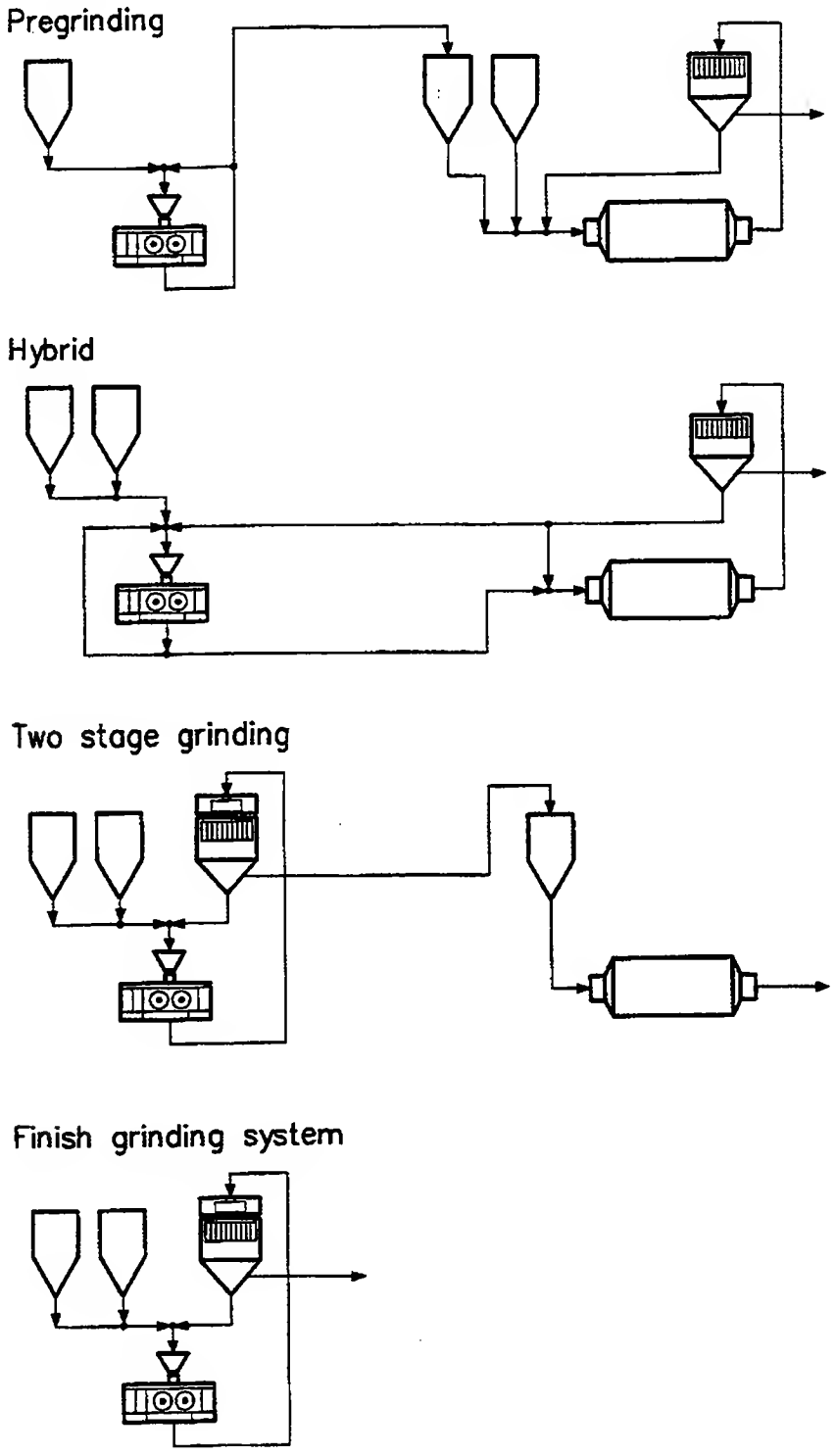
Figure 8: Configurations with Roller Press



3.2.1.1 Hybrid / Pregrinding System

Main Elements (Figure 9)

1) Press feed bins with weigh feeders (e.g. clinker, gypsum)
2) Roller press
3) Overflow bin or dividing gate for slabs
4) Additional mill feed bin (e.g. for wet additive)
5) Tube mill with two compartments
6) Rotor type separator: With bag filter or with cyclones
7) Mill filter: Bag type
8) Overflow bin or dividing gate for separator coarse material (not required if in pregrinding mode)

Operation of Roller Press

The press is fed by fresh feed, slabs and coarses from the separator. If the latter is set zero the press works as a pregrinding system with slab recirculation only.

Judgement

Production
Suited for all cement types if equipped accordingly.
Product characterized by narrow particle size distribution.

Operation
Specific energy consumption low.
Mill output > 200 [t/h].
Product temperature depends on selected separator (see chapter 3.1.2: System with rotor type separator).
System can be operated also without roller press.

- **Maintenance**
  Roller press: Low wear but very sensitive to foreign matters (metal etc.) Wear protection reconditioning rollers still in a development phase.
  System: Availability is depending on roller press.

Comment

Suitable for new installations (only pregrinding system).

Possibility for capacity increase of existing systems.

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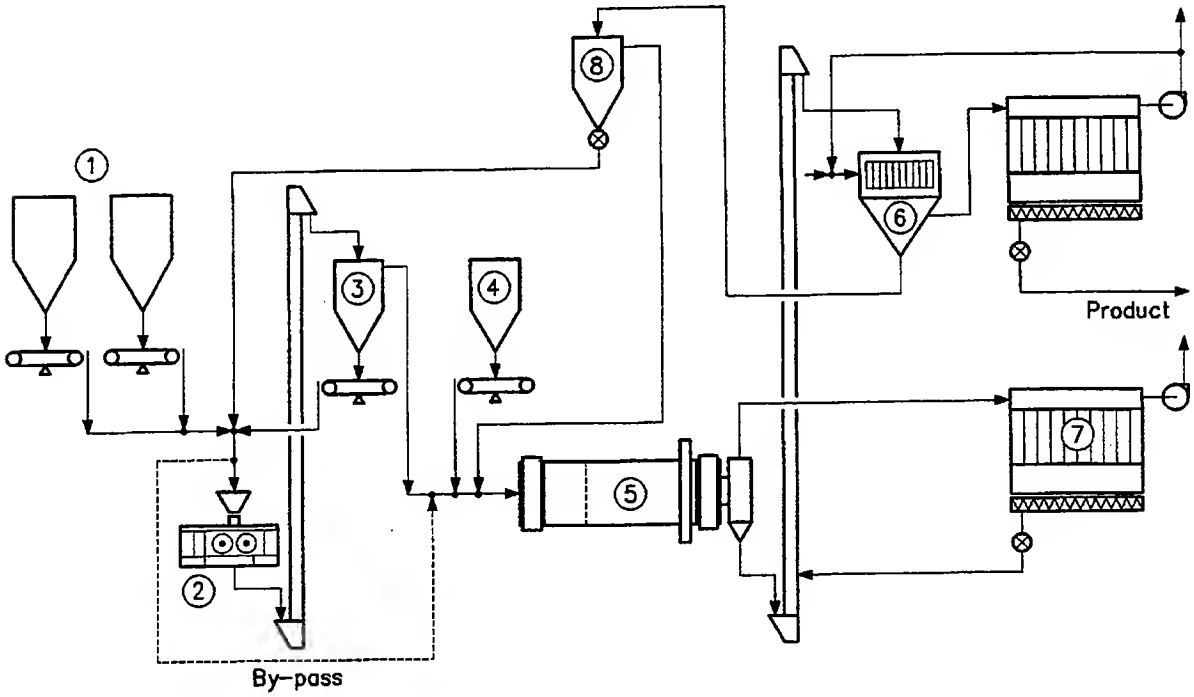
Figure 9: Hybrid System



3.2.1.2 Two Stage Grinding System

Main Elements (Figure 10)

1) Press feed bins with weigh feeders (clinker, gypsum)
2) Roller press
3) Rotor type separator with desagglomerator
   Separator with bag filter or cyclones
4) Intermediate bin with weigh feeder
5) Tube mill with one or two compartments (open or closed circuit)
6) Mill filter: Bag type
7) Rotor type separator: with bag filter or cyclones
8) Bin for additive with weigh feeder

Judgement

- **Production**
  Suited for Portland cement and also for composite cement; the additive can be fed directly to the tube mill.
  In the first stage (press with separator) the cement is ground 1'000 - 1'500 [cm^2/g] below desired product fineness and in the tube mill to the end fineness.
  Product characterized by medium to narrow steep grain size distribution.
- **Operation**
  System can only be operated without roller press if two compartment mill
  Specific energy consumption low.
  Max. mill output given by fineness and available press size ~ 150 - 200 (t/h).
  Product temperature medium to low. Cooling by mill ventilation, separator cooling and water injection.
- **Maintenance**
  Roller press: Low wear but very sensitive to foreign matters.
  Wear protection reconditioning rollers still in a development phase.
  System: Availability is depending on roller press.

Comment

Suitable for new installations (not favoured, complicated).

Possibility for capacity increase of existing systems.

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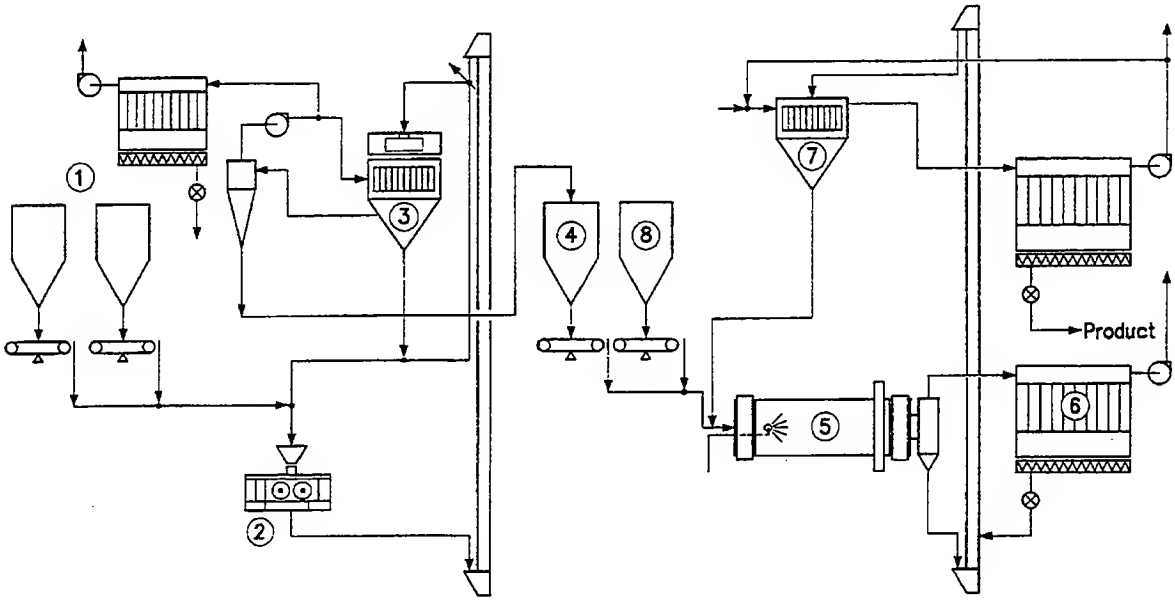
Figure 10: Two Stage Grinding System



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3.2.2 Vertical Shaft Impact Crusher

The vertical shaft impact crusher in front of the tube mill takes over a part of the grinding work from the first compartment of a tube mill. The material is crushed to μ 3 [mm].

Basic configurations are (Figure 11):

- **Vertical shaft impact crusher in open circuit**
  Material passes only once through the crusher.
  Crusher independent of mill, could serve several mills.
  Type of tube mill: Any type of mill.
- **Vertical shaft impact crusher in closed circuit**
  System works in closed circuit with vibratory screen. Due to screen sensitive against moisture in feed.
  Crusher independent of mill, could serve several mills.
  Type of tube mill: Any type of mill.

Comment

The utilization of the crusher is mainly seen as a retrofit for existing systems in order to increase grinding capacity and not for new installations.

Achievable increase: 15 - 20 [%].

Reduction specific energy consumption: 2 - 3 [kWh/t].

The application of the crusher is only sporadically.

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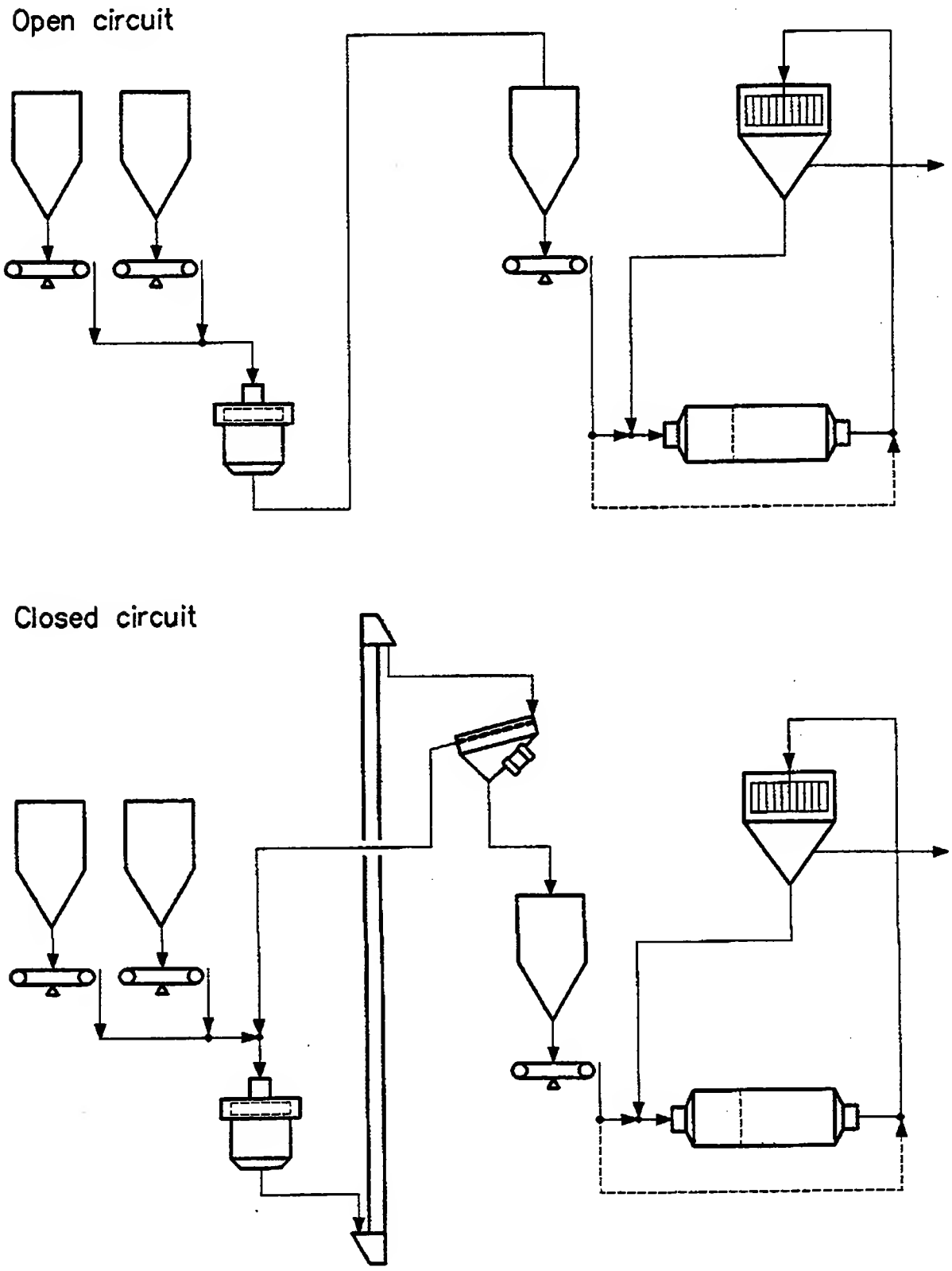
Figure 11: Vertical Shaft Impact Crusher



3.2.3 Vertical Roller Mill

The vertical roller mill, as known from the raw grinding, can be used as pregrinding machine. The basic mill design is as for raw grinding, however, the grinding tools, hydraulic system and drive are adjusted for clinker grinding. The mill is supplied with or without separator.

Possible configurations are (Figure 12):

- **Closed circuit with recirculation**
  Comparable with roller press with slab recirculation.
- **Closed circuit with screen**
  In both cases, the vertical mill can serve one or several tube mills. The feed size to the tube is ≤ 3 [mm].
- **Two stage grinding system**
  Comparable with roller press in two stage mode.

Comment

The achievable results are comparable with the roller press.

If the vertical mill is used as a retrofit for existing systems, capacity increases of 25 up to 100 [%] can be achieved depending on the configuration.

1 [kW] absorbed in the vertical mill replaces 2 - 2,5 [kW] in the tube mill.

The application of the vertical roller mill can mainly be found in the Far East (Japanese suppliers).

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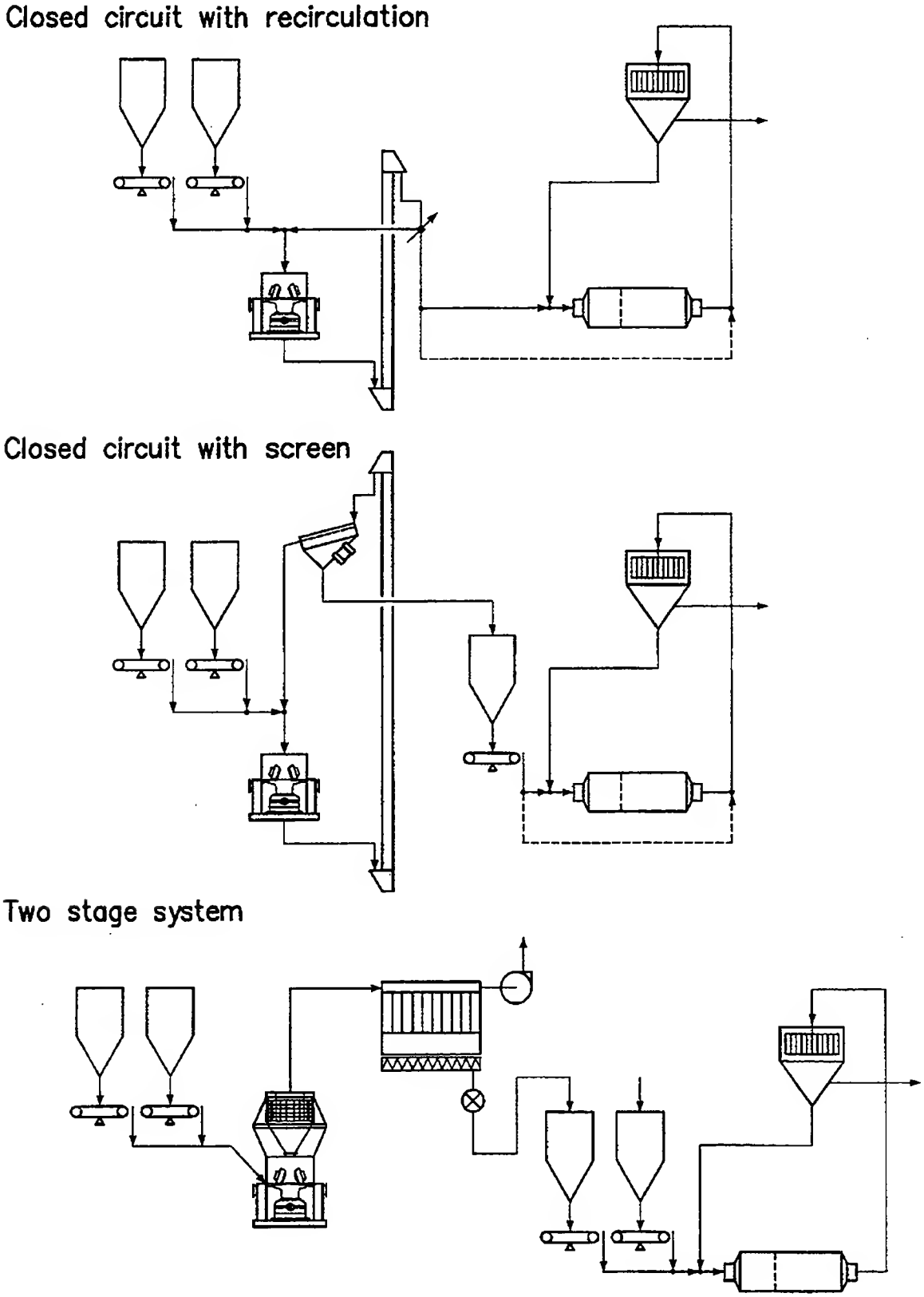
Figure 12: Vertical Roller Mill



3.3 Finish Grinding Systems

3.3.1 Roller Press (Figure 13)

Main Elements

1) Feed bins with weigh feeders
2) Roller press
3) Desagglomerator
   Options: Desagglomerator combined with separator
   Separate desagglomerator
4) Rotor type separator
   Configurations: Separator with bag filter
   Separator with cyclones and dedusting bag filter
5) Slab recirculation (if required by process)
6) Cement cooler: Not standard outfit

Judgement

- **Production**
  Suited for Portland cement and Portland cement with minor additives.
  Product characterized by narrow particle grain size distribution (particles angular shape instead of round shape as from tube mill).
- **Operation**
  Specific energy consumption very low.
  Mill output limited by max. available press size and fineness ~ 120 - 140 [t/h].
  Product temperature medium to low depending on separator set-up. Cooling only by fresh air supply to separator. Optionally cement cooler for product.
- **Maintenance**
  Since new developed system, long term availability figures not assured. Reconditioning wear protection rollers still item which must be further developed.

Comments

Possible grinding system of the future for Portland Cement due to simplicity and low energy consumption. Quality problems could occure if clinker has high C_3A content. Not suited for blended cement.

Each new installation needs detailed investigations regarding product quality.

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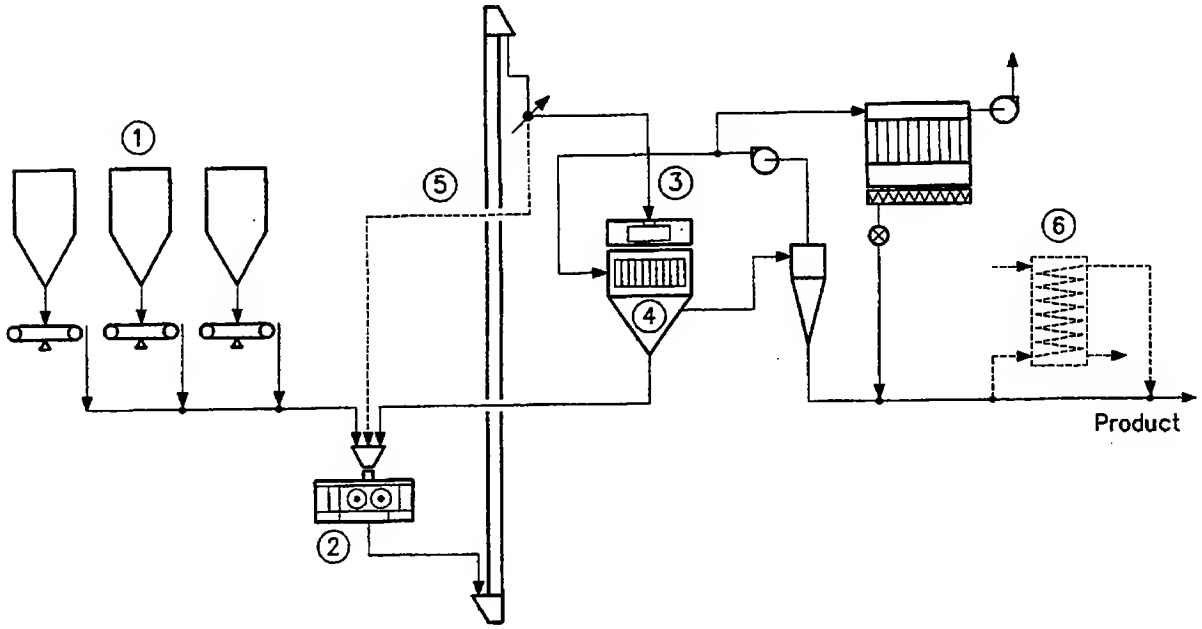
Figure 13: Finish Grinding System with Roller Press



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3.3.2 Vertical Roller Mill (Figure 14)

Main Elements

1) Feed bins with weigh feeders
2) Vertical roller mill with integrated rotor type separator
3) Mill filter: Bag type
4) Auxiliary furnace: Not standard outfit, mainly used for composite cement

Judgement

- **Production**
  Suited for Portland cement, Portland cement with minor additives and composite cements (if equipped accordingly).
  Product characterized by medium to narrow particle size distribution.
- **Operation**
  Specific energy consumption very low.
  Product temperature medium to low. Cooling by fresh air supply to mill.
  If clinker has high reactivity, possibly hot gas has to be supplied to the mill.
  Mill output strongly dependent on wear rate of liners table and rollers (output decrease with increasing wear rate).
  Mill output limited by fineness and available mill sizes ~ 120 - 140 [t/h].
- **Maintenance**
  Expensive wear parts table and liners.
  Long term availability figures not assured.

Comment

Alternatives to roller press system in finish grinding mode.

Not suited for all clinker types due to possible quality problems.

Good solution for composite cement due to excellent drying possibility.

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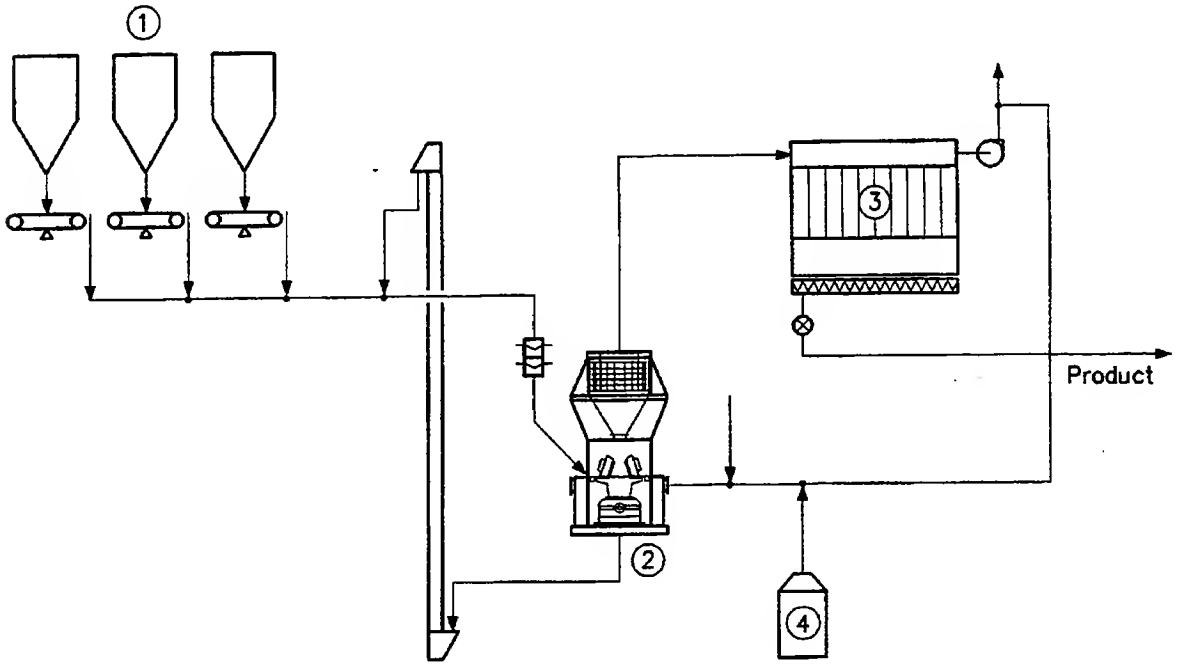
Figure 14: Finish Grinding System with Vertical Roller Mill



3.3.3 Horizontal Roller Mill (Figure 15)

Main Elements

1) Feed bin mill weigh feeders
2) Horizontal roller mill
3) Rotor type separator with bag filter
   a) Rotor type separator with bag filter and flash dryer
4) Auxiliary furnace: not standard outfit, only used for composite cement where the additive has a high moisture and is fed directly to the flash dryer.

Judgement

- **Production**
  Suited for Portland cement, Portland cement with minor additives and composite cements (if equipped accordingly).
  Product characterized by medium to narrow particle size distribution.
- **Operation**
  Specific energy consumption very low.
  Product temperature medium to low. Cooling by fresh air supply to separator (mill only dedusting).
  Mill output limited by fineness and available mill sizes ~120 - 140 [t/h].
- **Maintenance**
  Since new developed system, long term availability figures not assured.
  Wear problems similar to roller press.

Comment

Alternative to roller press and vertical mill.

Horizontal mills are still in a further development phase.

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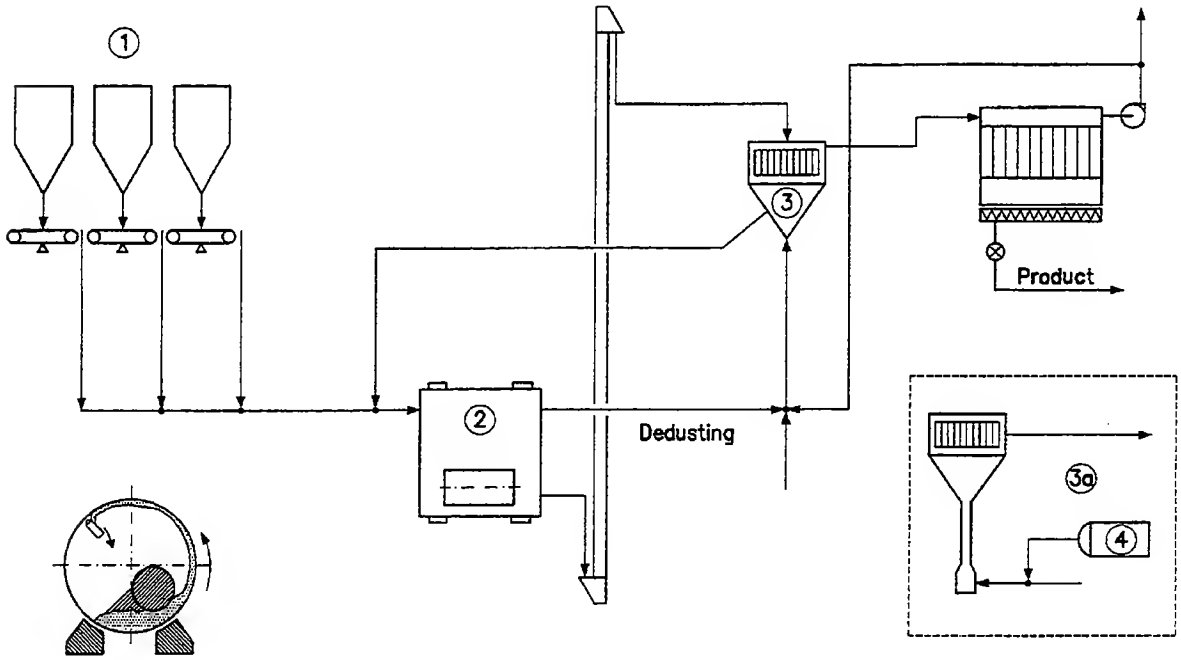
Figure 15: Horizontal Roller Mill



4. COMPARISON

For selecting a new system basically the following possibilities are given:

1. Tube mill closed circuit with rotor type separator

2. Tube mill closed circuit with roller press
2.1 Hybrid/pregrinding system
2.2 Two stage system

3. Finish grinding system
3.1 Roller press
3.2 Vertical roller mill
3.3 Horizontal roller mill

The following table summarizes important criteria in order to facilitate the selection:

	1	2		3		
		2.1	2.2	3.1	3.2	3.3
INVESTMENT COSTS [%]	100	110	120	100	110	110
SPECIFIC ENERGY [%]	100	85	75	60	70	65
PORTLAND CEMENT - LOW FINENESS - HIGH FINENESS	YES YES	YES YES	YES YES	YES LIMITED	YES LIMITED	YES LIMITED
COMPOSITE CEMENT	YES	YES	LIMITED	LIMITED	YES	LIMITED
CEMENT QUALITY	OK	OK	OK	*PARTLY OPEN	*PARTLY OPEN	*PARTLY OPEN
EXPANSION POSSIBILITY	YES	NO	NO	NO	NO	NO

*Partly open: Not recommendable if clinker is very reactive

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Cooling and Heating in the Cement Grinding Process

Hp. Fisch
PT 99/14489/E

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5. Water injection **618**
6. Cooling in the separator **621**
7. Cement coolers **622**
8. Heat balance (energy balance) **624**

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1. INTRODUCTION

Cooling and heating in the cement grinding process becomes more and more important. On one hand the market demands in many areas cool cement (e.g. < 60 °C) and on the other hand the cement manufacturing process requires an accurate balancing of the material temperature. The grinding process does not only require cooling but also heating is necessary in many cases to keep the process optimal and to assure a determined cement quality. Specially with the increasing production of blended cements with moist additives, the temperature must be neatly monitored and assured to efficiently produce a good quality product.

The following paper deals with cooling and heating of tube mill systems. Some considerations are also valid in an adapted way to other mill principles such as roller presses, vertical and horizontal roller mills.

What refers to drying and heat balances for various materials is dealt with in the paper "Drying Technology PT 96/14027/E". The heat balance at the end of this paper is adapted for a cement grinding system.

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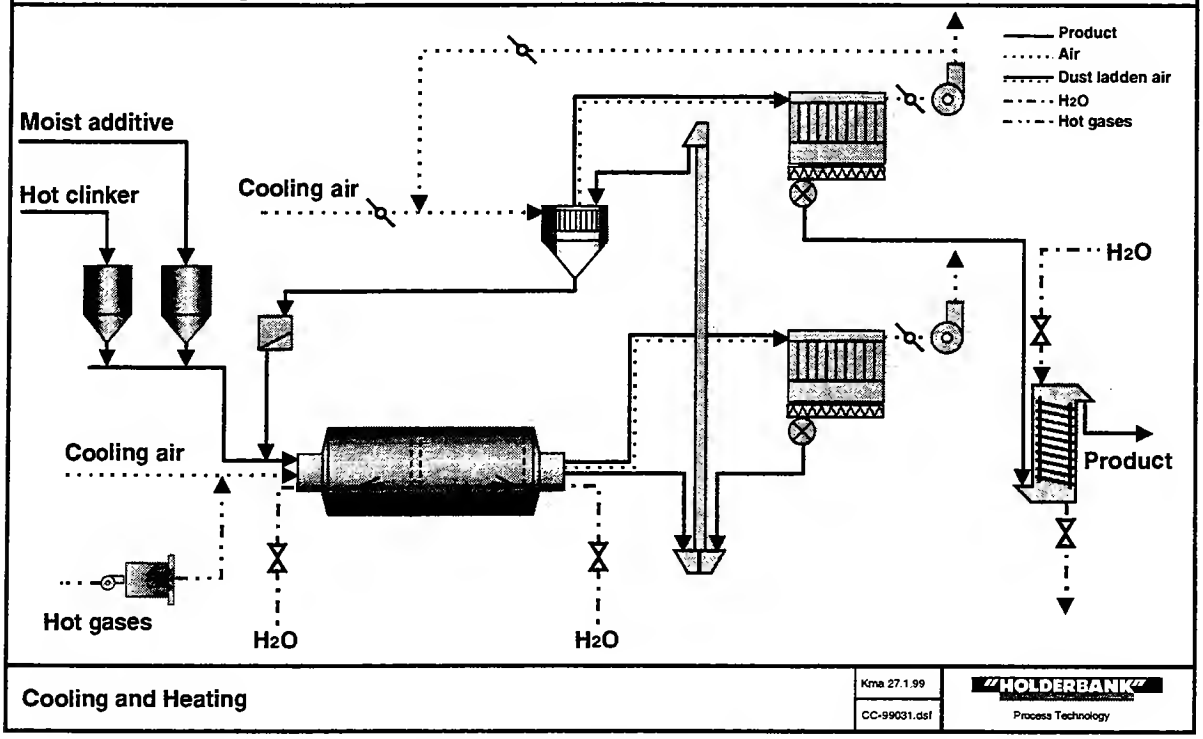
2. COOLING AND HEATING POSSIBILITIES

Fig.1 Cooling / heating possibilities



There are and were many possibilities for cooling and heating in the cement grinding process. The main possibilities are:

Cooling

- Air cooling of the mill (mill ventilation), where the air has a combined function of cooling and to some extent material transport through the mill.
- Air cooling in the separator by using an adjustable amount of ambient air to cool the mill discharge product which will be divided into finish product and separator returns.
- Water spray into the mill, depending on the conditions into the 1st and/or the 2nd compartment, to keep the mill discharge temperature within limits.
- Cement cooling with a cement cooler. The cement slides over a water cooled cylindrical surface and is cooled by convection.

Heating

- Hot gases/air can be used to evaporate the moisture in the mill feed and to keep the temperature profile within the mill above determined levels. Hot gas may be supplied by hot gas generators or kiln exhaust gases as well as clinker cooler exhaust air used.

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3. MILL VENTILATION AND DEDUSTING

3.1 Mill air cooling

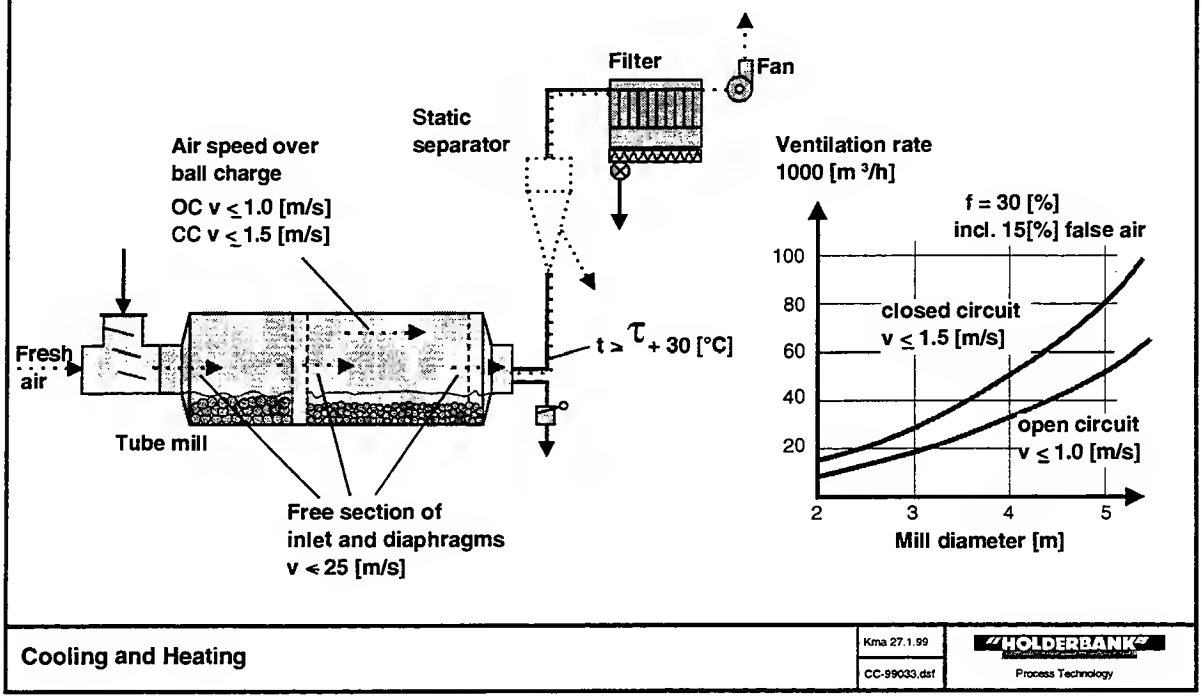
Purpose

The mill ventilation has 3 main functions

- mill (product) cooling
- transport of fines through mill
- fluidization of material within the mill

System

Fig.2 Mill ventilation



The mill inlet has to provide sufficient open area to draw in the ventilation air. Preferably stair type inlets are used.

Diaphragms and mill outlets should be designed to enable an adequate passage of air at reasonable air speeds to keep the air pressure low.

The mill outlet seal has to minimize the false air inleakage and therefore a pendular flap must be installed.

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The static separator is still used in older mill systems. For new installations the static separator is no more used as the new dynamic separators are more efficient.

Dimensioning of mill ventilation

- Air flow rate

  The installed air flow rate is only based on the air speed over the ball charge. Fig 2 shows the values based on nominal mill diameters, 30 [%] ball filling and 15 [%] false air. The air flow rates are therefore nominal fan volumes.

  Air speeds for system dimensioning are:
  - closed circuit mills $\leq$ 1.5 [m/s] over ball charge
  - open circuit mills $\leq$ 1.0 [m/s] over ball charge
- Inlet, outlet and diaphragms, maximum air speeds of $\leq$ 25 [m/s] related to gross open area.
- A false air rate of ~15 [%] has to be considered between mill outlet and fan. This results normally in air temperature of 5 [°C] below cement temperature at mill discharge. Higher temperature differences point to higher false air rates.
- The mill exhaust air temperature must always be kept $\geq$ 30 [°C] above the dew point temperature (τ) to avoid condensation problems. Excessive false air favors condensation.

3.2 Mill heating for blended cements

In many cases, specially when grinding blended cements with moist additives, heating is required instead of cooling.

Up to a certain moisture content in the feed, clinker and grinding heat are sufficient to evaporate the moisture. However often additional heat is required for drying. This heat can be taken from kiln exhaust gases, clinker cooler exhaust air or be generated by an auxiliary furnace (hot gas generator).

As a general rule it can be said, that the material fed to the mill must be dry at the intermediate diaphragm. If not, severe operating problems may arrive e.g. clogging of diaphragm, bad material flow behaviour etc.

The experience has shown, that material temperature below ~100 [°C] at the intermediate diaphragm reduces the grinding efficiency and the mill production (due to moist material and poor material flow). With such conditions, due to grinding heat of the 2nd compartment, the mill discharge temperature increases and must be cooled down again by water injection in the mill outlet to reach the appropriate product temperature at the mill discharge.

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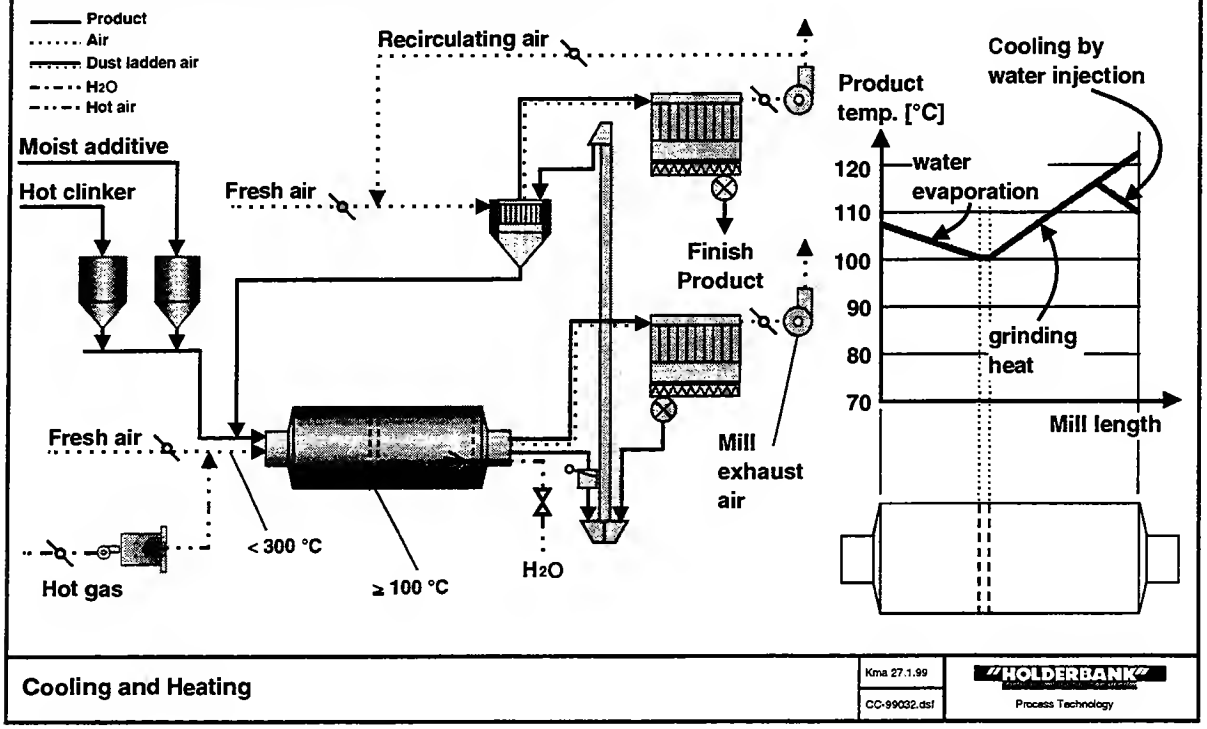
Fig.3 Mill heating for blended cements



In cases of drying needs, all separating air in the separator must be recirculated to keep the maximum temperature of the separator grits returning to the mill inlet and keep the maximum heat within the mill system.

For blended cements with lower moisture contents, drying can take place directly in the first grinding compartment. With higher moisture contents a drying compartment is necessary or the additives must be dried before feeding to the mill. The limit for drying in the grinding compartment is approx. 4 [%] mixed moisture in the mill feed, but depending on the clinker temperature and clinker proportion in the mix, as well as the available hot gas temperature.

The mill inlet temperature with hot gases should not exceed 300 [°C], as above this temperature adverse effects on gypsum dehydration will take place.

Special attention should be paid to dimensioning of the hot gas generator. Many times hot gas generators are sized for the maximum drying needs plus reserve. In these cases the turn down ratio must be such, that also minor moisture contents can correctly be dried with a good and stable operation of the furnace.

Bag filters for dedusting of the mill system must be insulated and cold spots avoided to eliminate the risk of condensation.

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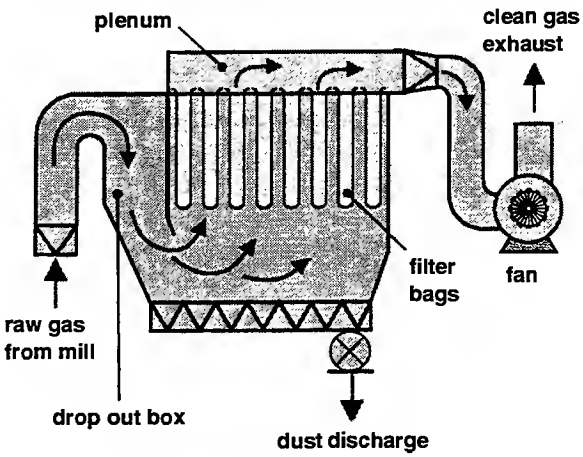
4. FILTER TECHNOLOGY

For dedusting of mills and separators only bag filters are used today, regardless of the fact that ordinary portland or blended cements are produced. Electrostatic precipitators are no more applied due to operational and environmental reasons.

The mill bag filter should only be used for mill ventilation and not additionally dedust other equipment. Also mill dedusting air drawn through the dynamic separator must be avoided due to operational difficulties.

Filter sizing

Fig.4 Filter technology



Filter sizing

Air cloth ratio

80 [m3/m2 x h] without grinding aid
60 [m3/m2 x h] with grinding aid

Bag dimensions

bag Ø 140 - 160 [mm]
bag lengths <4.5 [m]

Filter cloth

- needle felt/polyester - general purpose
- Dolomite
  PEAC - mixed needle felt
- new products
  (e.g.teflon treated fibres)

} - drying/grinding applications

raw gas dust content 300 - 800 [g/m3]
clean gas dust content < 25 [mg/m3]
Δp filter = 5 - 14 [mbar]

Compressed air

≥ 0.13 - 0.19 [m3/h x m2]

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A properly sized and well engineered filter avoids a lot of operating problems and saves costs.

Filter design

There are many different types, makes and philosophies to design and install bag filters. Preferred filters are pulse jet types. It is important to install a properly sized filter with sufficient compressed air. The compressed air must have decent air dryers, specially in tropic countries to avoid contact of moist air with cement and hence clogging of the bags.

In case of higher dew points and installations outside of buildings an insulation of the filter is necessary.

A dust drop-out box can be installed at raw gas dust contents > 300 [g/m^3], however not all suppliers favor such installations.

Walk-in plenums present advantages with regard to tight housings (less false air) and can be installed outdoors without problems.

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Filter operation

Special attention should be paid to the shut down procedure:

- Continue cleaning the filter bags for a couple of cleaning cycles before allowing the temperature to drop below the dew point. As the filter cools, moisture may condense on the bags once the dew point is reached.
- Do not store dust in the hopper. The presence of any moisture will cause it to set, requiring manual work for its removal.
- Allow bags to clean down after dust has stopped entering the hopper, but do not over clean.
- Check to see that all components are in the proper shut down mode.

5. WATER INJECTION

Purpose

Tube mills generate a lot of heat and together with hot clinker increase the product /temperature along the mill axis. Too high material temperatures in the mill may cause

- coating (hampering fluidization of material)
- false set of cement (by dehydration of gypsum to soluble anhydrite and then recrystallization to gypsum)
- cement storage problems in silo (lump formation).

If the cooling by fresh air and radiation/convection is not sufficient, water injection into the mill is necessary. The aim of water injection is to evaporate water and so to extract heat from the material.

Water injection systems

Fig.5 Water injection - systems

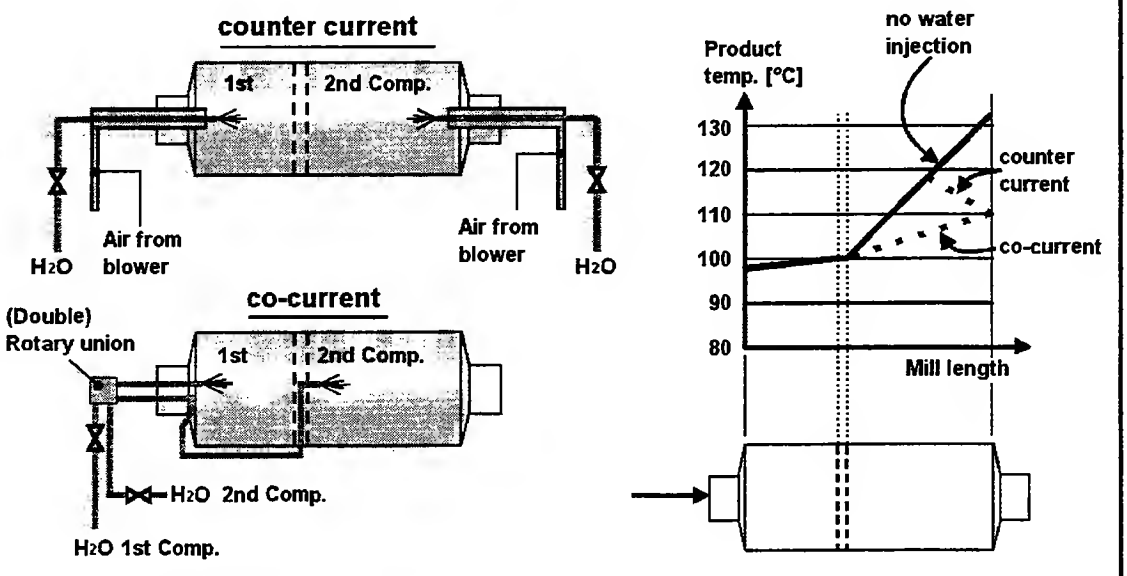

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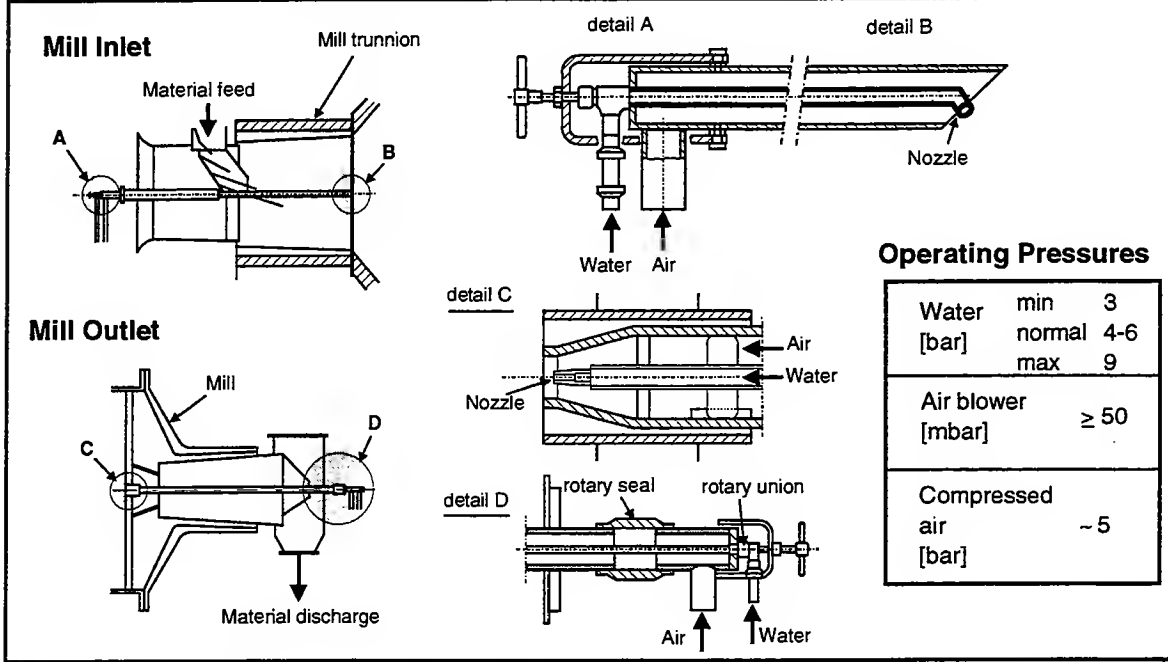
Fig.6 Water injection - devices



Water can be injected into the

- 1st compartment
- 2nd compartment
  - from inlet (co-current)
  - from outlet (counter-current)

The co-current injection into the 2nd compartment requires an installation of the water pipe from the center of the inlet around the shell of the first compartment to the center of the intermediate diaphragm. The installation is more complex but the cooling effect is more favorable, as the temperature increase is slight and the temperature level is lower than for counter-current installations (graph. of Fig. 5). The injection should be in form of a spray rather than a jet to keep the evaporation in the first part of the compartment.

Counter-current injections are easy installations (except with central drives) and cool down rapidly the last part of the compartment. In general, the temperature level over the second compartment is higher than with co-current injections. The injection should be a jet reaching far into the grinding compartment.

Counter-current systems are easy to install in mills with pinion or side drives. With central drives, injection glands around the central drive shaft have to be fitted. Many poor experiences with the tightness of these glands have been made and therefore it is not recommended to adopt such solutions.

If ever possible the water spray nozzles should be surrounded by tubes, carrying air from a blower to keep the nozzle always clean (Fig. 6). Leading compressed air together to the water pipe and to the spray nozzle is an other solution.

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Operation

Every time the cement mill is running or the mill ventilation is on, the cleaning air of the water injection system must be running. After a mill stop, the air cleaning system must continue for a few minutes.

Water is in general first injected into the second compartment and only when the cement temperature at the mill discharge exceeds 100 [°C]. Injection into the first compartment is only allowed when feeding hot clinker (mixed temperature ≥ 100 [°C]). Below these temperatures there is a risk of not sufficient evaporation of the water and clogging of diaphragms.

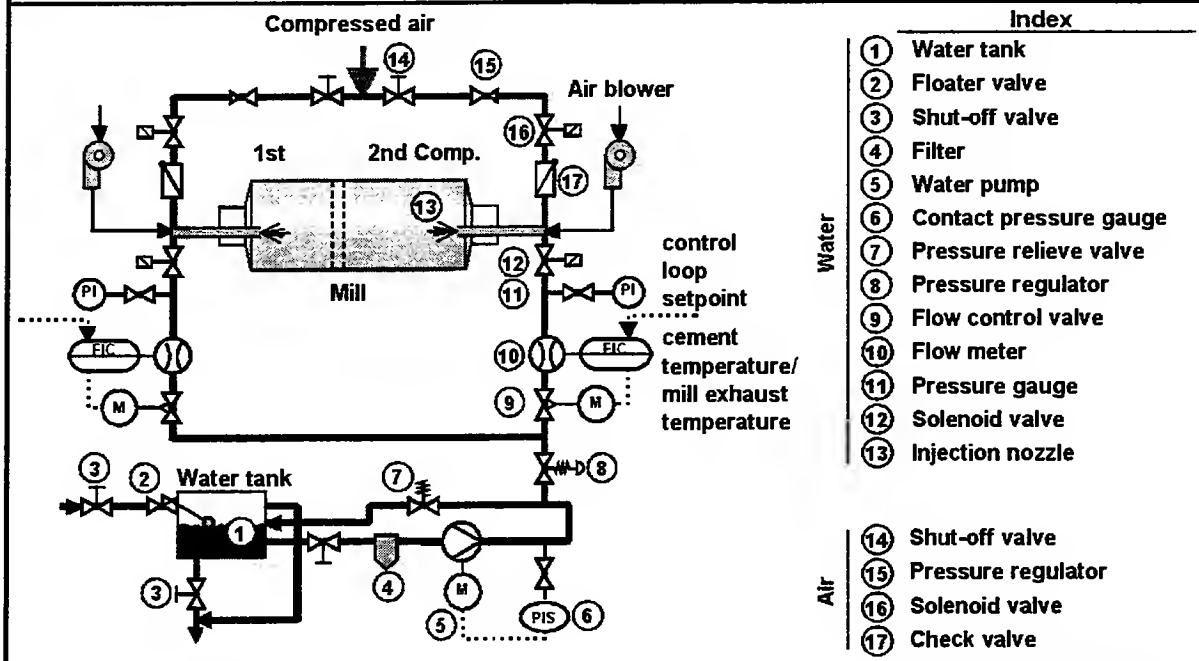
Water injection acts like grinding aid in the mill and fluidizes the material. The control of water injection should therefore always gradually be changed (no steps) to avoid rapid emptying or filling of the mill.

When no water is injected the solenoid valves must be closed and tight to avoid water from dripping into the mill.

In most cases the max. portion of water injected into the first and second compartment is done according to their percentage in length e.g. max 1/3 into the first and 2/3 into the second compartment.

Control

Fig.7 Water injection - control



The water injection is controlled with a control loop in function of the clinker or cement temperature.

In the first compartment either the clinker temperature of the mill feed or the cement temperature at the intermediate diaphragm are the process values measured.

In the second compartment either the cement (mill discharge) or the exhaust gas/air temperature can be taken. The exhaust air temperature is normally min. 5 [°C] below the cement temperature, depending on the false air rate.

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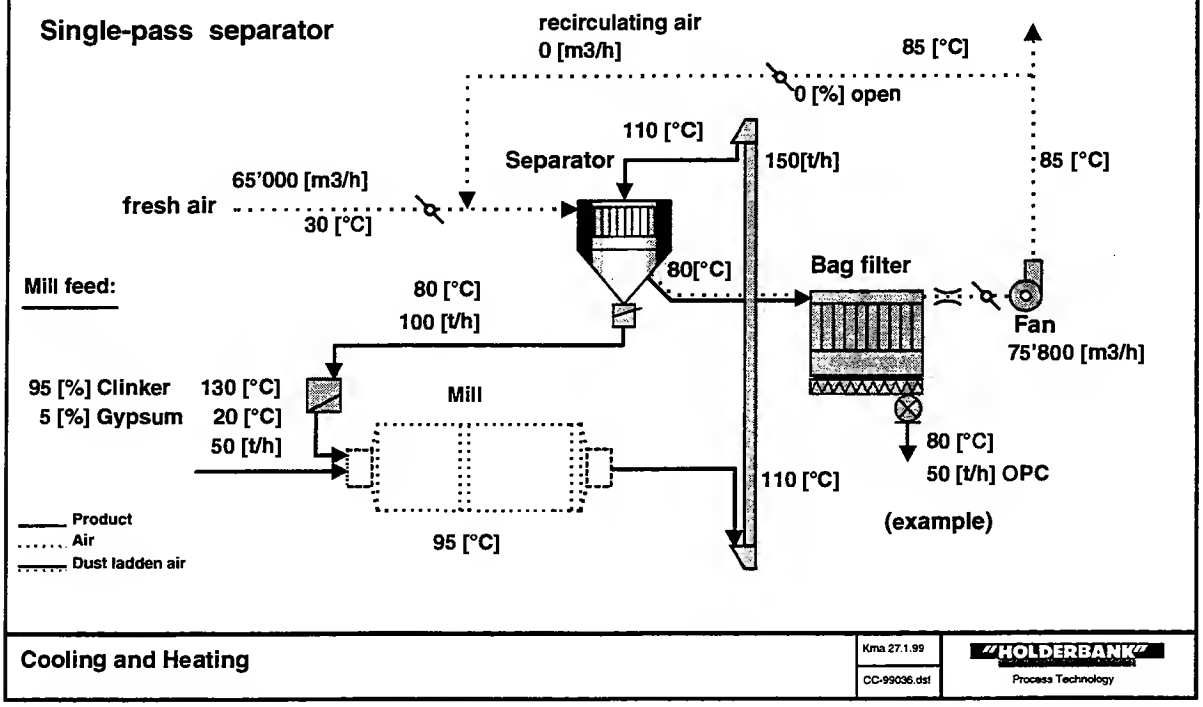
The control loop consists of a water flow meter which value is compared to the set point of the controller (based on measured temperature). Any change is transmitted the water flow control valve for a corresponding adjustment.

6. COOLING IN THE SEPARATOR

Fig.8 Cooling in the separator



Cooling of the cement, finish product and separator grits, can be done in the separator, in case the separator system allows for.

New separators can be fitted with bag filters as single pass separators. The air passes through the separator, filter and fan. The clean fan exhaust air can be blown to the atmosphere or recirculated to the separator inlet. Fresh air substitutes the exhausted air and does the cooling in the separator. The more fresh air with ambient temperature is used the higher is the cooling effect. The more air is recirculated the lower the cooling.

At least 5 [%] of the total separating air is always fresh air in form of false air inleakage. The cooling level is adjusted by the recirculating and the fresh air flap.

A cold mill should always be started up with a maximum amount of recirculating air to heat up the mill system. As soon as the operating temperatures of the mill system are reached then fresh air can be used to cool down the material to the target temperatures.

A maximum cement cooling capacity of a separator, cooling with fresh air is around 25 – 30 [°C]. For higher cooling needs, additional cooling facilities e.g. cement coolers have to be used.

Besides cooling the finish cement to the silo, this cooling method also cools down the separator grits returning to the mill inlet. Cool separator grits have on one hand a cooling effect in the first mill compartment but can on the other hand cool down too much. At too low material temperatures at the intermediate mill diaphragm the mill production is reduced. In this case the temperature level of the separator grits must be increased again by recirculating more warm separating air back to the separator inlet.

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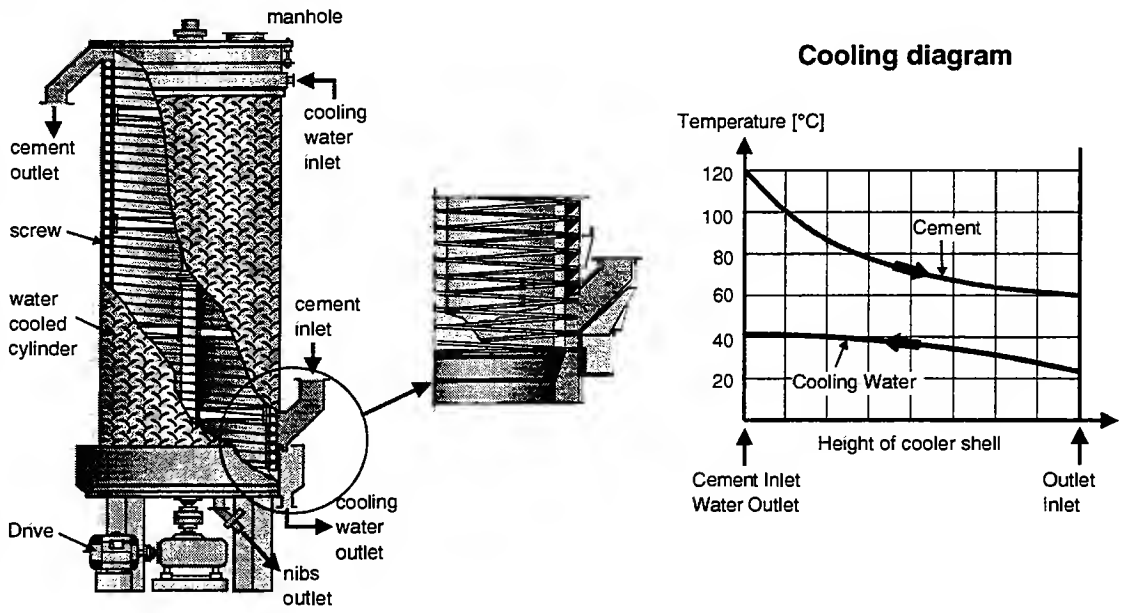
In any case a detailed heat balance must be calculated to give the necessary information for the prevailing situation and the cooling/heating needs.

Fig. 8 shows a tube mill system with a singlepass separator. The example shows the cooling effect at maximum fresh air amount used at the separator inlet. As a result, the material temperature at the intermediate diaphragm is low and leads to reduced mill production rate.

7. CEMENT COOLERS

Fig.9 Cement coolers



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Purpose

Cylindrical cement coolers are efficient and low energy consuming solutions to cool down the cement after the grinding process. Reasons for cooling may be

- customer requirement
- cement bagging and bag handling at low temperature
- avoid lump formation in the cement silos (<70 [°C]: reduction of water release from gypsum dehydration)

Principle

Water cascades in a thin layer over a cylindrical body from the top to the bottom and cools down the cooler shell. Hot cement is introduced to a type of screw conveyor at the bottom and conveyed up along the cooler shell to the discharge at the top. Intense heat transfer occurs from the cement powder at the inside of the shell, through the shell wall to the water layer at the outside. The cooling occurs in counter current, that means the best cooling is done with the lower water temperature at the cement discharge at the top of the cylinder.

The cooling water recirculates in a closed water circuit. A water cooler reduces the water temperature before feeding it again to the top of the cylinder.

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Performance

- Cement coolers exist up to 200 [t/h] for a single cylinder.
- The cylinder size varies with the cement throughput and the cooling capacity.
- Cooling the cement from 120 [°C] down to ~60 [°C] is common.
- The spec. cooling rate is approx. 1.25 [kJ/m °C h].
- Typically the cooing water is heated up from 25 – 40 [°C] along the cylinder.
- Spec. electrical energy consumption of cooler and water treatment plant is approx. 1 [kWh/t] cement.
- Increasing ovality of the cylindrical shell decreases the efficiency. A minimum clearance between screw flights and shell has to be aimed at.

Water treatment

- The water must have an acceptable quality, often the water must be softened by decalcifying units to avoid lime furring on the cooling surfaces.
- Lime furring can also be reduced through a cathodic protection of the shell.
- The water consumption in a closed water circuit refers only to the make-up water due to the losses in the water cooling units.

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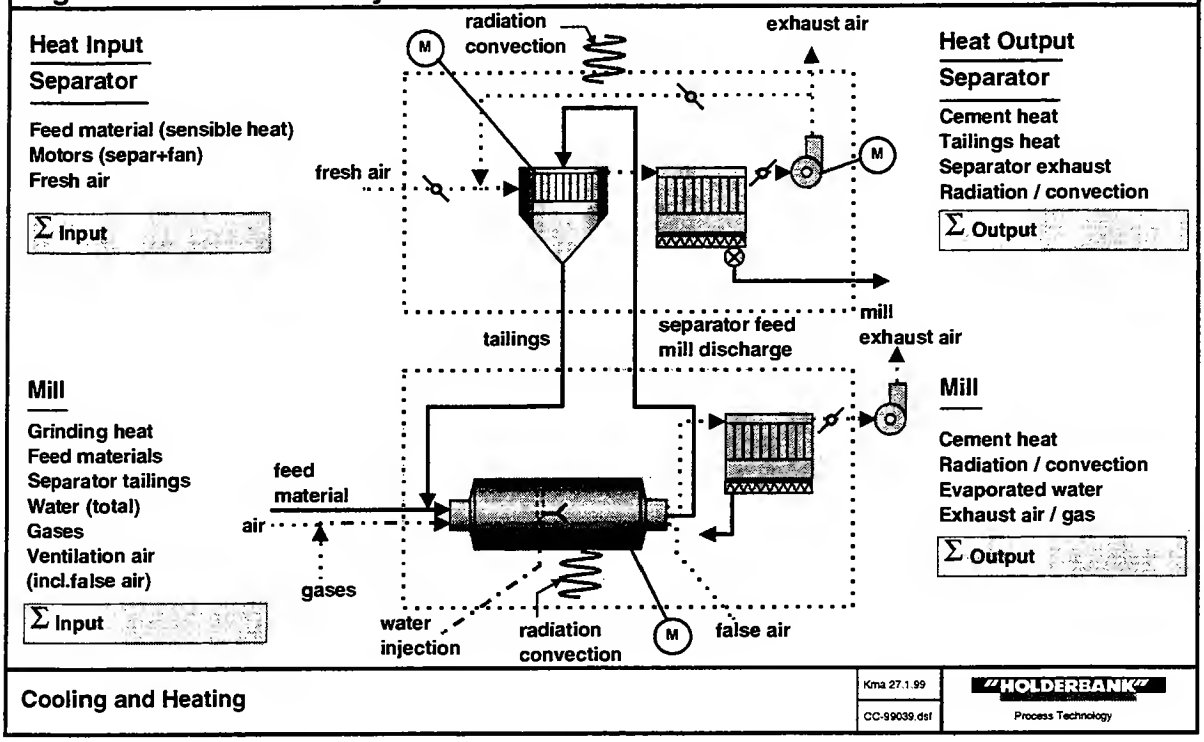
8. HEAT BALANCE (ENERGY BALANCE)

Fig.10 Heat balance - system



If ever cooling or heating needs must be calculated, the appropriate model is a heat balance calculation. The aim of such a calculation is to check the temperatures at the critical points and the quantities of cooling air, water or hot gases needed to fulfil the requirements.

In principle all heat inputs (energy) have to be equal to the heat outputs of the system in question.

Heatbalance:
Σ Inputs = Σ Outputs

Procedure

1) Define borders of the system(s); for cement grinding systems it is recommended to define two systems, separator and mill part.
2) Define inputs and outputs for each system.
3) Compute the balances based on

Heat	inputs	=	outputs	[kJ/kg product]
Air/gas flows	inputs	=	outputs	[Nm^3/h]

4) Observe the restrictions

- max. gas/air speeds in mill, mill inlet/outlet
- max. gas temperature at mill inlet (available and recommended)
- max. possible water injection rate for 1st and 2nd compartment

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- plant altitude (density of air)

Hints

- The balance can be equally done for any grinding equipment (tube mill, vertical mill, roller press, horizontal mill etc.) based on the same principle and adapting the systems according to the features of the equipment.
- The heat is always related to 1 [kg] of finish cement at the system outlet.
- The heat balance is based on the reference temperature of 20 [°C], therefore the Δt is always (t-20) and negative heats are possible.
- All grinding heats of the major electrical consumers are referred to their motor shafts and have to be converted to the mechanical power absorbed of this equipment if necessary.
- The evaporation heat of water consists of heating up the water to the operating temperature an the subsequent portion of evaporation energy.
- The radiation/convection losses are often assumed and are related to an estimated surface (e.g. mill surface times factor x) at an estimated surface temperature (e.g. t-10 [°C]).

Heat balance example

Fig. 11 Heat Balance Cement Mill Example "HOLDERBANK"

Heat Balance Mill:							
Grinding heat	$P_{abs} * \eta * 3.6 * 1000$	/G * 1000	5000 * 0.91 * 3.6 * 1000 /(180 * 1000)	91.0	[kJ/kg]	33.4	[%]
Separator tailings	G * (u-1)) * cp * (t-20) * 1000	/G * 1000	180 * (2-1) * 0.819 * (104-20) /(180 * 1000)	68.8	[kJ/kg]	25.2	[%]
Feed material heat	G * cp * (t-20)	/G * 1000	180 * 0.806 * (110-20) /(180 * 1000)	73.2	[kJ/kg]	26.9	[%]
Water (feed + injection)	$W_{inj} * 4.2 * (t-20)$	/G * 1000	5410 * 4.2 * (15-20) /(180 * 1000)	-0.7	[kJ/kg]	-0.2	[%]
Gases	$V_g * cp * (t-20)$	/G * 1000	11'000 * 1.529 * (450-20) /(180 * 1000)	40.2	[kJ/kg]	14.8	[%]
Air (including false air)	$V_a * cp * (t-20)$	/G * 1000	15'290 * 1.3 * (20-20) /(180 * 1000)	0.0	[kJ/kg]	0.0	[%]
INPUT :				**272.5**	**[kJ/kg]**	**100**	**[%]**
Cement heat ex mill	G * u * cp * (t-20) * 1000	/G * 1000	180 * 2 * 0.822 * (110-20) /(180 * 1000)	148.0	[kJ/kg]	54.3	[%]
Radiation / convection	k * A * (t-20) * 3.6	/G * 1000	14.15 * 1030 * (110-20) * 3.6 /(180 * 1000)	26.2	[kJ/kg]	9.6	[%]
Water (evaporated)	$W_{ev} * q$	/G * 1000	5'410 * (2501 + (1.88 * 105)) /(180 * 1000)	81.1	[kJ/kg]	29.8	[%]
Exhaust air /gas	$V_{ex} * cp * (t-20)$	/G * 1000	26'850 * 1.358 * (105-20) /(180 * 1000)	17.2	[kJ/kg]	6.3	[%]
OUTPUT :				**272.5**	**[kJ/kg]**	**100**	**[%]**

Heat Balance Separator:							
Separator Feed	G * u * cp * (t-20) * 1000	/G * 1000	180 *2 * 0.822 * (110-20) * 1000 /(180 * 1000)	148.0	[kJ/kg]	97.0	[%]
Motors separator + fan	$P_{abs} * \eta * 3.6 * 1000$	/G * 1000	282 * 0.8 * 3.6 /(180 * 1000)	4.5	[kJ/kg]	3.0	[%]
Separating air (fresh)	$V_s * 1.3 * (t-20)$	/G * 1000	3'032 * 1.3 * (20-20) /(180 * 1000)	0.0	[kJ/kg]	0.0	[%]
INPUT :				**152.5**	**[kJ/kg]**	**100**	**[%]**
Tailings	G * (u-1) * cp * (t-20) * 1000	/G * 1000	180 * (2-1) * 0.819 * (104-20) /(180 * 1000)	68.8	[kJ/kg]	45.1	[%]
Radiation / convection	k * A *(t-20) * 3.6	/G * 1000	14.15 * 550 * (104-20) * 3.6 /(180 * 1000)	13.1	[kJ/kg]	8.6	[%]
System exhaust air	$V_{sav} * cp * (t-20)$	/G * 1000	3'032 * 1.3 * (104-20) /(180 * 1000)	1.8	[kJ/kg]	1.2	[%]
Cement heat	G * cp * (t-20) * 1000	/G * 1000	180 * 0.819 * (104-20) /(180 * 1000)	68.8	[kJ/kg]	45.1	[%]
OUTPUT :				**152.5**	**[kJ/kg]**	**100**	**[%]**

Heat of evaporation $q = 2501 + (1.88 * t)$ [kJ/kg H_2O] heat of evaporation at exhaust temperature
Radiation / convection factor $k = ((\Delta t_{surface}-10) * 0.055) + 9.75$ [kJ/m² °C h]
Balance related to : 1 [kg] of cement
Reference temperature 20 [°C]

Cement grinding system

System:	Tube mill with separator				
	Mill diameter		4.6	[m]	
	Mill length		15.5	[m]	
	Motor power		5000	[kW] abs.	
[%]	Overall drive eff.		91	[%]	
Operating data:					
	Production		180	[t/h]	Pozzolana cement
	Composition	Clinker	71	[%]	gypsum
		Gypsum	4	[%]	4 [%] H_2O
		Pozzolana	25	[%]	8 [%] H_2O
	Clinker temperature		150	[°C]	
	Ambient temperature		20	[°C]	
	Water temperature		15	[°C]	
	Hot gas temperature		450	[°C]	
	Mill inlet temperature		~200	[°C]	
	Mill discharge temp. (cement)		110	[°C]	
	Mill discharge temp. (gas)		105	[°C]	
	Finish product temperature		104	[°C]	
	Mill fan flow rate		33000	[Nm^3/h]	

Index

G	[t/h]			mill production
u	[-]			circulation factor (feed/product)
t	[°C]			Temperature
$P_{abs.}$	[kW]			absorbed power (counter)
η	[-]			efficiency of drives
cp	[kJ/kg]			spec. heat value (related to reference temperature)
	[kJ/Nm^3]			
W	[l/h]			water to be evaporated
		index:	in	moisture in feed material (incl. water injected)
			ev	evaporated water
V	[Nm^3/h]			air/gas flow rate
		index:	g	gas
			a	fresh air
			ex	exhaust air
			s	fresh air separator
k	[kJ/m^2 °C h]			radiation/convection factor
A	[m^2]			radiation surface
q	[kJ/kg H_2O]			heat of evaporation

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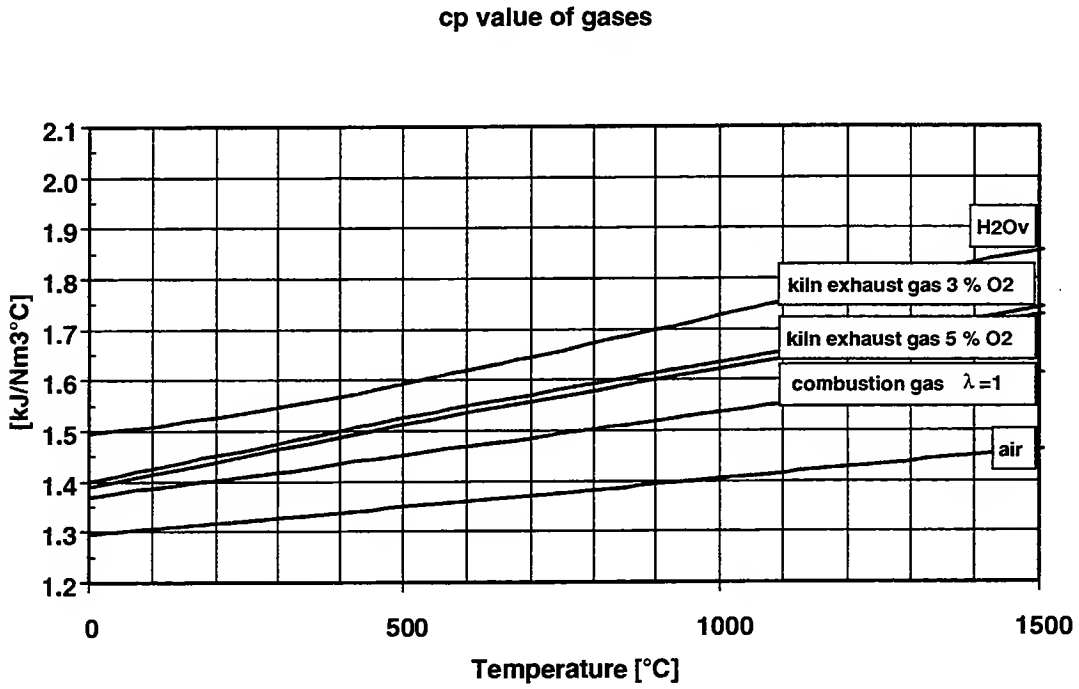
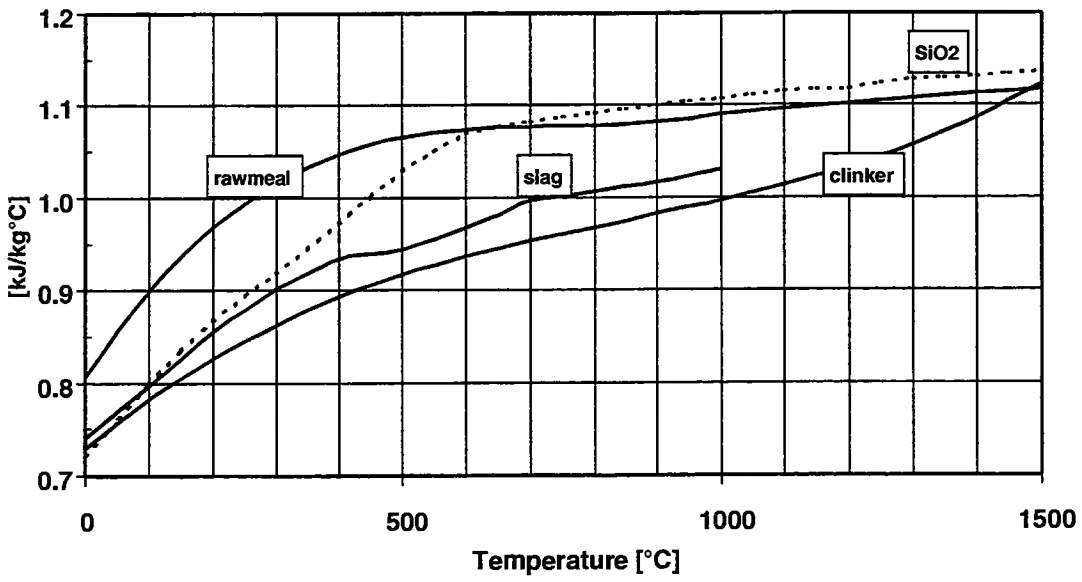
The cp-values are given in Fig. 12.

For the additives pozzolana, fly ash and gypsum the cp-values of raw material can be used.

Fig. 12





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Manufacturing of Blended Cement

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Summary

The additive components of blended cements (or masonry cements) are mostly ground together with clinker in a tube mill (compound grinding). The product specifications (fineness, homogeneity) can thus be fulfilled without major problems. Disadvantages are high power consumption and limited drying capacity within the mill.

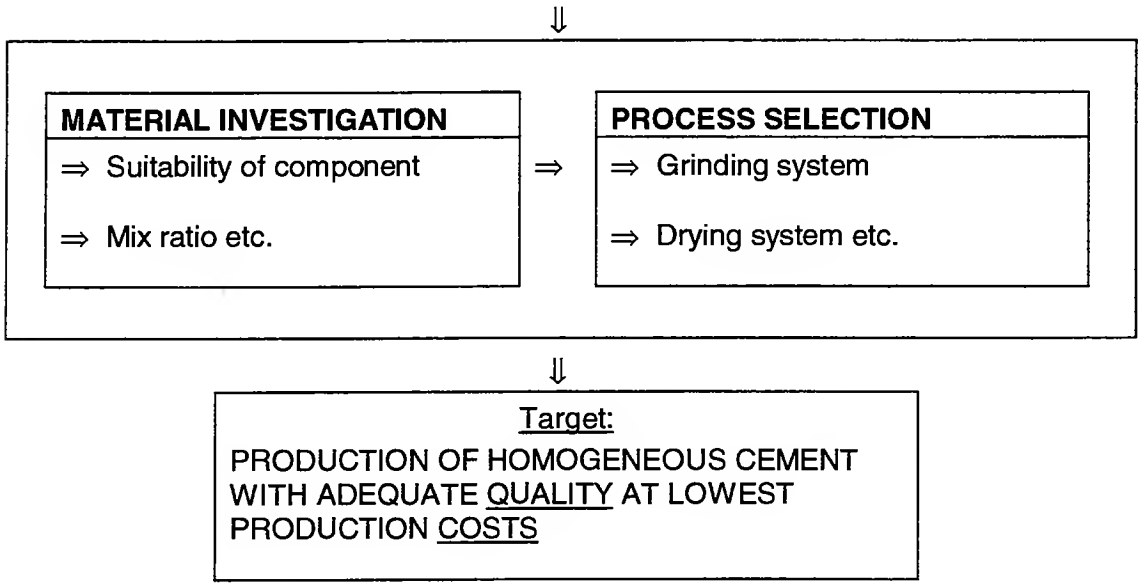
To overcome these disadvantages the additive components can be ground separately with subsequent proportioning and blending of the components. Grinding as well as drying represents no problem. However, attention has to be paid to the selection of the proportioning devices and the blending procedures to meet the standard specifications.

Small capacities are often produced in compound grinding whereas separate grinding should be preferred for large capacities.

1. INTRODUCTION

Prior to any manufacturing of blended cements a detailed investigation of the components to be blended with cement is mandatory.

Figure 1 Manufacturing of Blended Cement and Masonry Cement



The material investigations shall give some indications with regard to the suitability and the possible mix ratio of the additive component. These results and further evaluations of the properties of the additive component and the final product shall serve as a basis for the process selection. The material and process investigations have the same common target:

'Production of a homogenous cement with an adequate quality at lowest production costs.'

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2. PROCESS TECHNOLOGICAL PROPERTIES

The main task of the manufacturing of blended cement is to replace clinker by other hydraulic active materials.

Figure 2 Process Technological Properties of Blended Cement and Masonry Cement

Type of cement	BLENDED CEMENT			MASONRY CEMENT
Type of blend component	Synthetic pozzolan		Natural pozzolan	Limestone Dolomite
	Slag	Fly ash	Pozzolan Tuff etc.	Marl Raw meal etc.
Chemical properties	same as Portland-cement			qualitatively not comparable
Mix. proport. (%)	20 - 70	10 - 30	10 - 30	30 - 70
Grindability (kWh/t)	30 - 40	5 - 20	20 - 30	5 - 30
Moisture cont. (% H_2O)	0 - 20	0 - 35	5 - 15	0 - 20
Process aspects	directly ground and dried without further preparation		exploitation and/or crushing prior to grinding - drying required	

Hydraulic active materials are synthetic pozzolan (such as slag and fly ash) and natural pozzolan. From the material technological point of view a strict difference has to be made between blended cements and the so called masonry cements. The chemical properties of blended cements are comparable to those of ordinary Portland cement whereas masonry cements have due to their quite specific application not the same product properties.

As far as the process technological aspects are concerned the additive components used for the blended cements and masonry cements can be subdivided into materials which can without further preparation directly be ground to blended cements and into materials which require prior to grinding a further processing such as exploitation and/or crushing. The first group includes the cements containing industrial by-products such as slag cement and fly ash cement whereas the second group includes the pozzolan cement and masonry cement.

Nevertheless, the manufacturing processes of both blended cements and masonry cement are basically connected with the same problems; i.e. the relevant process technology know-how of blended cements are within some limits also valid for the masonry cement preparation.

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3. MANUFACTURING POSSIBILITIES

3.1 General

As it can be concluded from the rather different properties of the additive components there seems to be a large number of possible manufacturing possibilities.

It is not the target of this presentation to describe the alternatives in every detail but to summarize the outcome of an evaluation of a questionnaire concerning the manufacturing of blended cements. Considering these plant experience and furthermore taking into account the result of different laboratory tests carried out at HMC/TC the manufacturing of blended cements can roughly be subdivided into two methods; which are

- compound grinding of the additive components in one mill and
- separate grinding of both, the additive component and the clinker with subsequent blending

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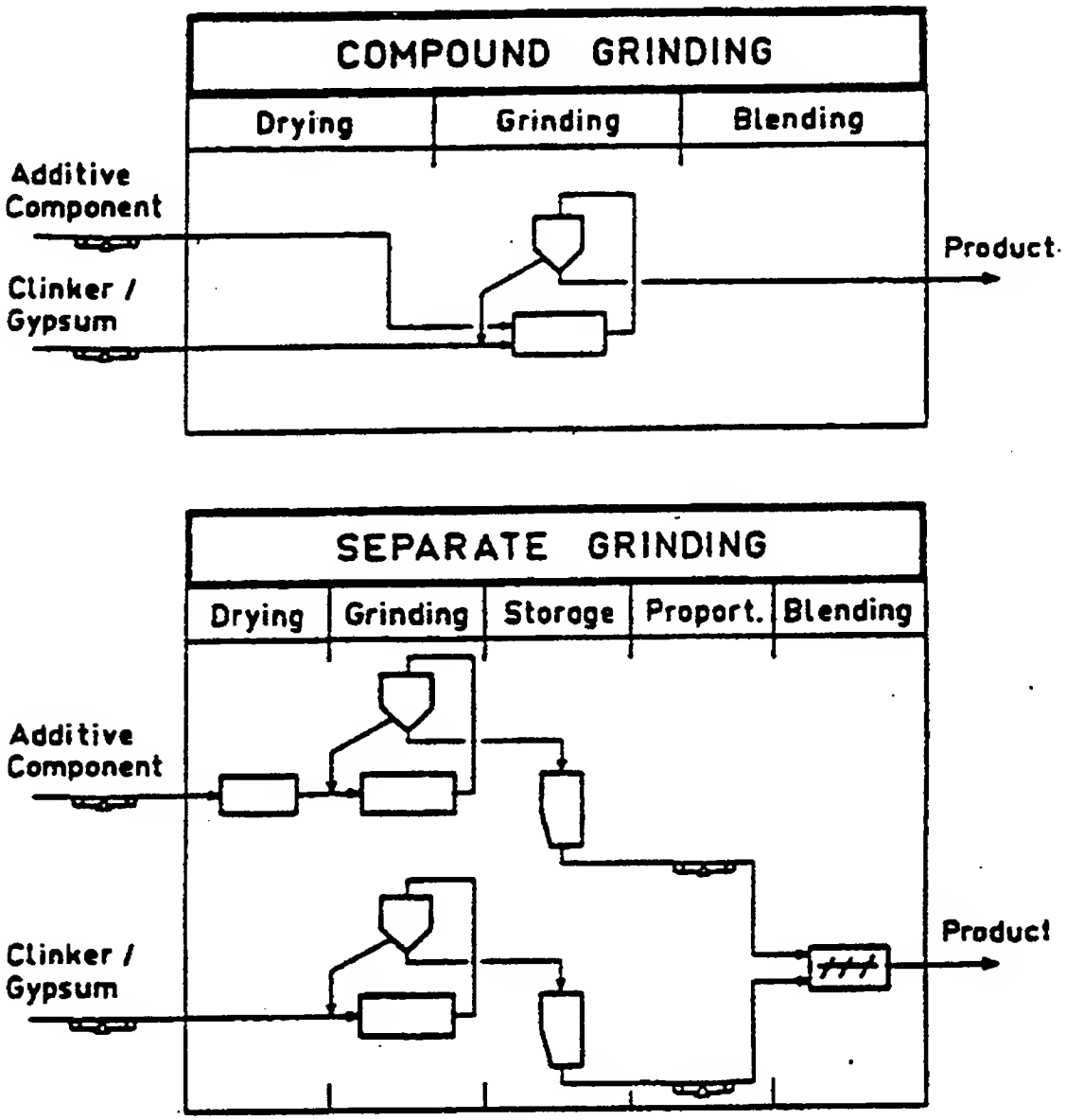
Figure 3 Manufacturing Possibilities for Blended and Masonry Cement



In the compound grinding method the components are ground and blended in one single piece of equipment. Depending on the moisture content of the additive component often also drying can be effected in the mill. Most of the blended cements and masonry cements in the "Holderbank" Group plants are produced by compound grinding.

If the additive components have a very different grindability as compared to clinker or in case of elevated moisture contents of the component a separate grinding is often more advantageous. The component can either be dried separately or it can be dried in a grinding-cum-drying system. Separate grinding requires in addition to the grinding and the drying system also intermediate storage, proportioning and blending facilities. Considering the larger number of equipment required in case of separate grinding, this method is often only feasible if large quantities of blended cements or masonry cements have to be produced.

Each process step as well as the selection of the relevant equipment has to be carefully evaluated by due consideration of the prevailing properties of the additive components and

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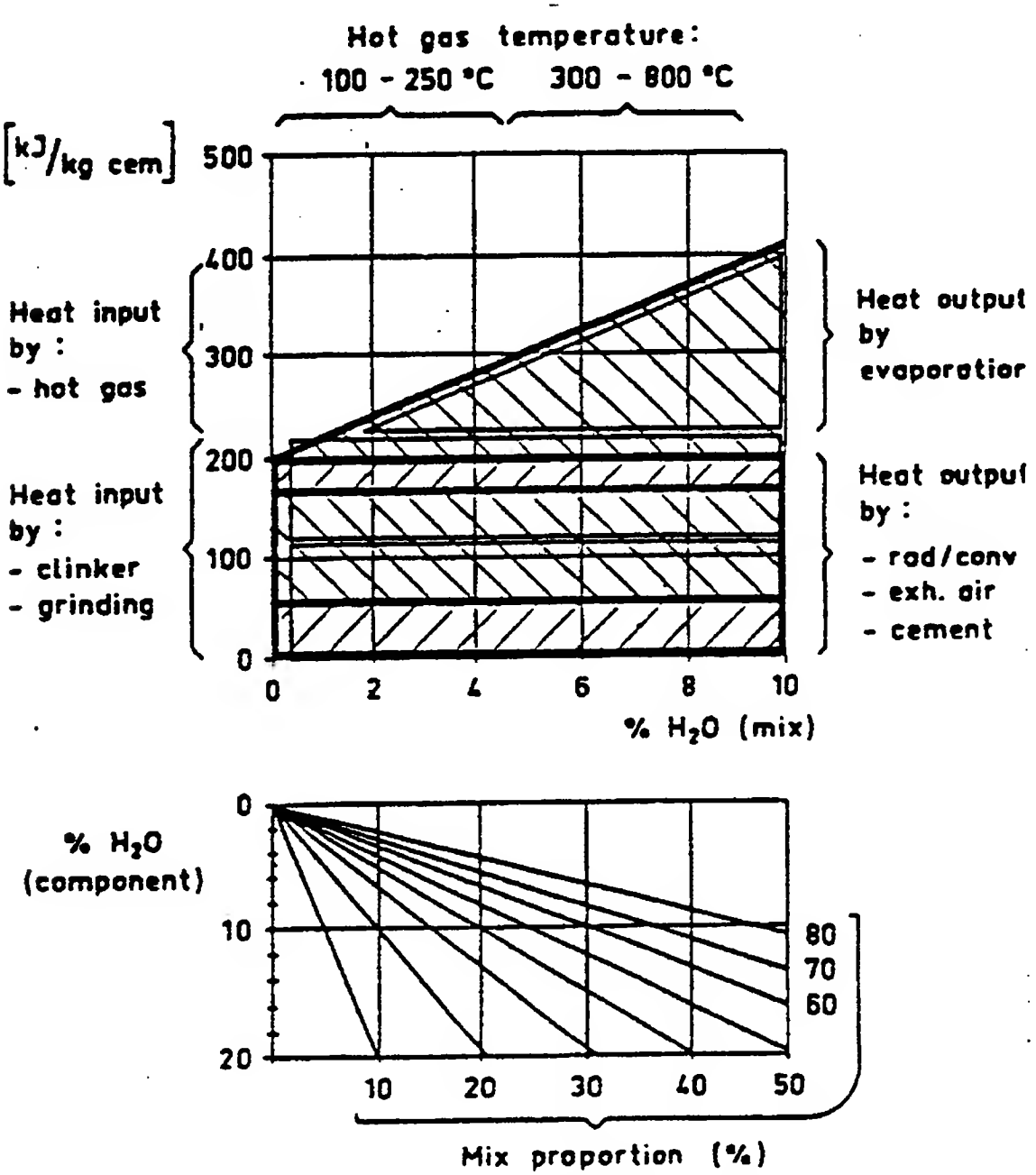
the required product specifications. The most important criteria concerning drying, grinding, storage, proportioning and blending are dealt with in the following chapters.

3.2 Drying

Most of the additive components have a high moisture content so that they have to be dried. The decision whether the component can be dried inside the mill or if a separate drying plant is required depends basically on the prevailing moisture content and the mix ratio of the additive component. To judge the limits of drying inside a mill a heat balance has to be elaborated.

Figure 4 Drying-cum-Grinding of Blended Cement



The simplified heat balance as shown in figure 4 is based on the following assumptions:

- Grindability of clinker and additive component more or less identical

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- Clinker temperature: 130 - 140 ° C
- Cement temperature: 100 ° C
- Mill exit air temperature: 90 ° C
- Air speed inside mill: 2 m/s

The heat balance assumes that the heat output by radiation, convection, cement and mill exhaust air remains constant independent of the relevant moisture content of the mixture. The heat output by evaporation is in turn directly related to the moisture content.

As it can be seen in figure 4 the total heat required for drying is only partially covered by the heat input by hot clinker and by grinding heat; i.e. the additional heat has to be provided by hot gases.

For a moisture content of the mixture of up to 4.0 % a hot gas temperature of 100 - 250 ° C is required. The hot gases can (if available) taken from a grate cooler. At a higher moisture content of the mixture the required hot gas has to be supplied by an auxiliary furnace with a temperature in a range of 300 - 800 ° C. Due to the potential risk of dehydration of the gypsum at these elevated temperatures a drying of the component in the mill is therefore in such a case often not recommendable.

If a substitution of the gypsum by anhydrite is not possible there are basically two possibilities to avoid the risk of dehydration; reduce the mix proportion to such a level that a low drying gas temperature is sufficient or separate drying of the additive component outside of the mill. Whether the first or second measure should be applied depends on the investment costs of a separate drying plant compared with the increase of the production costs due to an increase of the portion of clinker to be used.

If no hot gases at all and no separate drying facilities are available the mix proportion has consequently to be reduced to maintain a moisture content of the mixture which can be dried in the mill by the clinker heat and by the grinding heat only.

It is obvious that the conclusions as derived from figure 4 can not directly be transferred to every type of blended cement manufacturing. A heat balance has to be established for each case taking into account the relevant material properties as well as type and size of the mill equipment.

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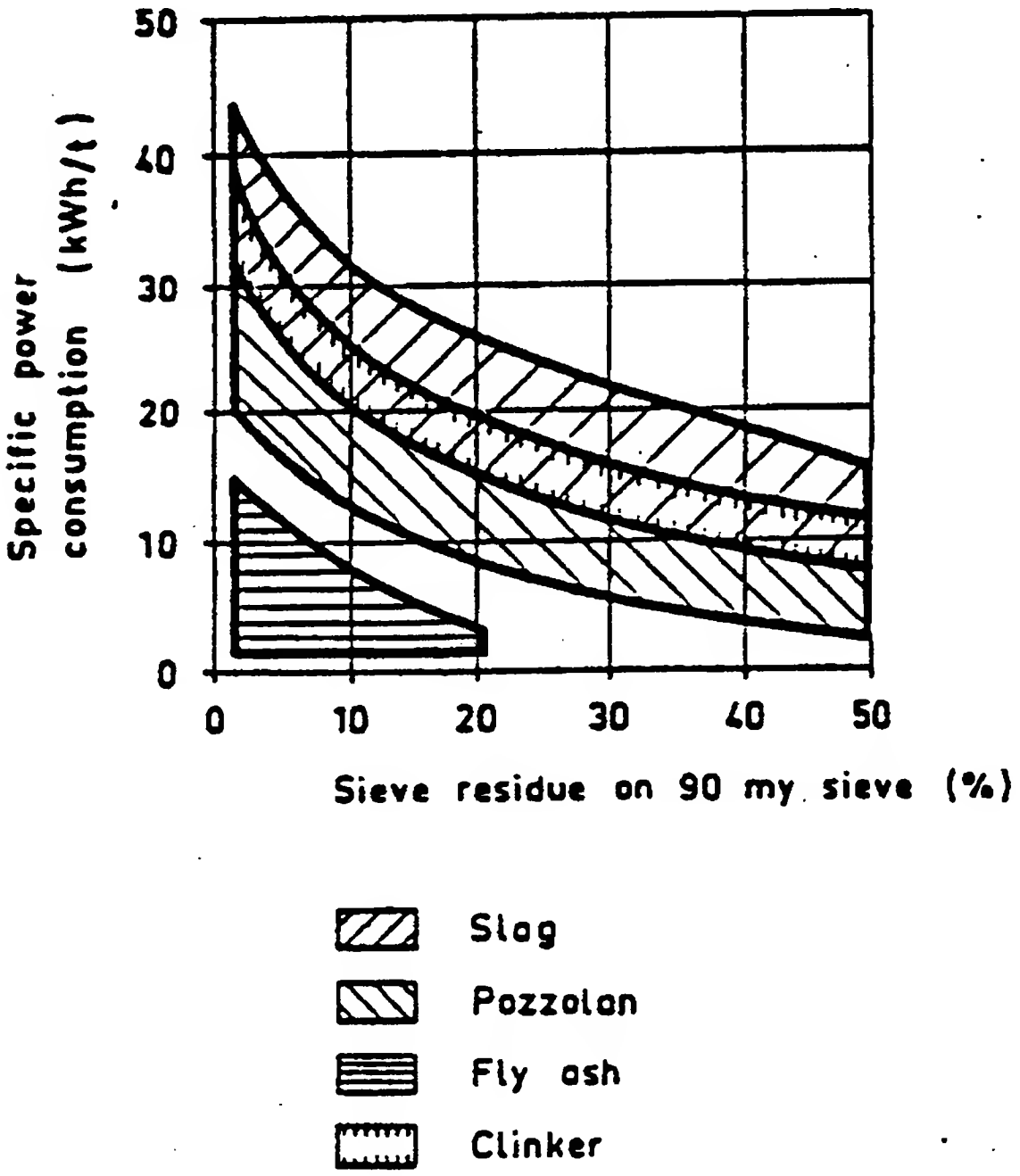
3.3 Grinding

The grindability of the individual components is of great importance for the decision which type of mill and which preparation system has to be selected.

The evaluation of the grindability tests are carried out at the HMC/TC laboratories shows considerable differences between the grindability of the individual components (figure 5).

Figure 5 **Specific Power Consumption (Test Mill) of Mix Components of Blended Cement**



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Slag shows as compared with clinker generally a poorer grindability whereas the pozzolan shows a better grindability than the clinker. The grindability of the fly ash can be judged as very good.

The grindability of the additive component alone is, however, not a definite indication about the reaction in the mill when they are ground together with clinker. No specific grinding problems have to be expected, if the grindability of the slag or of the pozzolan is in the same order of magnitude as the clinker. As soon as the grindability of the additive component is very different from the clinker an overgrinding of the easier grindable component has to be expected. This could lead to disturbances of the grinding operation and therefore result in a capacity drop and in an increase of the specific power consumption.

To avoid these disadvantages it is often recommended to grind components with widely differing grindabililty in two separate mills. The ball charge composition of each mill could in this case be optimally adjusted to the relevant additive component.

Fly ashes are due to their very fine granulometry not directly interground with the clinker but they are fed to the separator. The coarse particles of the fly ash are recirculated to the mill whereas the fine fraction is going into the final product. The coarse fraction of the fly ash acts depending on its carbon content also as a grinding aid; i.e. with the addition of fly ashes the mill output can often be increased.

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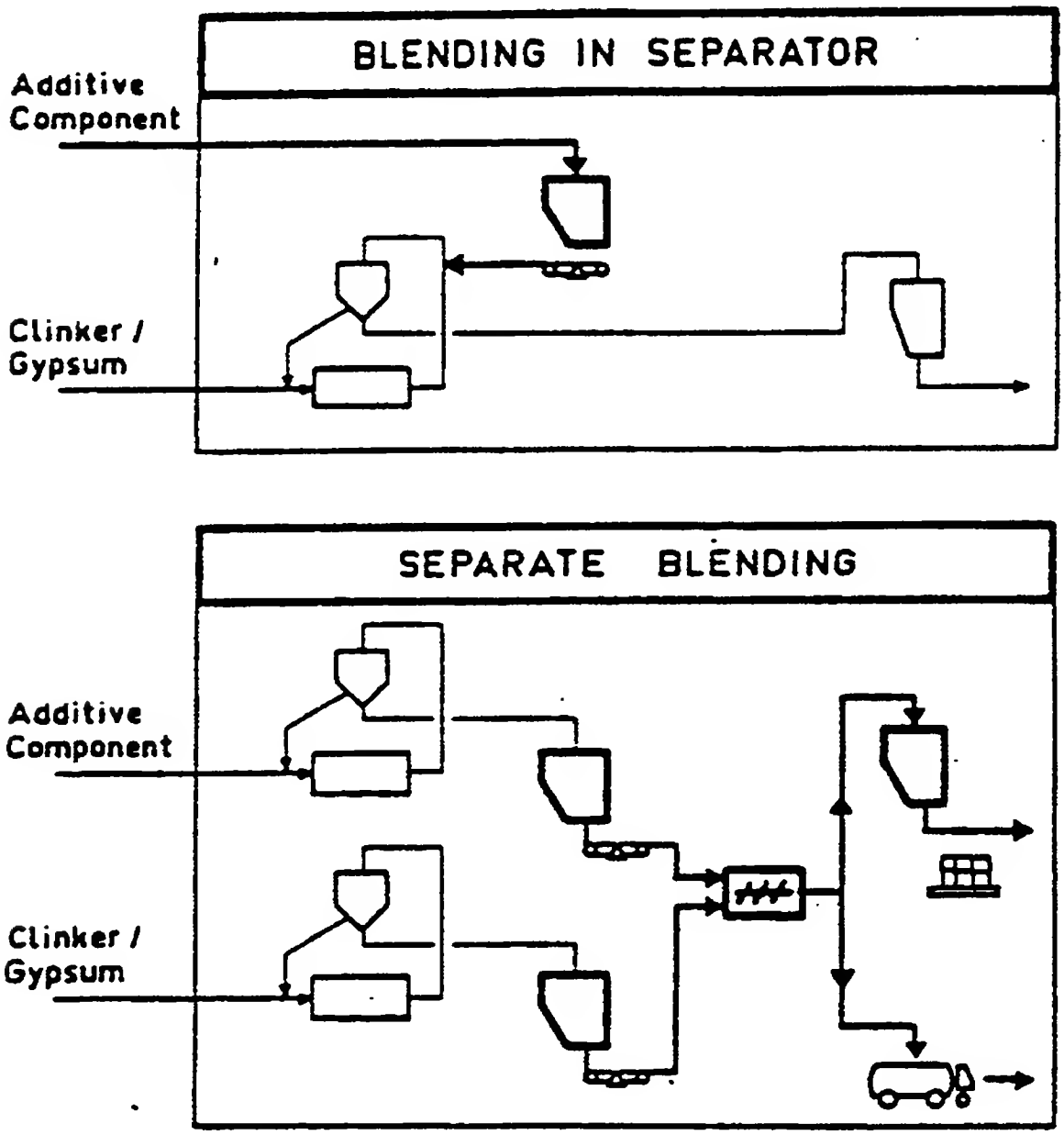
3.4 Storage, Proportioning and Blending

If the additive component is supplied in a very fine granulometry (e.g. fly ash) or in case of separate grinding, the component is very often fed to the separator where it is blended with the cement.

Figure 6 Storage, Proportioning and Blending of Blended Cement



Experience has shown that for a mix proportion of the additive of up to 30 % no overload of the separator has to be expected.

If the portion of the additive component is higher than 30 % the blending of the component has to be carried out in separate blending devices, such as forced mixers, mix screws, homogenizing silos, etc.

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For a separate blending of the components, they have to be stored in intermediate storage silos. The flow properties of the dried and ground materials lead often to the risk of uncontrolled material rushes and discharge problems.

To prevent material bridges or to avoid a piping effect where only material in the middle is discharged it is advantageous to design the silos for mass-flow.

As far as the proportioning of the components is concerned it has to be emphasized that an adequate proportioning is much more important than the subsequent blending process. The best blending device is not able to compensate proportioning errors.

For small quantities volumetric feeders such as rotary feeders and proportioning screws are often sufficient. If a high feeding accuracy is required it is recommendable to install gravimetric feeders such as weigh feeders or impact flow meters.

Depending on the local conditions either batch blending or continuous blending is used. Batch blending can for example be used where the blended cement is loaded directly into bulk carrier vehicles. Both blending methods can be used if the blended cements are stored in silos.

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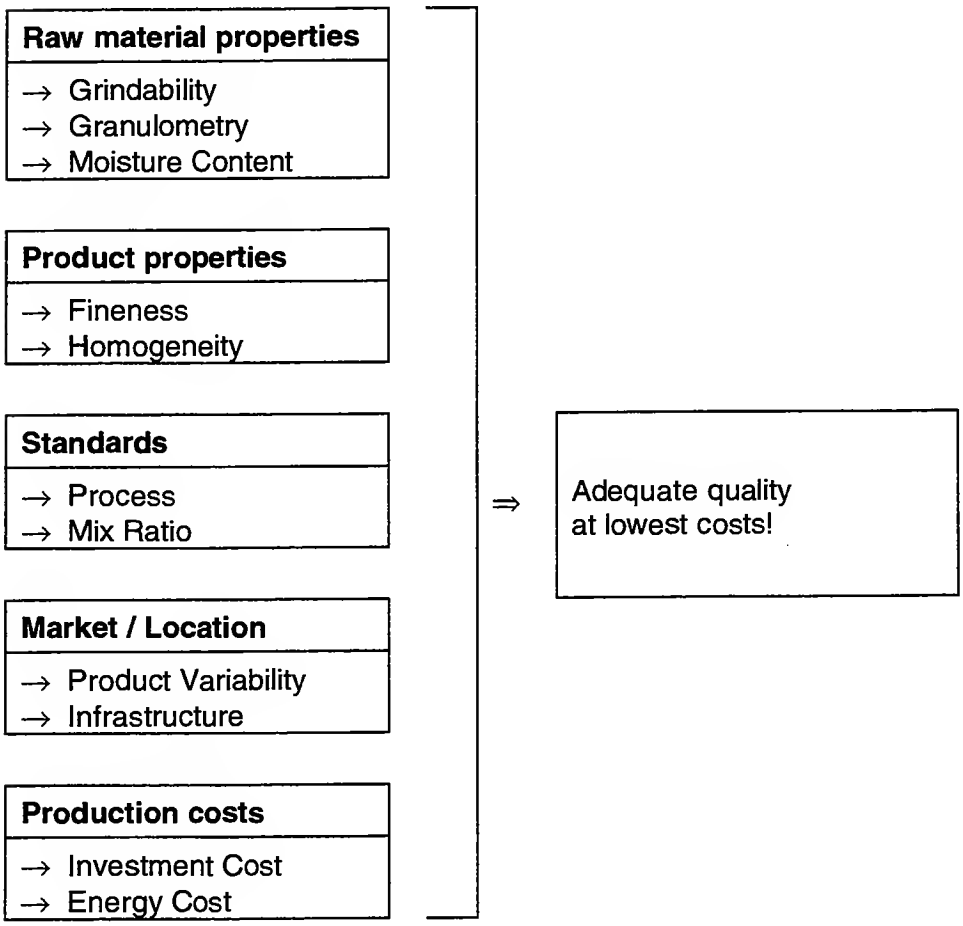
4. FURTHER SELECTION CRITERIA

As mentioned previously the selection of the grinding, drying and blending method is principally dictated by the raw material properties and the product specifications.

Further to these process technological criteria, however, also the relevant standards and regulations, the market situation, the plant location and last but not least also the production costs have to be considered.

Figure 7 Compilation of Process Selection Criteria



In some countries there are standards which specify the manufacturing of blended cements by compound grinding only.

If there are several customers who want different compositions of blended cements this demand can often only be satisfied by separate grinding and separate storage of the components.

The location of the cement plant and the location of the available additive component is of course also of influence for the selection of the preparation system. It is quite possible that the two locations to be far apart so that the individual components can for transport reasons only be transported whey they are already ground, which means that the components would have to be ground separately.

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If the blended cement accounts for only a small percentage of the total production of the grinding system the blended cement can often be manufactured by compound grinding of the components in an existing mill. Any higher costs of production incurred in this way in comparison to separate grinding are almost negligible.

On the other hand, if it is intended to change over production from Portland cement to blended cement, or if the blended cement accounts for a very high percentage of the total production, it is absolutely essential for a change in the grinding system to be studied in detail so as to ascertain both, the cost of production and the level of investment and to compare them.

When design work is being carried out for a new grinding system all aspects that have been referred to so far, must be looked into in detail to ascertain an adequate quality at lowest costs!

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Roller Press

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1. INTRODUCTION

The roller press is a relatively new development in the cement manufacturing process. This efficient comminution machine is applied mainly for clinker but is also used for raw material and slag. The size reduction of the material takes place under high pressure between two rotating rollers.

The press can be installed in combination with a tube mill (new installation or expansion of an existing mill) or as a finish grinding system.

2. GENERAL DESIGN AND WORKING PRINCIPLE

2.1 General Design (Figure 1)

The design of the roller press is dependent on the supplier, however, the basic construction is very similar for all makes.

The main elements are:

- Press frame
- Pair of rollers
  - One fixed roller
  - One movable roller
  - Each complete roller consists of the roller body with wear protection, two rolling bearings and two sliding blocks (bearing housings). The lubrication can either be by oil or grease. The rollers can be with/without water cooling.
- Rollerdrive (two units)
  - Planetary gear with torque arm
  - Drive shaft (fixed or movable)
  - Electric motor (fixed or variable speed)
- Hydraulic system
  - Cylinders (two or four)
  - Nitrogen accumulators (two or several)
  - Power pack consisting of oil tank, pump, motor and valves

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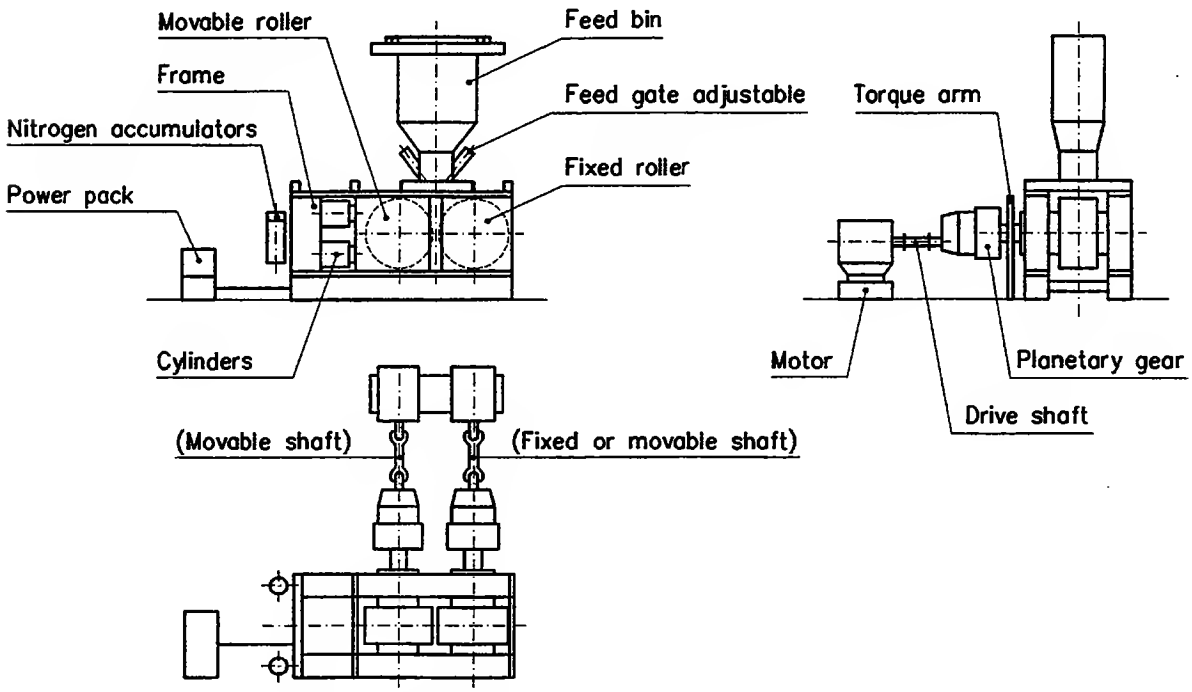
Figure 1 General Design



- Feed bin with level indicator or load cells (not part of the press)
- Adjustable feed gate

The area around the rollers is covered in order to prevent dust emission.

2.2 Working Principle (Figure 2)

Bulk material is fed to the gap between two rollers which are driven in counter-rotation. The necessary press force is created by a hydraulic-pneumatic system which works like a spring.

- Feed
  It is distinguished between choke feed and starved feed.
  - Choke feed: A column of material lays over the rollers. In this case, in order to assure smooth operation, it is very important that the material is deareated before it enters the gap. The choke feed is the common solution.
  - Starved feed: The feed is dosed into the gap (no material column). The advantage is that the air can easily escape, however, this solution requires a very accurate dosing equipment.
- Pressing process (size reduction)
  - The compaction of the material with the density ρo starts at the nip angle α. The maximum is achieved at the so called attack angle β (center of pressure) where the material reaches the slab density ρ_G.
  - Between α and β an interparticle comminution takes place where each particle transmits the force imposed on it to another particle. Best results are achieved with materials of wide particle size distribution with a maximum filling of the voids. In the pressing process a simple size reduction takes place but also flaws and fissures within the particles are produced reaching into the micro range.

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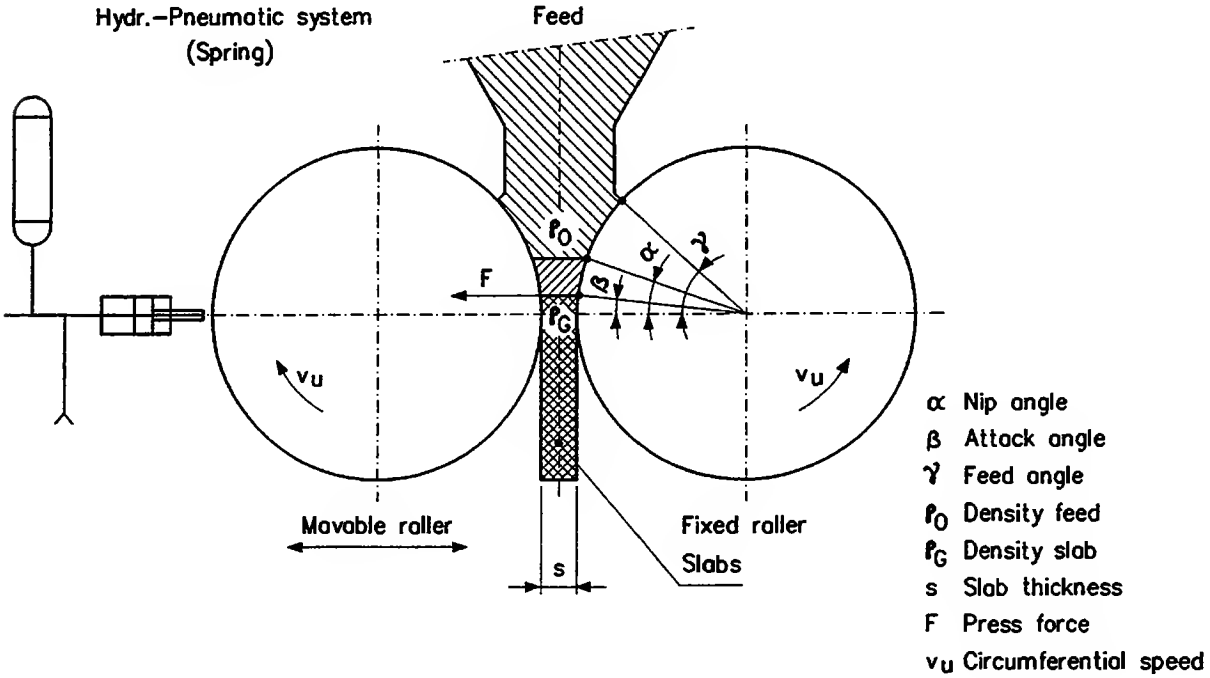
- Product
  - The material leaves the press in form of slabs which normally have the gap thickness **s**. For further processing these slabs must often be desagglomerated.

Figure 2 Working Principle



3. DIMENSIONING AND DIMENSIONING CRITERIA (FIGURE 3 AND 4)

Of interest for the user of a roller press are the calculation of the press throughput, the press force, the absorbed power and the specific energy consumption. For projects and comparison of presses further calculations are used such as specific press force and specific surface load.

- Press throughput ❶
  The throughput is given by the dimensions of the press and the density of the slab. This density is dependent on the operating parameters (e.g. pressure) and the feed material. The gap opening is a function of the material, roller diameter and roller surface and varies between 15 and 45 [mm]. Guide values for ρ_G are:
  - Clinker 2.4 - 2.5 [t/m^3]
  - Raw material 2.3 - 2.4 [t/m^3]
  - Slag 2.2 - 2.3 [t/m^3]
- Press force ❷
  The press force is given by the number and diameter of the hydraulic cylinders and the operating pressure. The operating pressure varies from supplier to supplier.

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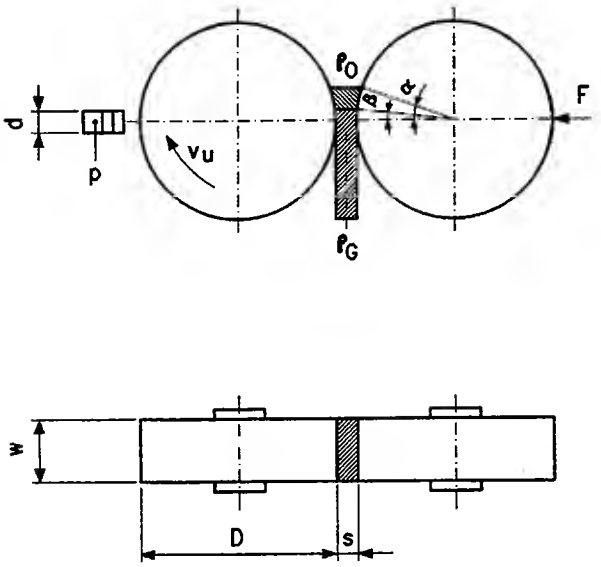
- Absorbed motor power ❸
  The absorbed motor power can be calculated by means of the press force, the circumferential speed of the rollers and the attack angle β.
  The attack angle β is dependent on the roller diameter, the gap opening and the material.
  Guide values for β are:
  - Clinker 2.3 - 2.85 [°]
  - Raw material 2.85 - 4 [°]
  - Slag 1.7 - 2.3 [°]

Figure 3 Dimensioning



Symbol	Designation	Unit
$\dot{M}$	Press throughput	(t/h)
$\dot{G}$	Product	(t/h)
D	Roller diameter	(mm)
w	Roller width (Slab width = Roller width)	(mm)
s	Slab thickness	(mm)
v_u	Circumferential speed	(m/s)
ρ_G	Density slab	(t/m^3)
ρ_0	Density feed	(t/m^3)
F	Press force	(kN)
d	Diameter hydr. cylinder	(mm)
z	Number hydr. cylinders	(-)
p	Hydr. pressure	(b)
K	Spec. press force	(kN/m^2)
L	Spec. surface load	(N/mm^2)
P	Abs. motor power	(kW)
E	Spec. energy consumption	(kWh/t)
α	Nip angle	(°)
β	Attack angle	(°)
u	Circulating factor	(-)

- Specific press force ❹
  - The specific press force is the press force divided by the projected roller area. Most suppliers allow a specific press force of ~ 8500 [kN/m^2].
  - In order not to overstress the wear protection on the rollers, HMC recommends ~ 6000 [kN/m^2] (to be subjected to further discussion).
- Specific surface load ❺
  The specific surface load is the press force divided by the part of the roller surface which is under pressure. This load should not exceed 250 [N/mm^2] (to be subjected to further discussion).
  Guide values for α are:
  - Clinker 6.9 - 9.2 [°]
  - Raw material 9.2 - 12.6 [°]
  - Slag 5.7 - 7.5 [°]

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- Specific energy consumption ❻ ❼
  - The specific energy consumption **E** is based on press throughput. Guide values are:
    - Clinker 2.5 - 3 [kWh/t]
    - Raw material 2.0 - 2.5 [kWh/t]
    - Slag ~3 [kWh/t]
  - If a press is in closed circuit with a separator (finish grinding) or has a slab recirculation the specific energy consumption is multiplied by the circulating factor **u**.
  - The specific energy consumption **Ec** is then based on the material leaving the press circuit.

Figure 4 Dimensioning

(1) Press throughput

$$\dot{M} = \frac{w \cdot s \cdot v_u \cdot \rho_G \cdot 3{,}6}{1000} \quad (t/h)$$

(2) Press force

$$F = \left(\frac{d}{1000}\right)^2 \cdot \pi \cdot z \cdot p \cdot \frac{100}{4} \quad (kN)$$

(3) Absorbed motor power

$$P = \frac{2 \cdot F \cdot \beta_{rad} \cdot v_u}{\eta} \quad (kW)$$

η Efficiency gear and motor

(4) Specific press force

$$K = \frac{F \cdot 10^6}{D \cdot w} \quad (kN/m^2)$$

(5) Specific surface load

$$L = \frac{F \cdot 10^3}{w \cdot \frac{D}{2} \cdot \alpha_{rad}} \quad (N/mm^2)$$

(6) (7) Specific energy consumption

$$E = \frac{P}{\dot{M}} \quad (kWh/t)$$

$$E_C = \frac{P \cdot u}{\dot{M}} \quad (kWh/t)$$

4. OPERATIONAL ASPECTS

4.1 Installation of Roller Press (Figure 5)

The roller press can be installed in various configurations.

- In combination with tube mill
  - Roller press as pregrinding unit
  - Roller press as pregrinding unit with slab recirculation
  - Two stage grinding system (semi-finish)

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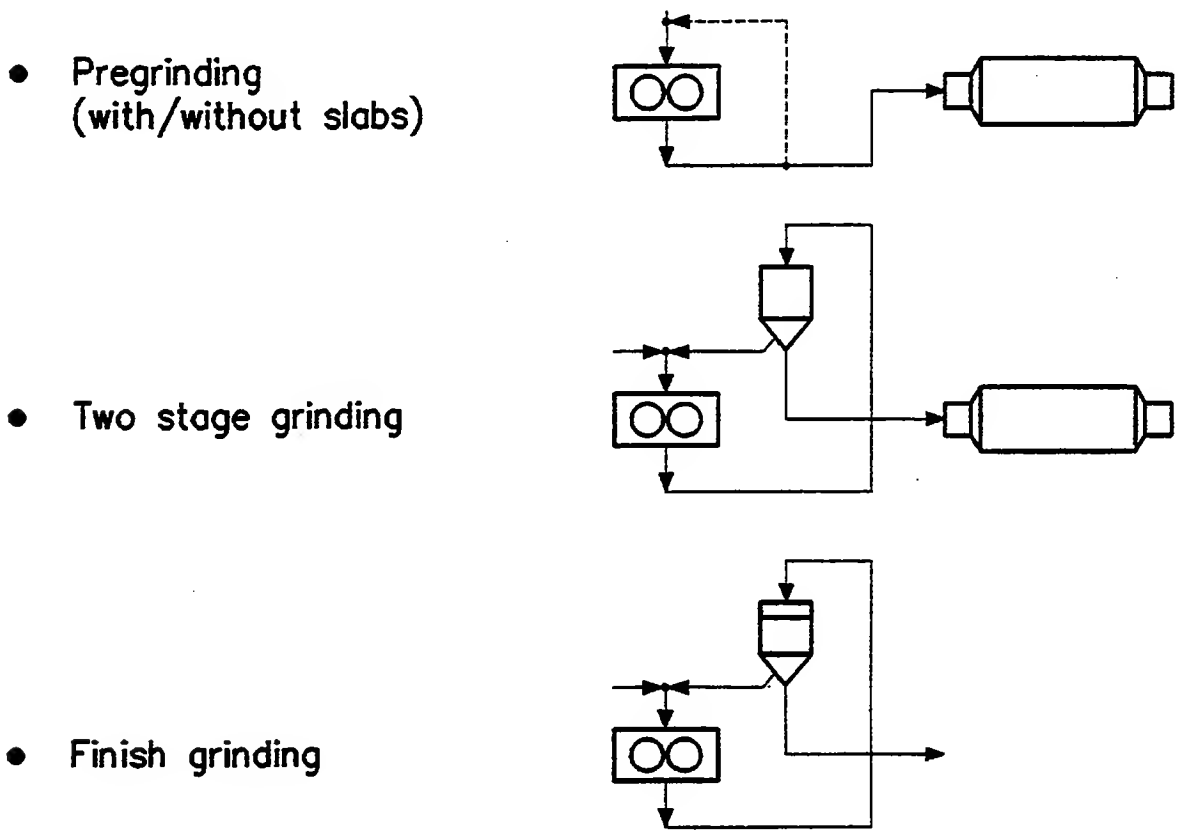
- Finish grinding system
  - The roller press is in closed circuit with a desagglomerator and a separator
- The detailed description with advantages and disadvantages is given in the paper VA 93/4014/E, Cement grinding systems.

Figure 5 Installation of Roller Press



4.2 Feed Granulometry and Roller Speed (Figure 6)

The maximum feed size depends on the roller diameter, the roller surface and the supplier. Up to 50 [mm] is a normal given maximum size.

A commonly used value is also

- max feed size = 2 x gap opening

A strong relation exists between the granulometry of total feed and the circumferential speed of the rollers. If the feed contains a high amount of very fine material and the speed of the rollers is high (~ 1.8 m/s) the press tends to vibrate and a smooth operation is not possible.

- Pregrinding without slab recirculation
  Coarse feed to the press, vibration normally not a problem.
- Pregrinding with slab recirculation
  This is the critical case since the slabs contain a high amount of fine material. Figure 6 shows the relation between circumferential speed and circulating load. At a circulating load of 2 (total feed : product) the roller speed should not exceed 1.5 - 1.55 [m/s].

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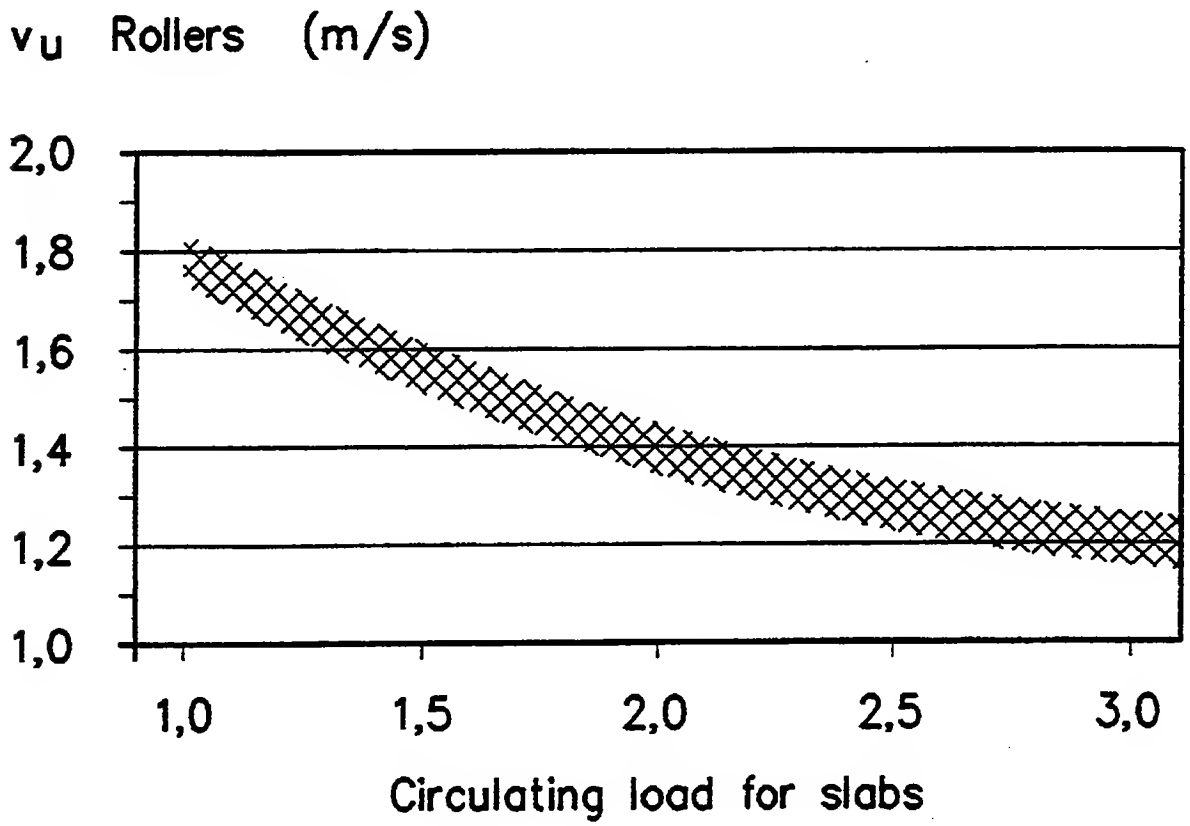
- Two-stage and finish grinding system
  Not critical since the return from the separator does not contain very fine material.

The speed range of the "Holderbank" roller presses is 1 - 1.55 (m/s).

Figure 6 Circumferential Speed of Rollers



4.3 Adjustable Feed Gate (Figure 7 and 8)

An important influence on the press operation (material intake, gap opening) has the feed gate. All press manufacturers supply their presses with adjustable feed gates. A very sophisticated gate is shown in Figure 7.

Functioning

- Lower gate 1: The two tongues have only an open or closed position. With this feature the feed to the press can be interrupted and the press be stopped with a full feed bin. (hydraulically or pneumatically activated).
- Upper gate 2: This is the real adjustable feed gate which allows:
  - Flow regulation to the press (changing chute opening)
  - Excentrical distribution of the material
  - This part of the gate can also be motorized or kept manually, e.g.:
    * fixed roller side manually
    * movable roller side motorized

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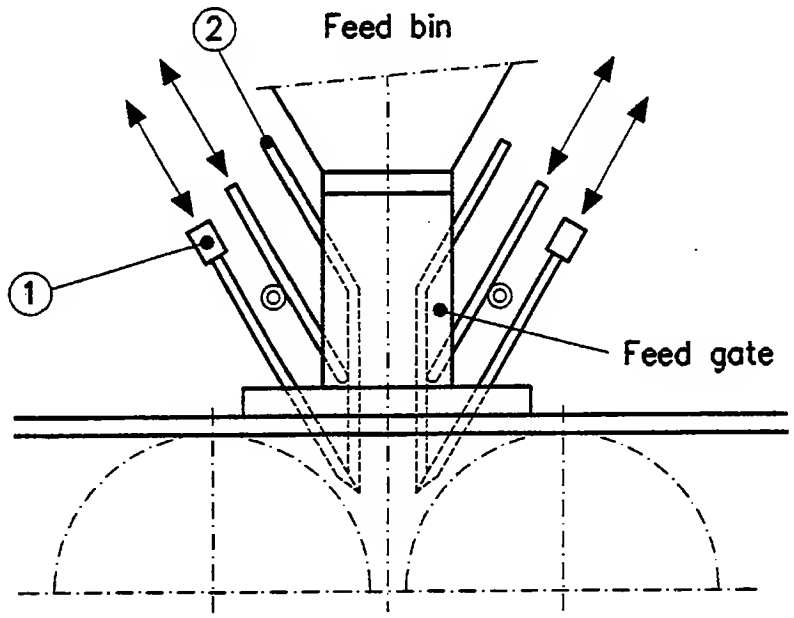
Figure 8 shows a typical example how the gap opening can be influenced by the position and the opening of the gate. Very often best results are achieved if the opening is moved towards the fixed roller and with this position also equal power draw of the rollers is achieved.

The best position must be found by trials.

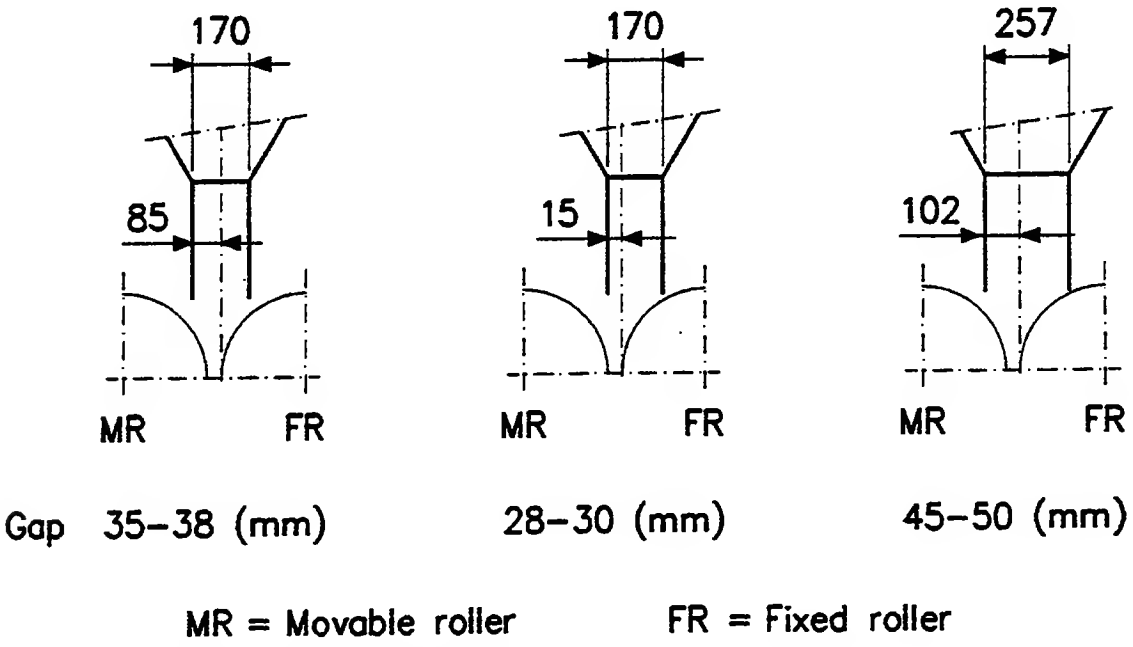
Figure 7 Adjustable Feed Gate



(1) Lower gate, open – closed position

(2) Upper gate, continuously adjustable

Figure 8 Influence on Gap Opening



4.4 Operating Parameters (Figure 9)

The degree of comminution depends on the specific energy which is transmitted into the material. As shown in chapter 3 - Dimensioning - the absorbed power of the press is linear to the press force applied and the latter is linear to the operating pressure. The operating pressure can mainly be influenced by 3 parameters which are:

- Accumulator, nitrogen charge setting (initial)
- Hydraulic system, set point pressure
- Initial gap, zero setting of gap

The following relations exist:

- Initial nitrogen pressure < set point pressure hydraulic system < operating pressure
- If a large difference exists between initial nitrogen pressure and set point pressure hydraulic system the accumulators work like a hard spring.
  If a small difference exists between initial nitrogen pressure and set point pressure hydraulic system the accumulators work like a soft spring.

The press force (operating pressure) can be increased by (or vice versa):

- Reduction of initial nitrogen pressure
  - Set point hydraulic pressure kept constant
  - Zero gap setting not changed
- Increase of set point hydraulic pressure
  - Initial nitrogen pressure kept constant
  - Zero gap setting not changed
- Zero gap setting increased
  - Initial nitrogen pressure kept constant
  - Set point hydraulic pressure kept constant

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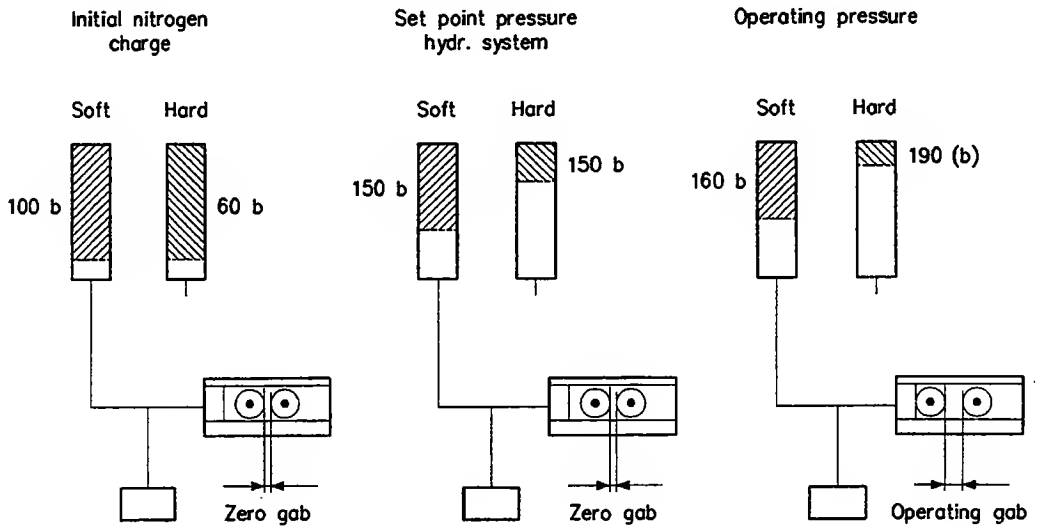
The most efficient working point (absorbed power, hard or soft spring) must be found during commissioning by trials.

Figure 9 Operating Parameters



5. MECHANICAL ASPECTS (FIGURES 10 AND 11)

5.1 Wear

Wear on a roller press mainly occurs in the feed chute on the roller surface and on the cheek plates. The wear rate depends on the feed material (abrasiveness, moisture etc.) and on the operating parameters of the press.

5.1.1 Rollers

The wear rate itself on the rollers is low (for clinker 1-2 g/t) and does not present a problem, however, the entering of foreign matters (grinding balls, metal pieces etc.) can lead to the destruction of the roller surface. The improvement of the resistance of the roller surface is still an ongoing process and no final solution is found.

5.1.1.1 Welded protection (common solution)

The welding consists of buffer layers and hard facing. The so called "peeling off" takes often place if metal pieces enter the press (large pieces of welding are falling off). Smaller repair work can be done in the press. Reconditioning should be made in a workshop.

5.1.1.2 Stud lining

KHD introduced this type of protection.

Wear resistant studs are welded onto the rollers.

The roller body is protected by the feed material which is pressed into the voids between the studs. The reliability of this solution must still be proven.

5.1.1.3 Clamped on tyres

No experience available

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5.1.1.4 Clamped or bolted on segments

This solution reduces mainly the down time of the press if a damage occurs.

This is a fairly new approach and not enough experience is available. However, a trend can be seen towards this solution.

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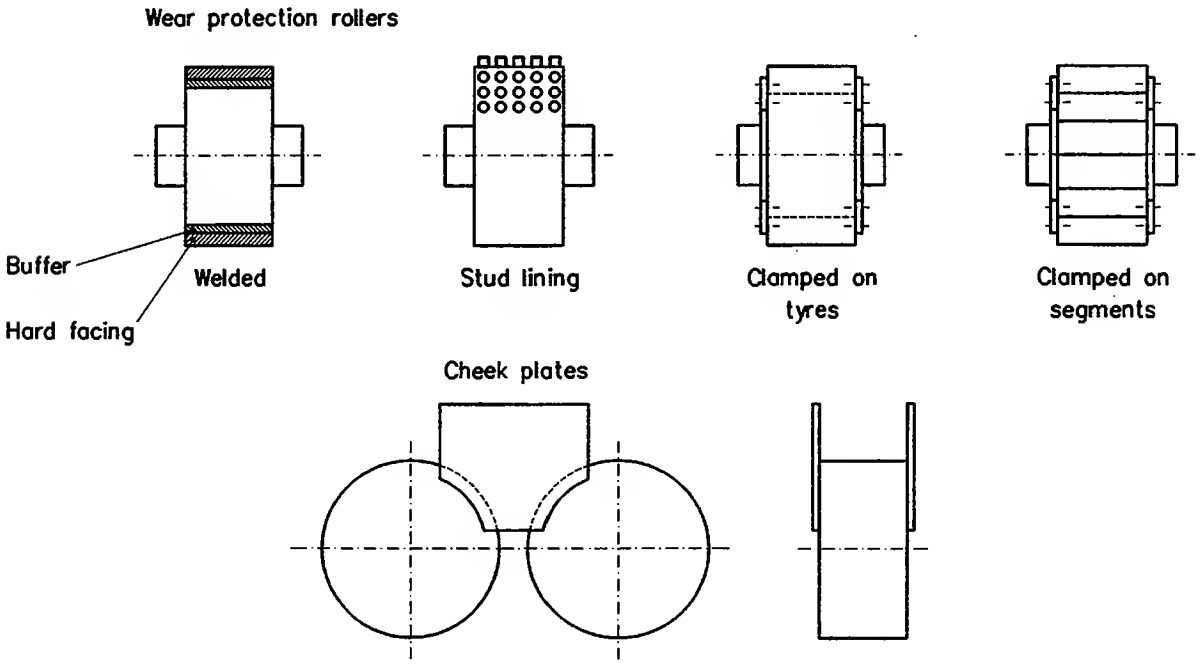
5.1.2 Cheek plates

The function of the plates is to avoid the escape of material sidewards from the rollers. They are made from wear resistant material and are easily exchangeable.

Figure 10 Mechanical Aspects



5.2 Press Protection (Figure 11)

In order to keep the risk of roller surface damages as low as possible all foreign matters must be removed from the press feed (fresh feed and recirculating material). The feed material must pass below an electromagnet and a metal detector. The detector is linked to a by-pass gate. Metals which are not removed by the magnet will be detected by the metal detector and will by-pass the roller press.

Some presses have in addition a screen between clinker storage and press feed bin for a first separation of large foreign matters and oversized feed.

5.3 Movable Feed Bin

In order to simplify the disassembly of the rollers and to do major repairwork on the press the installation of a movable feed bin is advantageous.

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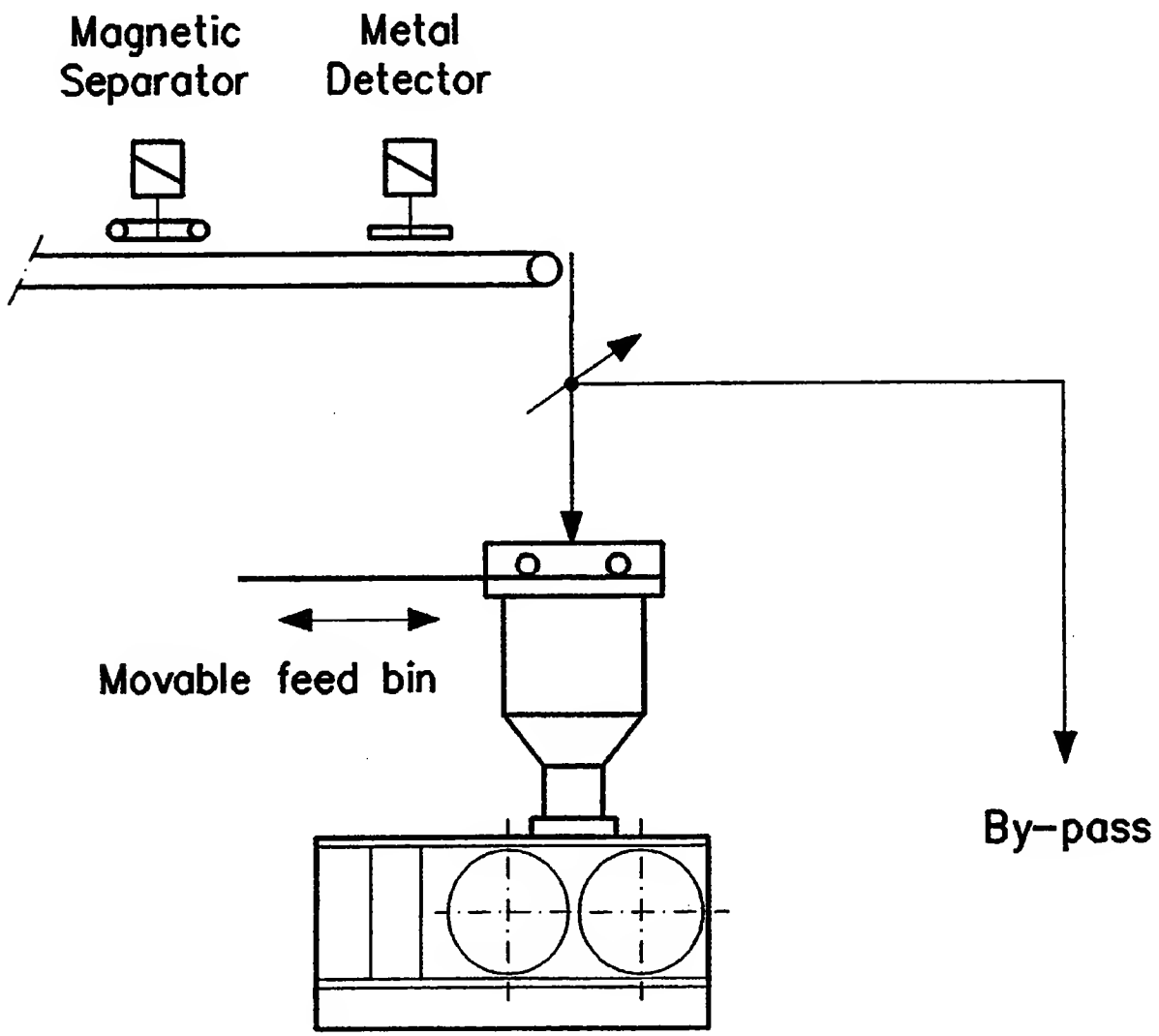
5.4 Inspections

A continuous operation of a roller press can only be achieved if the critical areas or parts of the press are regularly inspected and serviced.

- During operation
  - Vibrations of press and press drive
  - Bearing and oil temperatures (if oil lubricated)
  - Lubrication system
- During stop
  - Wear parts (rollers, cheek plates, inside feed chute)
  - Bearings and bearing sealings
  - Lubrication system, water cooling system
  - Hydraulic system, pressure setting accumulators
  - Drive including gear boxes, shafts, clutches, electrical motors

Figure 11 Press Protection



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6. MAIN SUPPLIERS

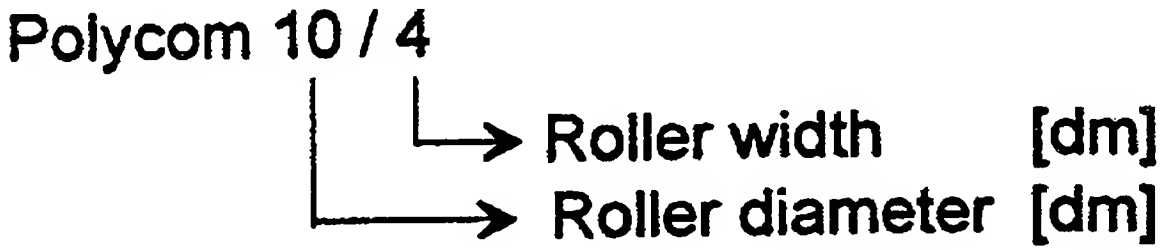
6.1 Krupp - Polysius (Figure 12)

6.1.1 General

Krupp - Polysius, besides KHD, is mainly responsible for the design, application and further development of the roller press.

6.1.2 Make:

Make **Polycom 10 / 4**



6.1.3 Standard sizes: 9

Power range:	300 - 2500	[kW]
Throughput:	90 - 700	[t/h] clinker
Specific press force:	~ 6500	[kN/m^2]

6.1.4 Design features

Frame: Bolted design, on the fixed roller side a slide block can be added to the press in order to remove the rollers (both on one side).

Rollers: Wear protection by hard face welding, tyres or segments, water cooling of shaft standard outfit.

Roller bearings: Self aligning spherical roller bearings with grease lubrication.

Hydraulic system: 2 x 2 hydraulic cylinders and on each side one nitrogen accumulator.

Drive system (both rollers identical): Shaft mounted planetary gear with torque arm, safety coupling; universal joint and electrical motor.

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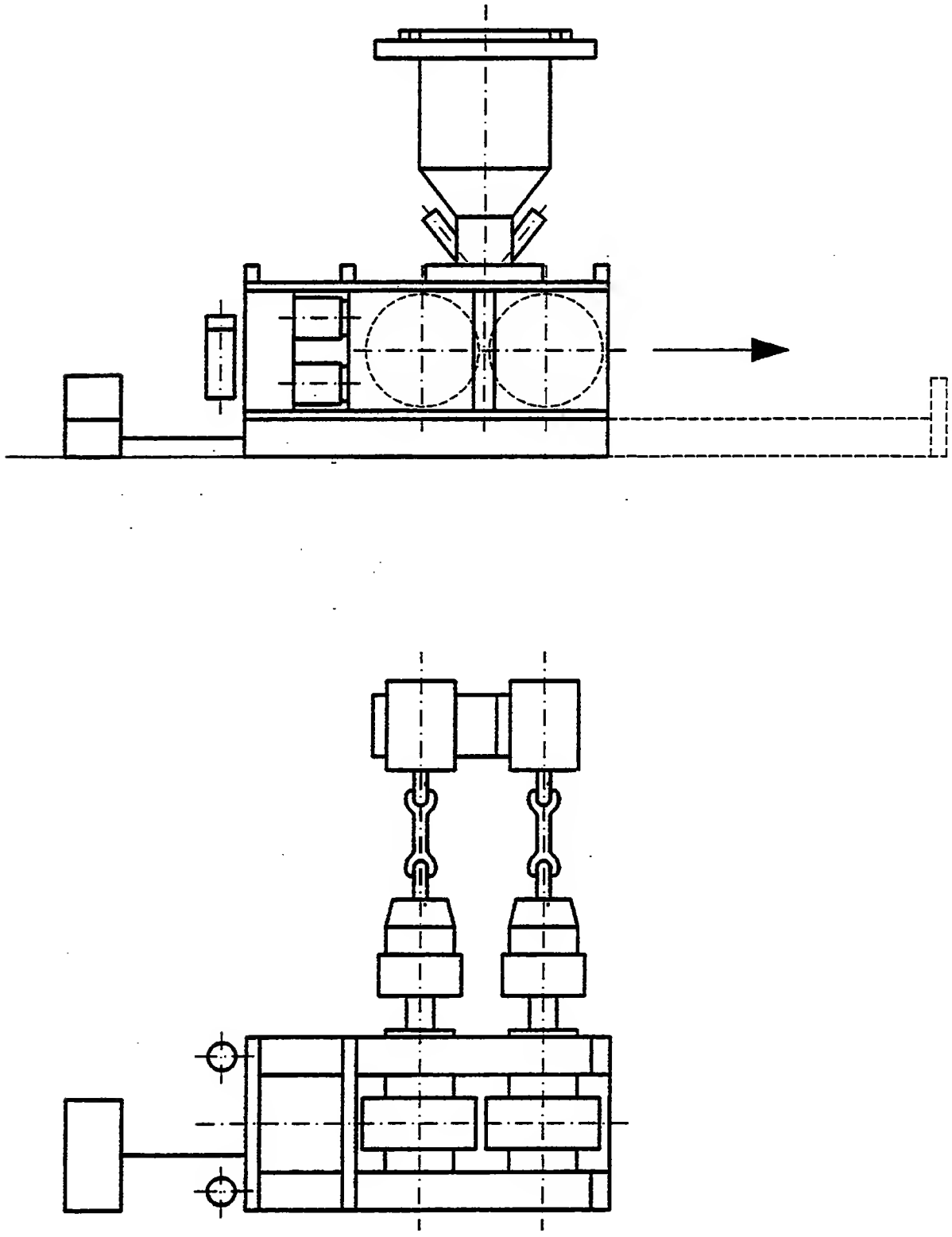
6.1.5 Experience

Very experienced supplier, wear protection rollers still in a development phase.

Figure 12 Krupp Polysius



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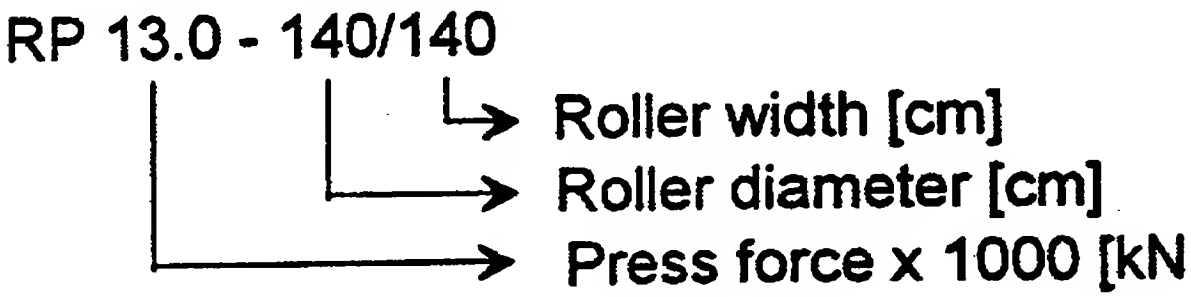
6.2 KHD - Humboldt Wedag (Figure 13)

6.2.1 General

Same comment as for Krupp-Polysius

6.2.2 Make:

Make **RP 13.0 - 140 / 140**



6.2.3 Standard sizes: 7 (various speeds within each size)

Power range: 150 - 4200 [kW]

Throughput: Not specified

Specific press force: ≤ 7000 [kN/m^2]

6.2.4 Design features

- Bolted design: Feed chute and top girders must be dismantled for removing rollers.
- Rollers: Wear protection by hard face welding, welded on studs or segments with welded on studs, water cooling of shaft standard outfit.
- Roller bearings: Cylindrical roller bearings with oil lubrication.
- Hydraulic system: One large hydraulic cylinder on each side including three nitrogen accumulators (2 large, 1 small).
- Drive system (both rollers identical).
  Shaft mounted planetary gear with torque arm, fluid coupling, universal joint and electrical motor.

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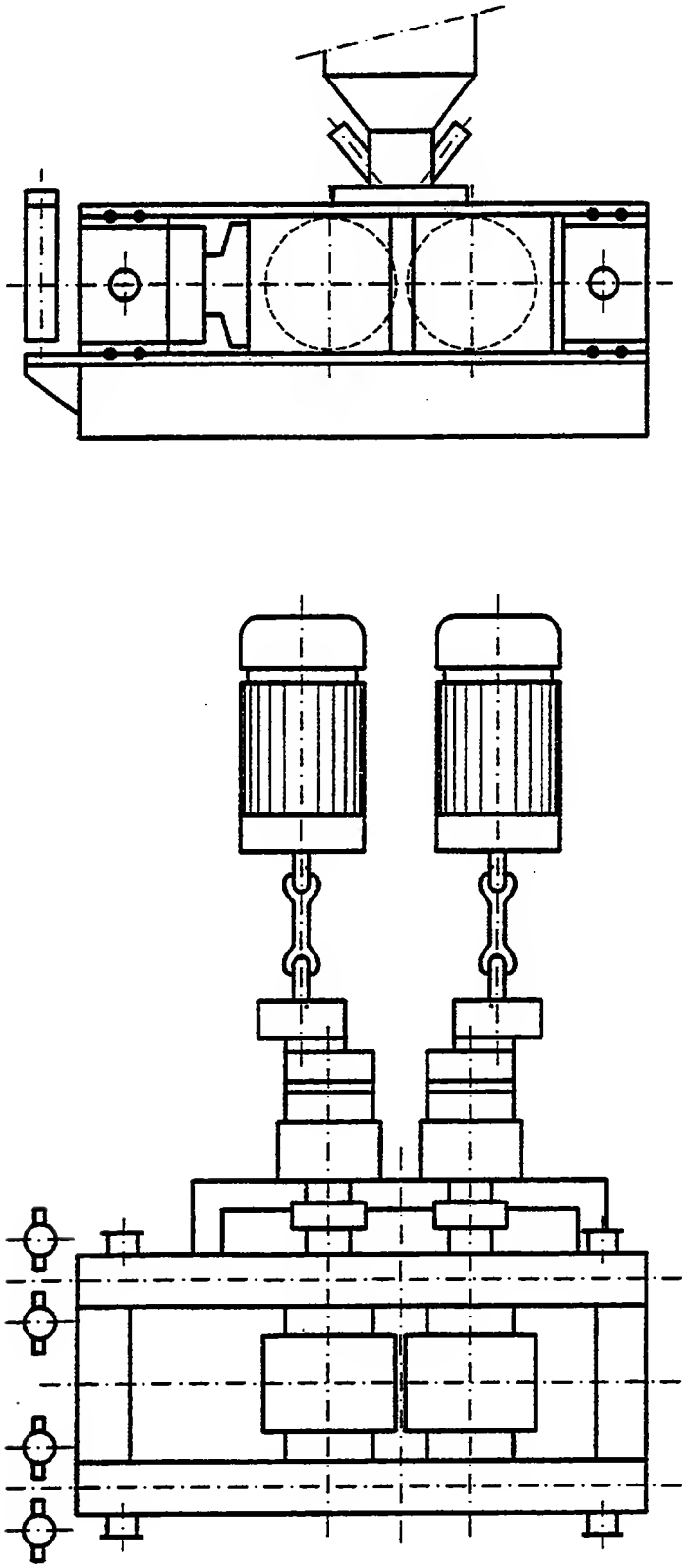
6.2.5 Experience

Very experienced supplier, wear protection rollers still in a development phase.

Figure 13 KHD Humboldt Wedag



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6.3 F.L. Smidth-Fuller (Figure 14)

6.3.1 General

In earlier days FLS sold the Köppern roller press. Fuller manufactured their own roller press.

The newest design is a combined FLS-Fuller press

6.3.2 Make:

HRP 1.25

Projected roller area

Standard sizes: 8

Power range:	503 - 3024	[kW]
Throughput:	167 - 995	[t/h] (clinker)
Specific press force:	8000	[kN/m2]

6.3.3 Design features

- Frame: Bolted design, one side can be hinged down for removing the rollers.
- Rollers: Wear protection by hard face welding or segments, optional with water cooling for shaft.
- Roller bearings: Self aligning spherical roller bearings with oil lubrication.
- Hydraulic system: 2 x 2 hydraulic cylinders.
- Drive system movable roller: Shaft mounted planetary gear with torque arm, universal joint shaft, mechanical torque limiting coupling, electrical motor (preferred SC motor slip ring type).
- Drive system fixed roller: Double gear type transmission shaft, foot mounted planetary gear, torque limiting coupling, electrical motor.

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6.3.4 Experience

New press, however, proven standard elements are used.

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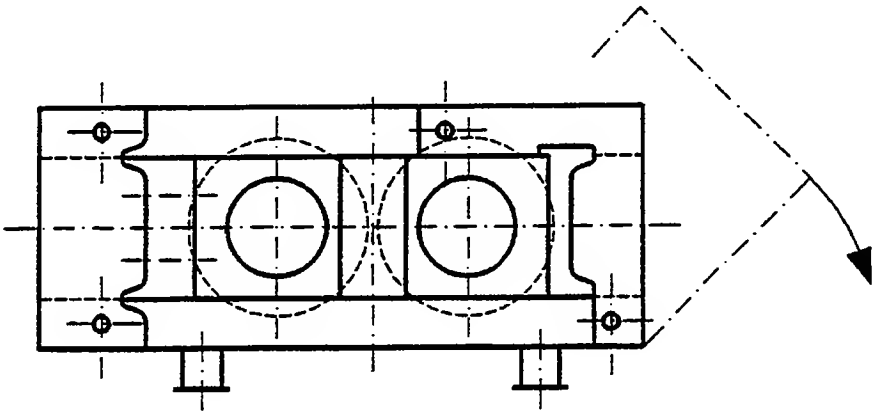
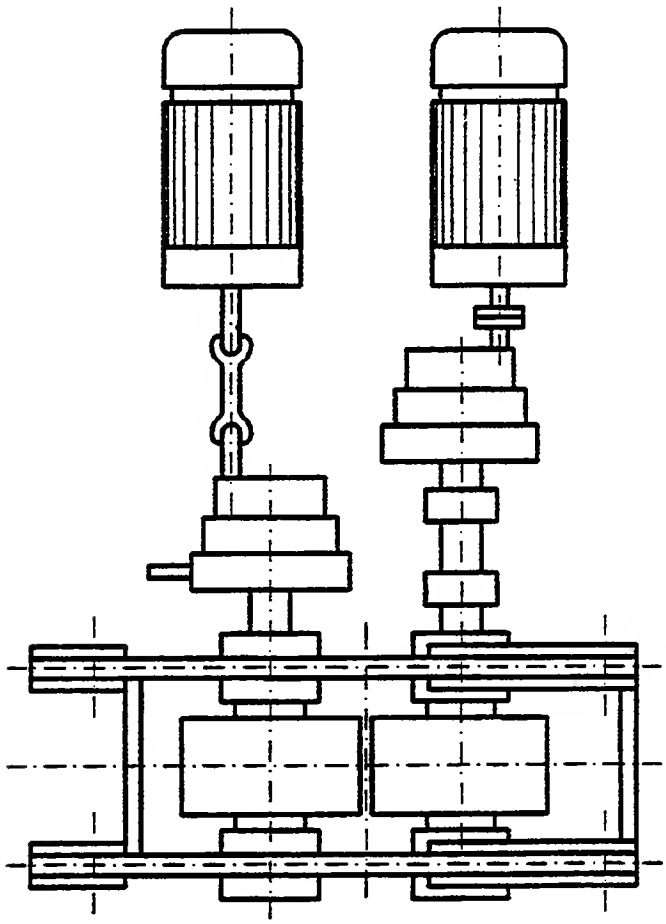
Figure 14 F L Smidth - Fuller





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C. Guiot
10/05/2000

Binder 5

Process Technology II

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Chapter 1

Kiln Systems

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Kiln Systems - Overview

Urs Gasser
PT 99/14501/E

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SUMMARY

Today's kiln systems for burning cement clinker of major importance use a rotary kiln. Exceptions are vertical shaft kilns still used in certain geographical areas. With very rare exceptions, new plants use the dry process. However, there are still important markets where older wet process plants are predominant (USA, Russia).

A first classification of the process can be made based on the water content of the kiln feed:

	< 1% water	dry-process
10 ...	12% water	semi-dry-process
17 ...	21% water	semi-wet-process
25 ...	40% water	wet-process

- Dry-Process
  - Precalciner kiln with 4 to 6 cyclone stages (contemporary technology):
    * Separate tertiary air duct
    * 50 - 60% fuel to the precalciner
    * Large capacities possible > 10000 t/d
    * Up to 4000 t/d in 1 string
    * Heat consumption < 3000 kJ/kg possible (6 stages)
    * Sensitive to circulation phenomena (-> kiln gas bypass!)
  - 4-stage cyclone pre-heater kiln (standard technology 1970 to 1980):
    * Cyclone stages (co-current flow) for raw meal preheating
    * Large application world wide
    * Capacities of up to 4500 t/d technically possible
    * Heat consumption: 3150 to 3350 kJ/kg cli
    * Sensitive to circulation phenomena (-> kiln gas bypass!)
  - 2-stage cyclone pre-heater kiln:
    * Less sensitive to circulation phenomena than 4-stage pre-heater
    * Higher heat consumption than pre-heater with more stages
  - Shaft pre-heater kiln:
    * Counter current heat exchange between hot gas and raw meal
    * Practical efficiency inferior to cyclone pre-heater
  - Long-dry-kiln:
    * Rather simple equipment
    * High dust emission from kiln tube
    * Without heat exchange internals: high heat consumption of up to 5100 kJ/kg cli
    * With chains and/or crosses: 4200 kJ/kg cli achievable

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- Semi-dry and semi wet process
  - Grate pre-heater kiln (LEPOL, ACL):
    * Raw meal must be suitable to be nodulised with water (semi-dry)
    * 3450 kJ/kg cli (no waste heat available for primary raw material drying)
  - Long rotary kiln and suspension preheater:
    * Filter cakes fed or slurry injection into vertical dryer; rather rare cases
- Wet-process
  - Long wet kiln:
    * Fed with raw meal slurry of approx. 32 - 42% water content
    * Internal heat transfer improved by chains
    * High heat consumption of 5300 to 6300 kJ/kg cli due to evaporation of water
    * Heat consumption reduced by slurry thinners for a slurry with 25 - 30% H2O
    * Slurry preheaters can reduce kiln size and improve heat exchange

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1. PROCESS REQUIREMENTS FOR KILN SYSTEMS

The kiln system has to be designed to cope with the requirements of the chemical process during which the kiln feed material is converted into cement clinker.

This process as a whole is endothermic and takes place at maximum material temperatures of 1450°C. Receiving its thermal energy from hot gases of up to 2000°C generated by combusting fuels, it is also referred to as pyroprocess.

Type of reaction and temperature development are compiled in "sequence of reactions occurring in a rotary kiln" (table 1) and graphically as the "quasi-qualitative variation of minerals with temperature" (figure 1).

The chemical process taking place in the kiln system where raw meal (input) is converted to cement clinker (output) can be subdivided into the following five steps:

1. Drying
2. Preheating
3. Calcining
4. Sintering
5. Cooling

Process and equipment has been developed and improved with the aim at performing these steps forever improved economy, which means

- High availability
- Low heat consumption
- Low power consumption
- Higher unit capacity
- Stable kiln operation
- Good, uniform clinker quality

Table 1 Sequence of Reactions occurring in a Rotary Kiln

Temperature range (°C)	Type of reaction
Heating Up	
20 - 100	Evaporation of free H2O
100 - 300	Loss of physically absorbed water
400 - 900	Removal of structural H2O (H2O and OH groups) from clay minerals
> 500	Structural changes in silicate minerals
600 - 900	Dissociation of carbonates CO2 driven out)
> 800	Formation of belite, intermediate products, aluminate and ferrite
> 1250	Formation of liquid phase (aluminate and ferrite melt)
approx. 1450	Completion of reaction and re-crystallisation of alite and belite
Cooling	
1300 - 1240	Crystallisation of liquid phase into mainly aluminate and ferrite

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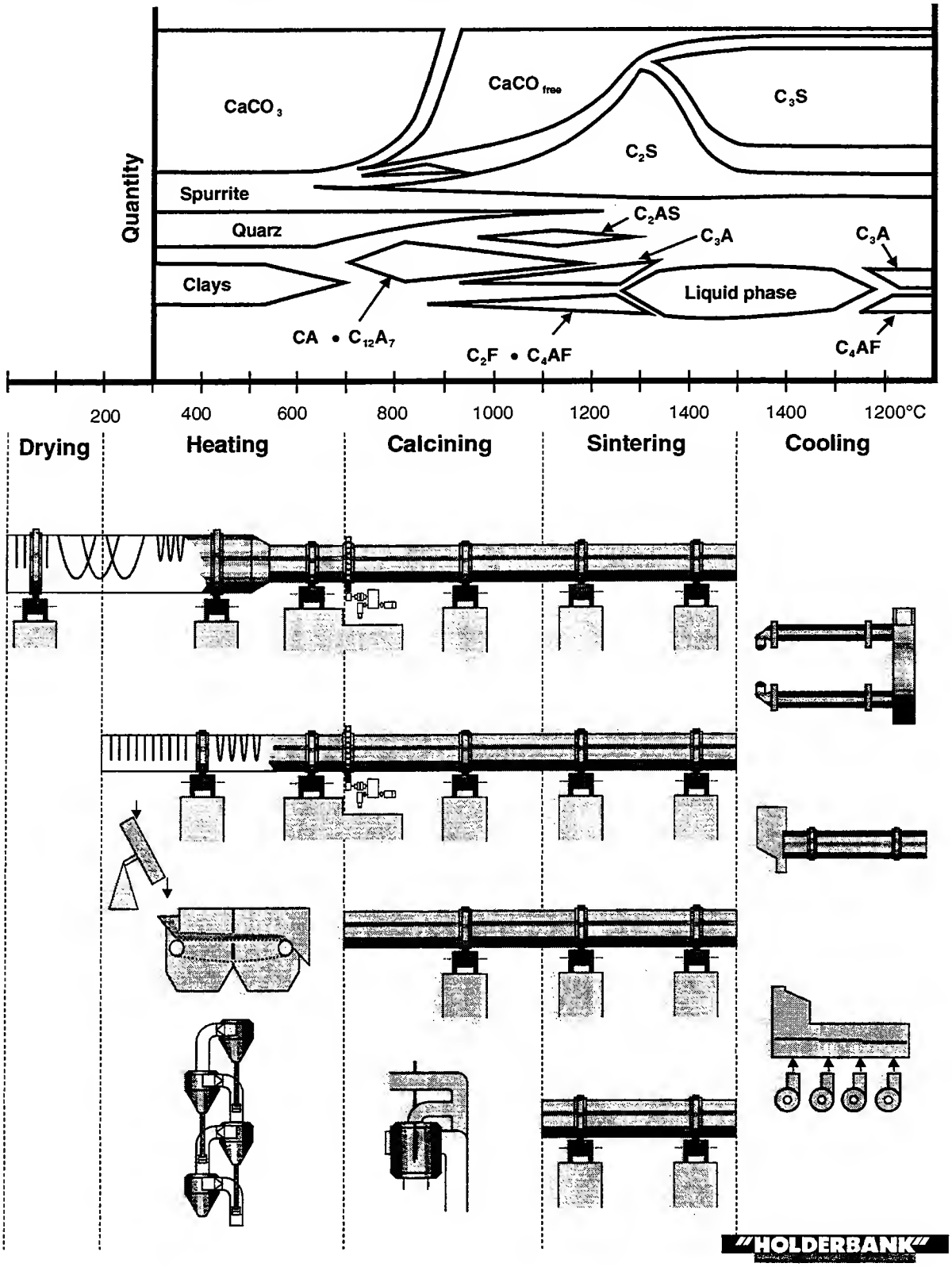
Heat exchangers in a kiln system



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Process Technology

2. PROCESS TYPES

2.1 General

The criterion normally used to distinguish the process types is the moisture of the kiln feed material. Four basically different process types for clinker burning can be defined:

Process Type	Feed Material	Cons.	Feed Moisture	Feed System
Dry process	Raw meal	Dry	< 1% H2O	Mechanic, pneumatic
Semi dry process	Nodules	Moist	≈ 10 ... 12% H2O	Mechanic, pneumatic
Semi wet process	Filter cake, nodules	Moist	≈ 17 ... 21% H2O	Mechanic, pneumatic
Wet process	Slurry	Liquid	≈ 25 ... 40% H2O	Hydraulic

Table 1 gives a general survey of the various rotary kiln systems in operation for industrial clinker production. Shaft kilns, which are still used in China or experimental systems such as sintering grates or fluidised beds, are not considered in the scheme.

We can distinguish two main groups of kiln systems:

a) Long kilns with or without internal heat exchanging installation

b) Short or medium kilns with external preheaters
(e.g. suspension preheaters, grates or external slurry preheaters)

The heat consumption of burning depends strongly on the water content of the kiln feed

This can be illustrated by the typical specific heat consumption: The fuel consumption of wet kilns is nearly twice as high as for modern dry process suspension pre-heater kilns.

The comparison of the heat economy within each process group (dry or wet) shows clearly:

The more intensive the heat-exchange for drying and preheating, the lower the heat consumption.

Other than based on the feed moisture, kiln systems can be grouped in different ways:

Process Type	wet semi wet semi dry dry	>25% H2O in feed 17 - 21% H2O in feed 10 - 12% H2O in feed < 1% H2O in feed	Slurry nodules from slurry nodules from meal raw meal
Production Mode	batch+cont. continuous	< 200 t/d 300 t/d – 10'000 t/d	shaft kilns rotary kilns
Heat Consumption		2900 kJ/kg cli (700 kcal/kg cli) > 6000 kJ/kg cli (> 1400 kcal/kg cli)	state of the art system long wet or dry kilns, not optimum operation
Power Consumption		20 to 65 kWh/t cli	kiln feed to clinker cooler

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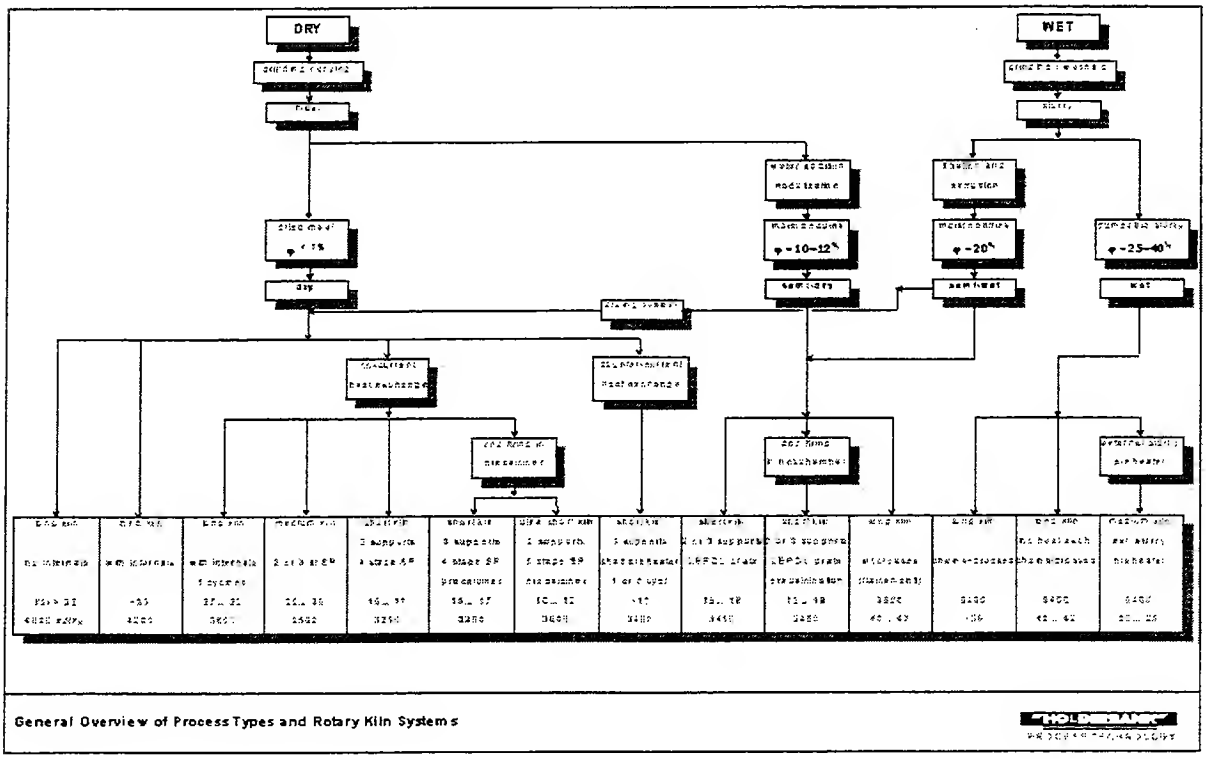
OVERVIEW OF KILN AND PROCESS TYPES



General Overview of Process Types and Rotary Kiln Systems

When the concept for a new plant is developed, not only the present situation but also the possible future developments of all relevant factors must be taken into account.

The following main parameters must be considered when selecting the kiln system:

- Raw material:
  * moisture content
  * grindability
  * homogeneity of deposit
  * number of components for raw mix
  * chemical composition (sulphur, chlorides, alkalis, organic compounds etc)
  * filtration properties of slurry (for semi-wet process only)
- Plant installation and operating costs
- Requirements for clinker quality (e.g. low alkali clinker)
- Aspects of environmental protection (emission of dust, SOx, NOx, etc)
- Technical standard of the country

Long wet (and dry) rotary kilns are the oldest and most simple type of installation to produce cement clinker. The pyroprocess takes place in a long rotating tube, which has usually internal equipment to improve heat transfer, and, in wet kilns, to reduce dust loss. Unit capacities of up to 2000 t/d are typically achieved, higher outputs are possible, however, they require kilns of gigantic dimensions.

Today, economy requires plants for 3000 to 10'000 t/d. Therefore new plants are almost always based on the dry process with preheater, pre-calciner and reciprocating grate cooler. The semi wet process for a new plant could be preferred in special cases, e.g. where raw material with a high natural moisture must be used (e.g. quarry below water level).

The three following graphs illustrate the development of the significance of the various processes within the Holderbank group, which can be considered representative of the global situation.

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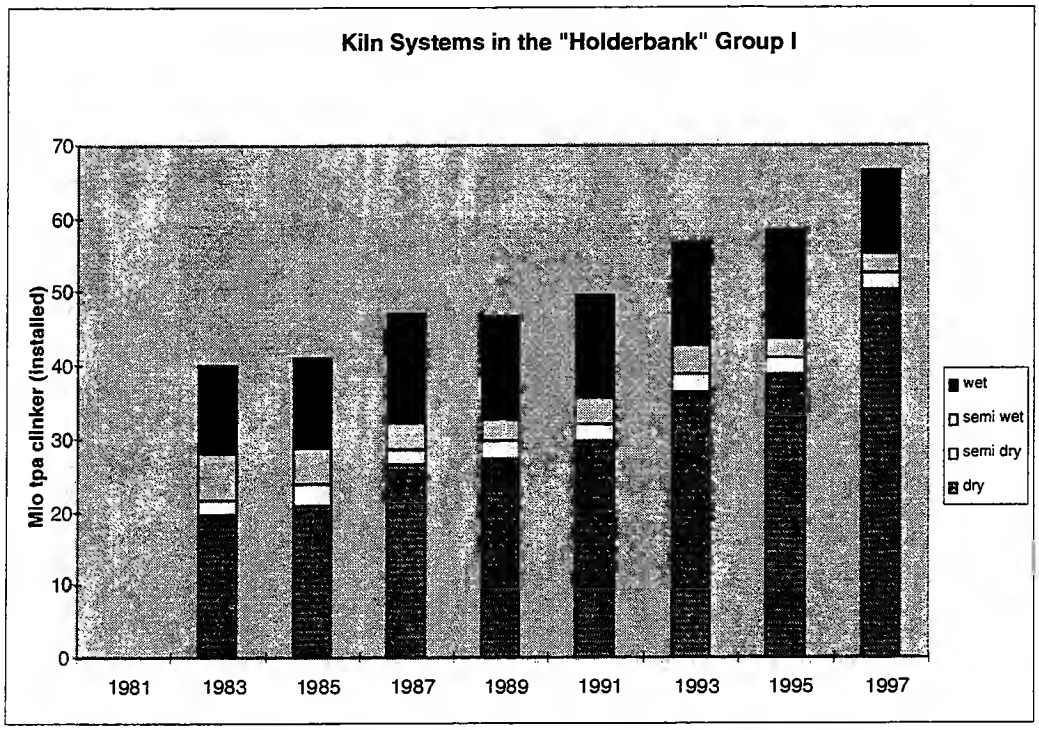
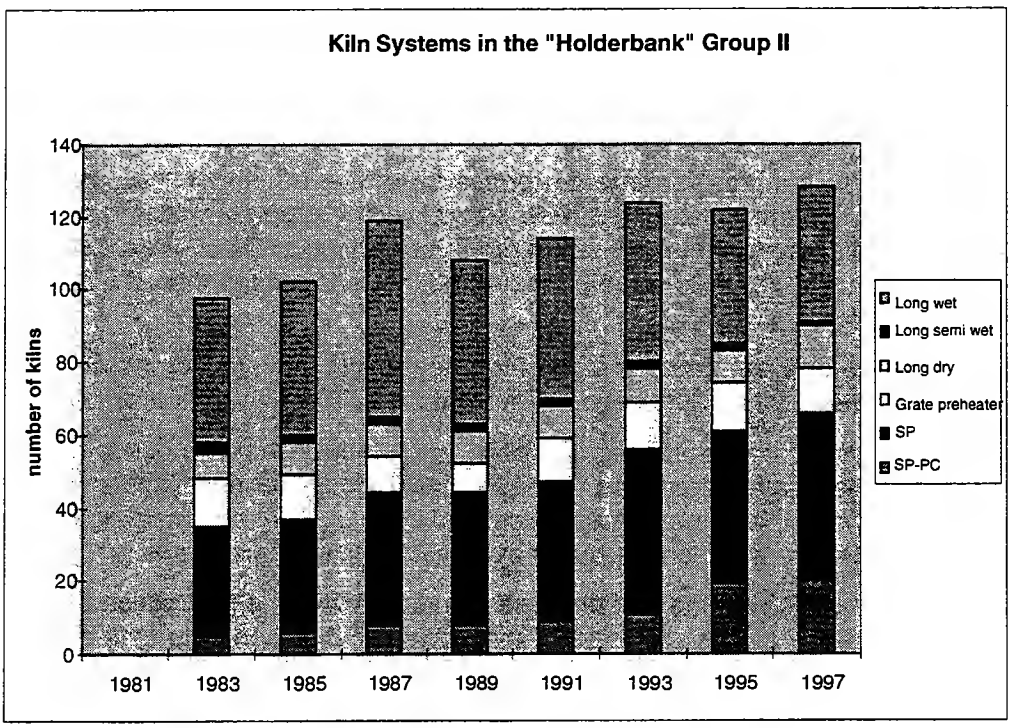
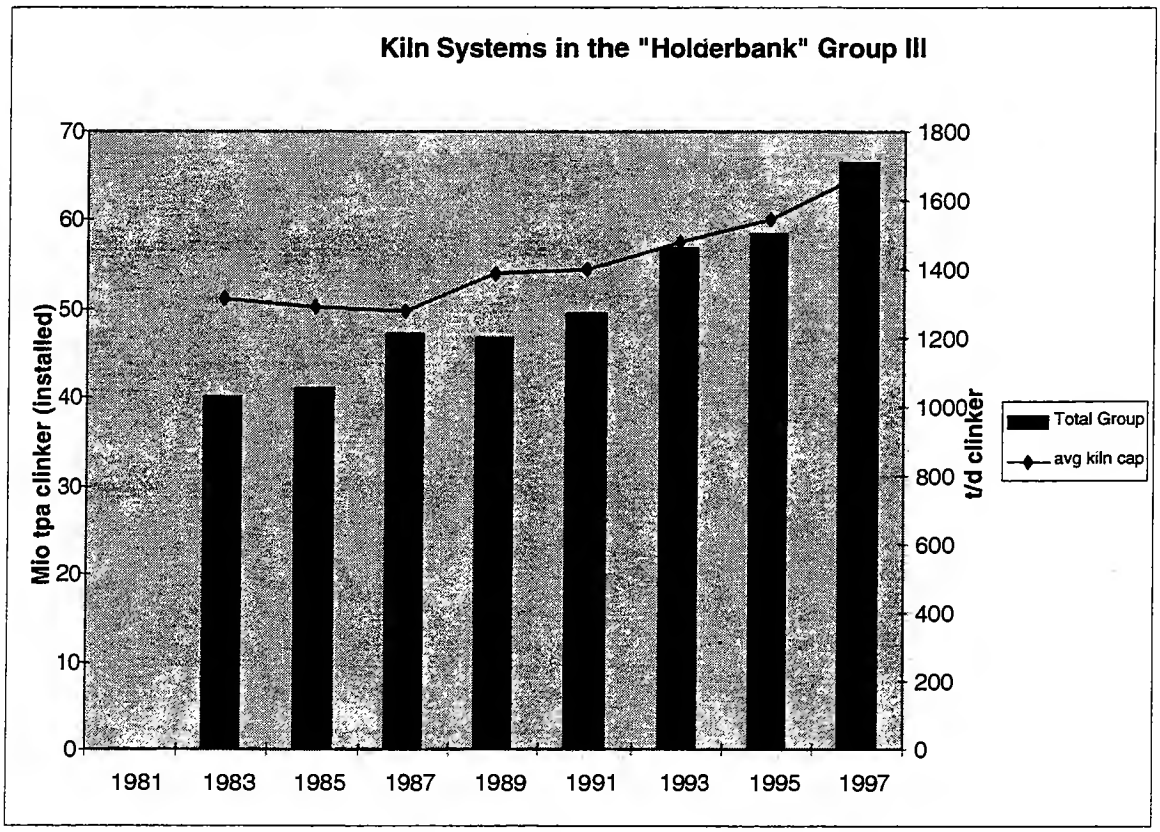

Kiln Systems in the "Holderbank" Group I
Mio tpa clinker (installed)
0
10
20
30
40
50
60
70
1981
1983
1985
1987
1989
1991
1993
1995
1997
wet
semi wet
semi dry
dry


Kiln Systems in the "Holderbank" Group II
number of kilns
0
20
40
60
80
100
120
140
1981
1983
1985
1987
1989
1991
1993
1995
1997
Long wet
Long semi wet
Long dry
Grate preheater
SP
SP-PC



3. WET PROCESS

3.1 General

The wet process was the most important process for clinker burning in the past and almost all plants were wet. Heterogeneous quarries and corrective addition were no problem; stirring of the liquid slurry in the slurry tanks provides very good batch-wise blending. Grinding was done in slurry mills, which consume 30%, less energy than dry ball mills, but at higher lining wear rates.

The disadvantage of the wet process is the high heat consumption. Compared to e.g. a suspension preheater kiln, the difference is more than 2000 kJ/kg clinker or 60 to 70%!

Today, with efficient dry homogenising technology available, the wet process is no longer applied for new plants. Investments as well as operating costs of a wet system are higher than for dry systems of the same output. Technical development allows using more efficient kiln systems even where wet plants would have been built in earlier times.

Another reason for preferring the wet process in the past was the production of low alkali cement (alkali content < 0,6%) and the fact that difficult circulation problems are easier to control in wet kilns. Today secondary firing or efficient bypass installations with precalciner are possibilities to keep these problems under control also in modern kiln systems.

Because of the lower specific gas volume and the shorter rotary part, rotary kiln dimensions as well as gas handling, dedusting and fuel preparation can be designed accordingly smaller. Although new wet kilns are no longer considered for new plants, they still play an important role in the US as well as in many countries of Eastern Europe and Central Asia.

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3.2 Long Wet Process Kilns

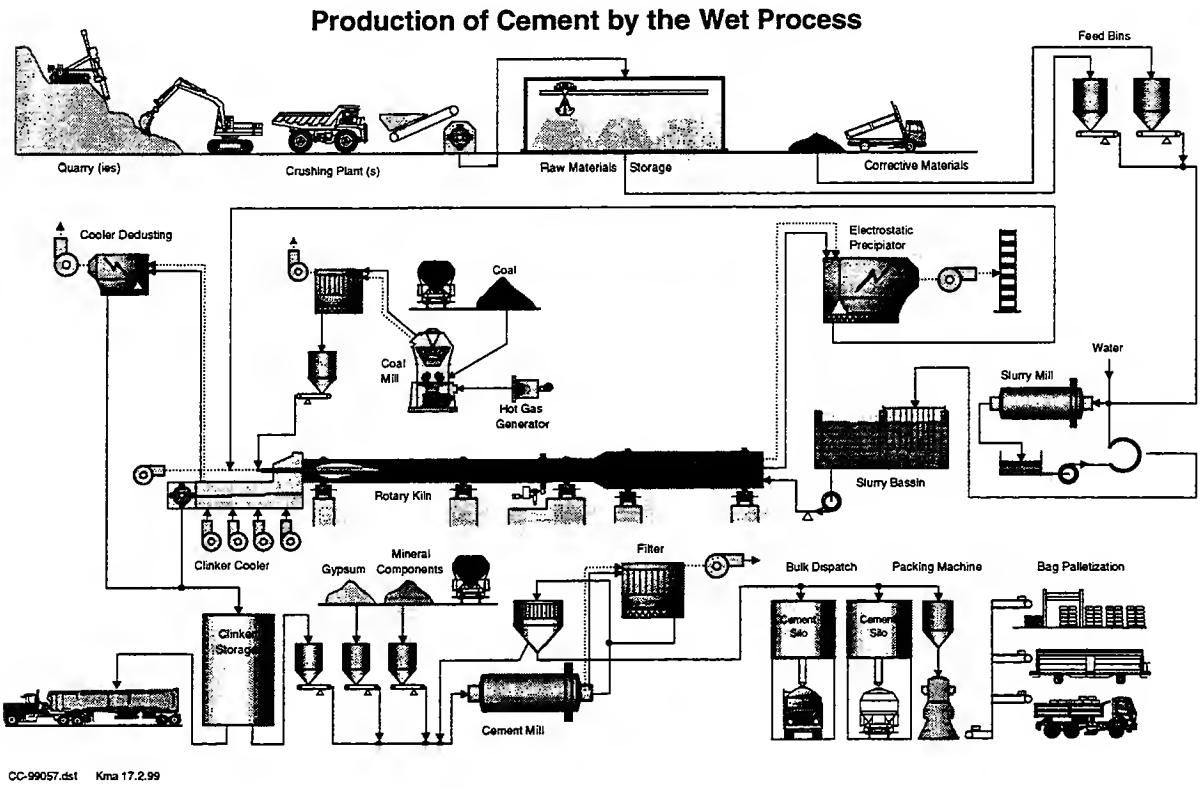
Long wet kilns have been the most commonly used burning reactors for a very long time, but because of the high water content of the feed, their heat consumption is up to twice as high as for modern dry systems.

The milled and homogenised raw material is a slurry with a water content of typically 32 to 42% and is pumped to the kiln inlet.

In the first zone heat transfer for the evaporation of water is always increased by means of chain systems (extended surface, higher relative velocities, increase of turbulence). The chain systems should also reduce the dust losses and clean the kiln shell. These internal heat exchanger installations require very special know-how, based to a large degree on experience (see separate paper 'chain systems').

In order to decrease fuel consumption the water content should be kept as low as possible. The limit is normally the pumpability of the slurry. It is basically possible to further reduce the slurry moisture by using slurry thinners. This technology has been successfully applied and will provide an economical advantage if adequate quantities are available at low cost, e.g. as industrial by-product.

Example: Beauport (Canada): 28% feed moisture



Wet kilns are relatively insensitive to circulation problems because the critical temperature ranges are in the rotary part of the kiln (see also 'circulation phenomena').

Low alkali clinker can be produced from high alkali raw material simply by selectively wasting of dust: The highest enriched kiln dust (e.g. from the last precipitator compartment) is removed from the process (i.e. dumped onto a dust pile) as necessary. The rest of the dust can be reintroduced to the kiln by dust scoops or insufflation into the burning zone.

Today, discarding dust creates increasing problems because of restrictive permitting of dust piles.

Note:
Kiln dust cannot just be blended to the slurry because it would react and thicken the slurry.

Typical technical data for long wet kilns with chains:

Heat consumption q	5'000 ... 6'300 kJ/kg cli (1'200 ... 1'500 kcal/kg cli)
Kiln exit gas temperature	150° ... 250°C
System pressure drop	0,5 ... 1,0 kPa
Dust emission in % of clinker production	5 ... 100%

Probably the largest wet process kiln in the world is installed at Holnam's Clarksville plant (Michigan USA). This kiln has a diameter of 7,6 m and a length of 232 m with a daily capacity of about 3'600 t.

3.3 Wet Process Kilns with Slurry Preheaters

External Slurry Preheaters

In order to improve the heat exchange between gas and slurry and to reduce the kiln size, external slurry preheaters have been developed by MIAG (Kalzinator) and Krupp (Konzentrator). Both of them are revolving drums with special internal packing. These drums have about the same diameter as the kiln, its length being slightly smaller than the diameter. The capacity of these machines is limited to 800 -1000 t/d and frequently operating problems arise. Very often, external preheaters were large sources of false air.

Internal Slurry Preheaters

F.L. Smidth designed a slurry pre-heater system integrated into the kiln compartment, which should avoid the disadvantage of external slurry preheaters. In practice, this construction turned out to be very sensitive to clogging.

A better system developed by Fives Cail Babcock is installed in the three kilns at Obourg. Lifting buckets and chain curtains produce a slurry curtain that keeps back a high amount of dust and improves heat exchange.

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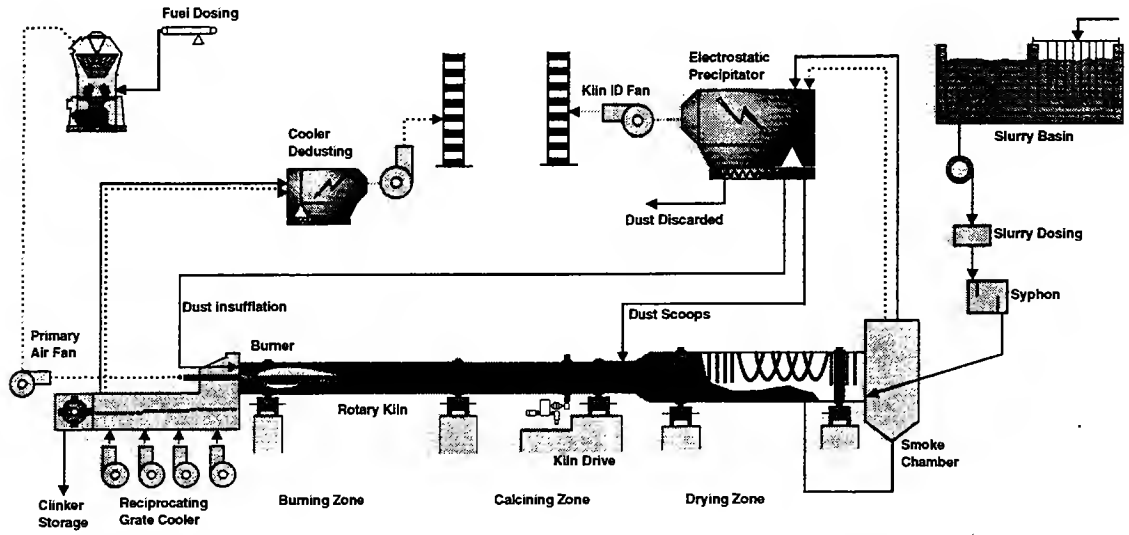
Wet process kiln



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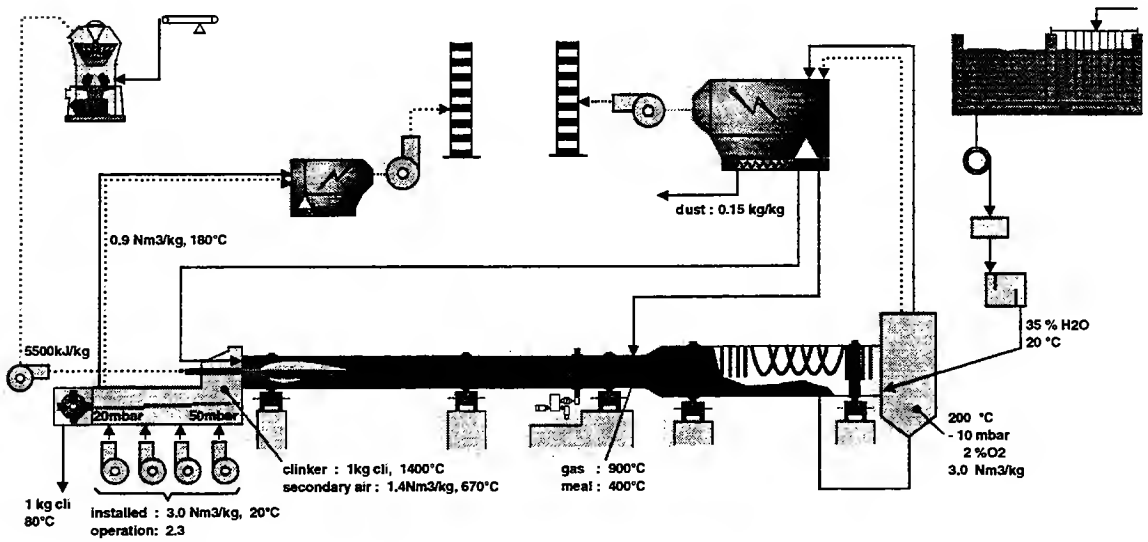
Process Technology

Wet process kiln



Capacity	:	100	- 3600	t/d	Kiln feed	:	slurry, 28 - 43	% H2O
Diameter	:	2.5	- 7.5	m	Kiln circumferential speed	:	10 - 20	m/min
Length	:	40	- 230	m	Kiln slope	:	2.5 - 4	%
Length/Diameter	:	30	- 38		Total chain weight	:	10 - 15	% rel. to capacity
Kiln load (inside lining)	:	0.4	- 0.8	t/m3d	Dust losses	:	0.05 - 0.25	kg/kg cli
BZ load (inside lining)	:	40	- 100	t/m2d	Spec.heat consumption	:	5000 - 7000	kJ/kg cli

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4. SEMI WET PROCESS

4.1 General

A process is considered semi-wet if the kiln feed is produced from wet slurry. A mechanical water extraction process reduces the water content of the kiln feed to 17 to 21%. A number of filter presses operating batch-wise are commonly used, but also continuous filter band presses or similar equipment would be possible.

4.2 Semi Wet Process Long Kilns

Principally, long kilns with heat exchanger crosses can be fed with slurry, filter cakes or dry meal. Feeding filter cakes is a straightforward and simple solution and is used by Italcementi in some cases.

4.3 Semi Wet Grate Preheater Kilns

Most of the semi-wet systems use a grate preheater kiln fed with filter cakes.

A grate preheater system includes a short rotary kiln (similar to a four stage preheater kiln) where only calcining and sintering take place. For drying, preheating and partial calcining, a travelling grate is installed in front of the kiln, where heat of the kiln exhaust gases is used.

For the semi-wet grate kiln, the slurry must be prepared in a special way so it can be fed to a travelling grate:

The pumpable slurry as starting material is fed to filter presses where the moisture content is reduced to approx. 20% applying a filtration pressure of 15 to 20 bar. In a special type of extruder (Siebkneter), the filter cakes are converted into cylindrical nodules (diameter 15 ... 20 mm, length 30 ... 50 mm) and then fed to the preheater-grate. The economy of this way of preparation depends strongly on the filtration properties of the slurry.

Operating and performance data are similar to the semi-dry grate preheater systems described under 5.2.

Characteristic data of a semi-wet grate pre-heater system:

Feed Nodules made from Moisture Content of the Feed	Slurry Filter Cake 10 ... 12%
Heat consumption q	3770 kJ/kg cli (□ 900 kcal/kg cli)
Exit gas temperature after grate	100° ... 120°C
System pressure drop	2,6 kPa

Example of a semi-wet LEPOL kiln:

AB's kiln 10 at the Lägerdorf plant (Germany)

Maximum kiln capacity: *3'600 t/d*

Kiln dimensions: *ϕ 6.0/5.6 m x 90 m*

Grate dimensions: *5.6 x 61.7 m*

Secondary firing with Fullers earth (special)

(Shut down; replaced by semi wet precalciner kiln in 1996)

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Production of Cement by the Semi-Wet Process

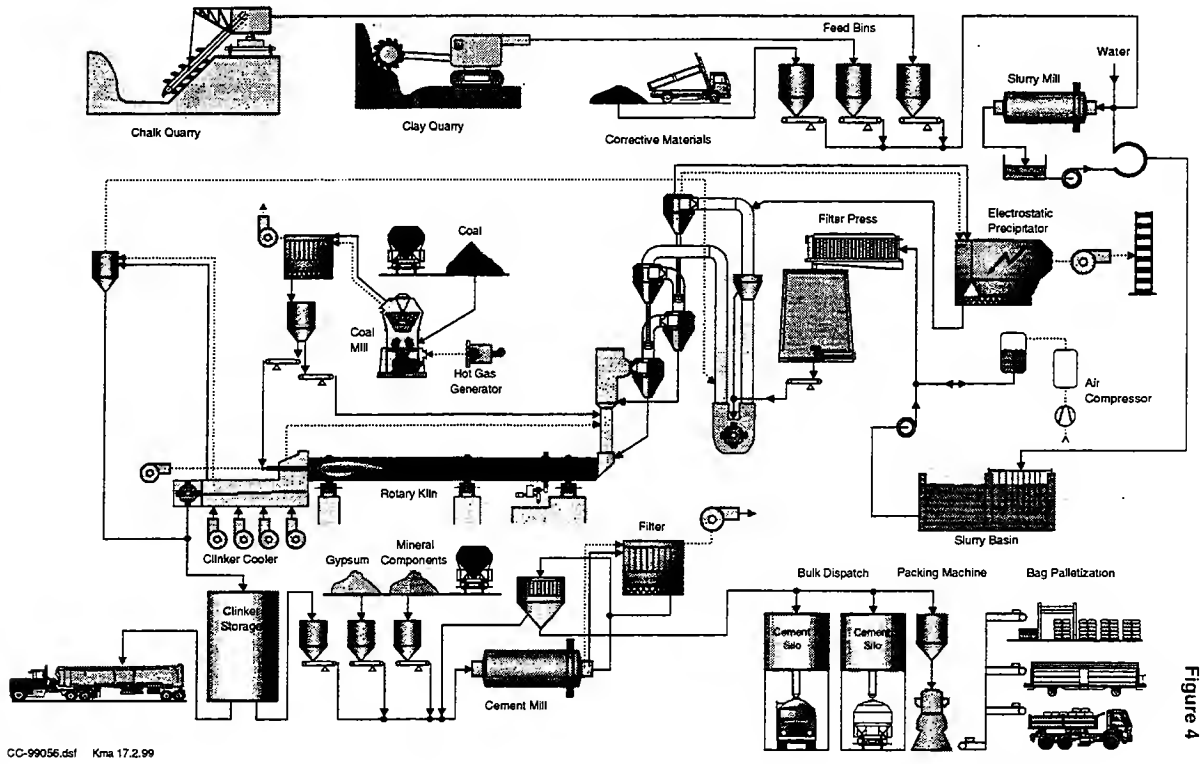

Figure 4

4.4 Semi-Wet Suspension Pre-heater Kiln

The suspension preheater kiln is normally fed with dry meal (details see separate paper). However, there are some rare cases where suspension preheater kilns are fed with nodules prepared from slurry. These nodules should not be too strong because they must be cracked by thermal shock or abrasion before being fed to the kiln system via top stage of the pre-heater.

A two-stage pre-heater kiln operated with semi-wet nodules was e.g. the Liesberg plant. There, the nodules were cracked in a vertical dryer before being fed to the preheater.

The first modern kiln system using this principle has been built in the late 1980's by FLS in Aalborg Cement's RORDAL plant. It is a three stage two string kiln system with precalciner for a capacity of 4000 t/d. The high operating cost of the filter presses has been avoided by directly injecting the slurry into a drier-crusher followed by a vertical drier. The semi-wet process was selected because the raw material (chalk) is mined under water and has very high natural moisture.

From the "Holderbank" group:

Example of a semi-wet pre-heater/pre-calciner kiln:

AB's kiln 11 at the Lägerdorf plant (Germany)

Maximum kiln capacity: *4'500 t/d at 3900 kJ/kg*

Kiln dimensions: *ϕ 4.8 x 65 m; 2 supports, gearless friction drive*

Preheater: *3 stages, 2 strings*

Utilisation of various alternative fuels in both firings

Supplied by Polysius; start-up: 1996

Filter cakes produced in already existing filter-presses

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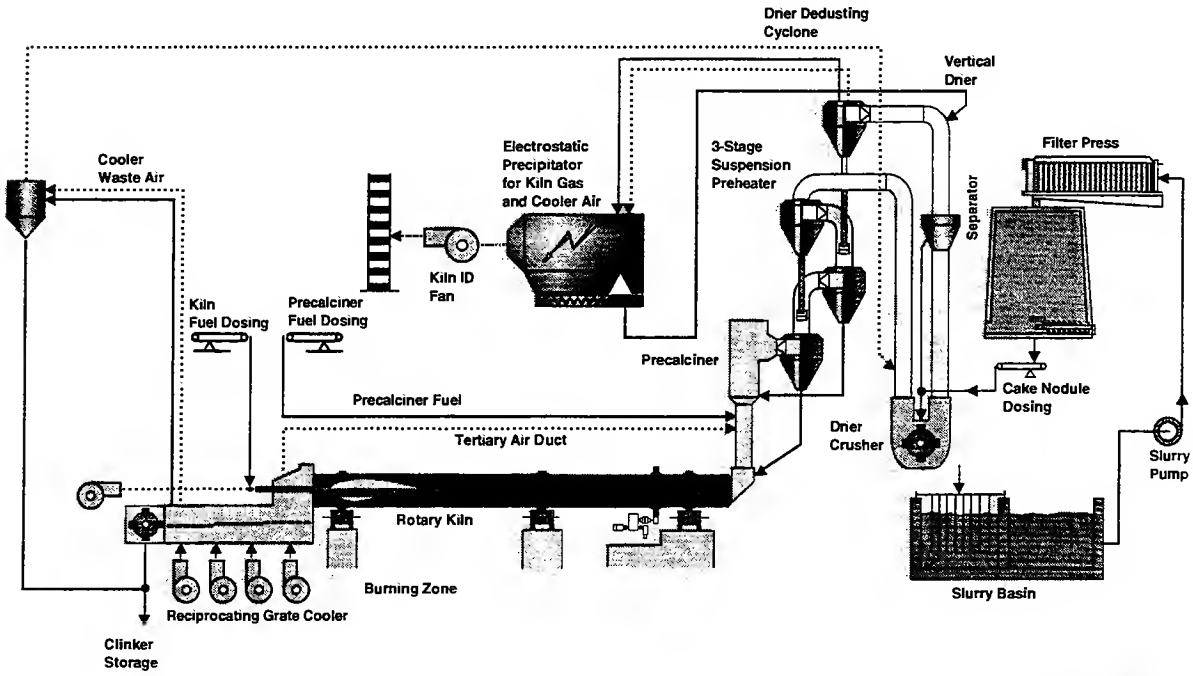
Semi wet process kiln (with 3-stage SP and PC)



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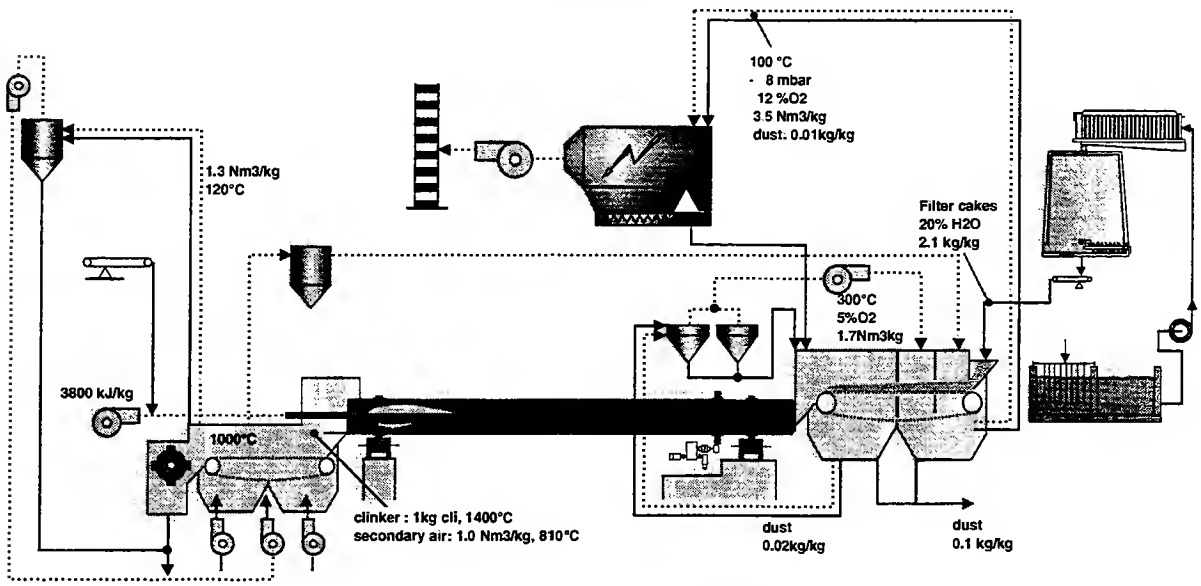
Process Technology

Semi wet process kiln (with grate preheater "LEPOL")



Capacity	:	300	-	3000	t/d	Kiln feed	:	filter cakes, 18 - 23 % H2O
Diameter	:	3	-	5.6	m	Kiln speed	:	1 - 2 rpm
Length	:	45	-	90	m	Kiln slope	:	3 - 3.5 %
Length/Diameter	:	11	-	16		Spec.heat consumption	:	3600 - 4500 kJ/kg cli
Kiln load (inside lining)	:	1.5	-	2.2	t/m3d			
BZ load (inside lining)	:	80	-	150	t/m2d			
Grate load	:	8	-	10	t/m2d			

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5. SEMI DRY PROCESS

The semi-dry process is characterised by the fact that kiln feed nodules are made from dry raw meal. Water is added in order to produce nodules with 10 - 12% moisture.

5.1 Semi-Dry Process Long Kilns

There are long kilns with heat exchanger crosses fed with nodules. This system was applied by Italcementi and looks very similar to an installation for semi-wet feed material.

5.2 Semi-Dry Process Grate Pre-heater Kilns

The grate preheater kiln is by far the most popular semi-dry system.

The principle of the grate preheater system for the semi-dry process is identical to the one used for the semi-wet process. What is different is the feed preparation:

The dry raw material is mixed with water (10 ... 12%) and nodulised in a drum or preferably on a rotating plate (pan noduliser). This system can be used only for raw materials containing plastic components enabling the formation of nodules that are resistant against thermal shock and abrasion. The main factor influencing plasticity is the mineralogical composition, especially the presence of montmorillonite.

On the grate, heat exchange from the gas to the nodules forming a fixed bed layer of approx. 20 cm thickness is excellent. In some grate preheaters, precalcination is done successfully, often using even waste fuels (such as Fullers earth, acid sludge, waste lubricating oils etc.) utilising secondary firing.

The only successfully working travelling grate pre-heater was available from Polysius and became known under the name LEPOL system (American licensee: Allis-Chalmers, ACL system).

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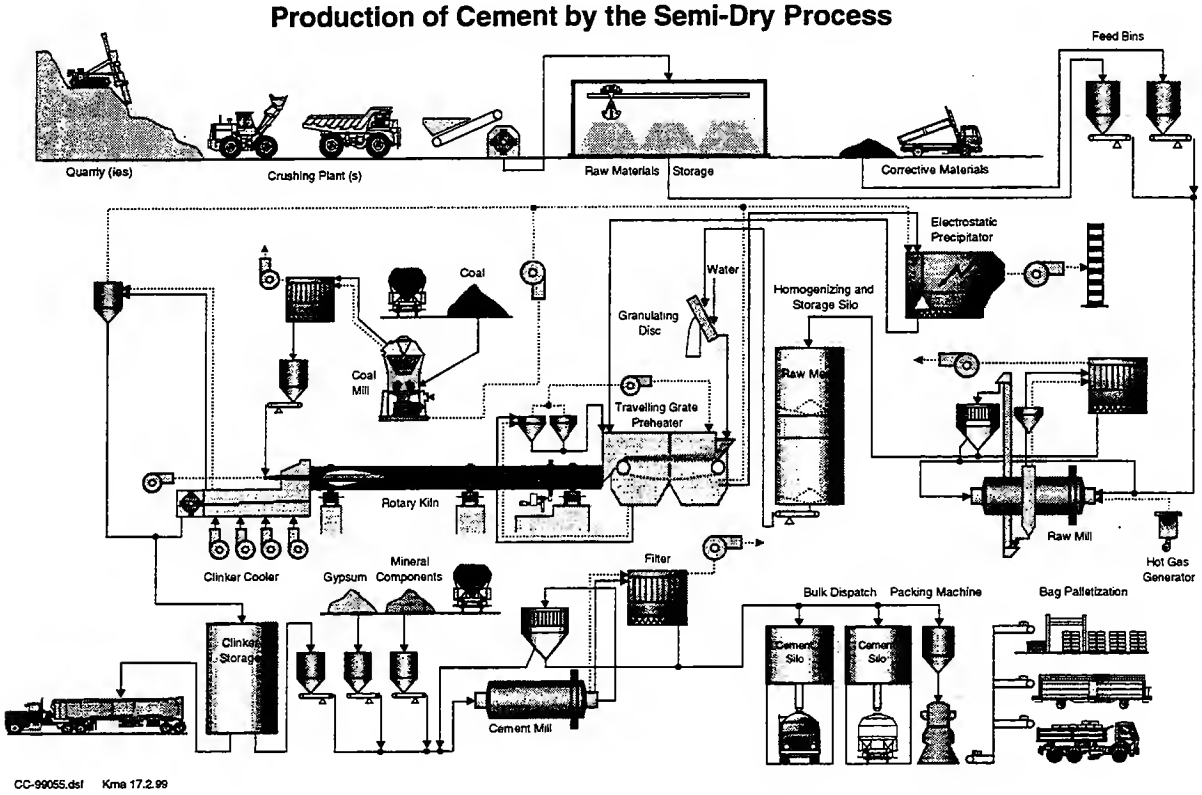
Production of Cement by the Semi-Dry Process



This principle sketch shows a LEPOL kiln fed with nodules made out of dry raw meal. LEPOL kilns built after 1945 are equipped with two-pass grates; i.e. the exhaust gas is led twice through the nodule bed from top to bottom:

The hot kiln gas passes first through a bed of dry and preheated nodules and subsequently, after an intermediary dedusting once again trough a layer of moist incoming nodules. It is believed that the nodules survive throughout the process resulting in a clinker with very uniform size.

Furthermore, dust loads in the kiln atmosphere and dust emission out of the system are low. The nodules on the grate let only pass the fine dust while the coarse particles are retained.

In cases of increased trace compound concentrations (especially alkali) in the raw material, the fine dust separated in the electrostatic precipitator is largely enriched with them. Only a small amount of dust has to be discarded to reduce the balance of these compounds in the kiln system. This effect makes the LEPOL kiln quite suitable to produce a low alkali clinker with rather low heat consumption. For this reason, it has been chosen in many cases, particularly in the USA.

The following limits and disadvantages have to be considered:

- Only raw materials with good plastic properties can be used (semi-wet: filter cake nodules -> good filtration properties are required)
- The grate chain is subject to wear.
- Uneven temperature distribution across the grate can cause difficulties.
- Additional theoretical heat consumption due to the water content of the feed (partially compensated by a low exit gas temperature).
- Exhaust gases cannot be used in drying and grinding systems.

Characteristic data of a semi-dry grate pre-heater systems:

Feed nodules made from	dry raw meal
Moisture content of the feed nodules	10 ... 12%
Specific heat consumption q	3450 kJ/kg cli (= 820 kcal/kg cli)
Exit gas temperature after grate	100 ... 120°C
System pressure drop	2.6 kPa

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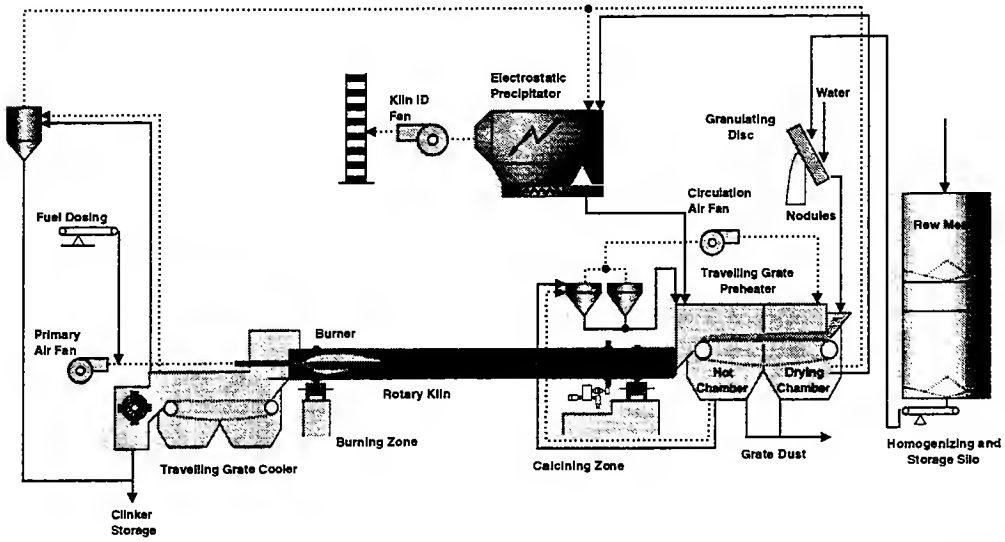
Semi dry process kiln (with grate preheater "LEPOL")



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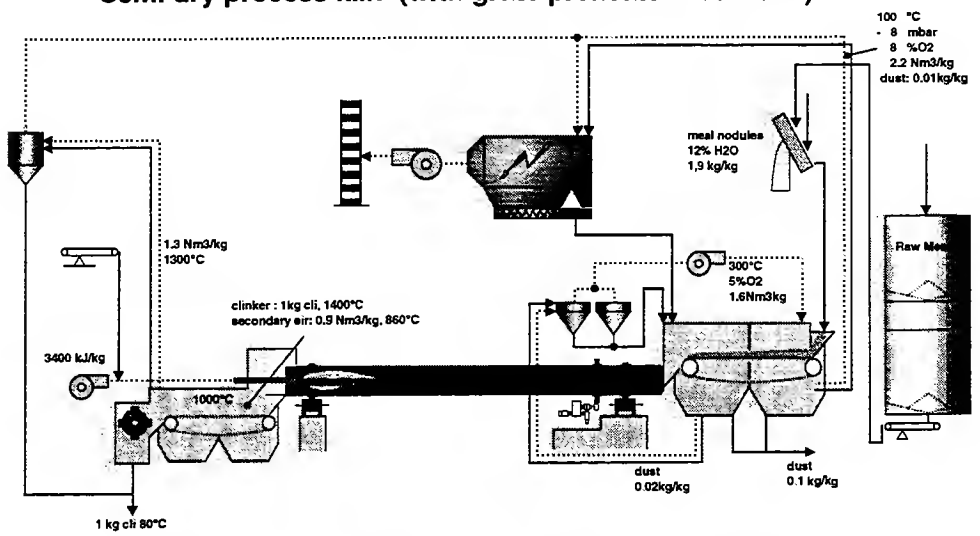
Semi dry process kiln (with grate preheater "LEPOL")



Capacity	:	300	- 2000	t/d	Kiln feed	:	meal nodules, 11 - 14	% H2O
Diameter	:	3	- 4.8	m	Kiln speed	:	1 - 20	rpm
Length	:	35	- 80	m	Kiln slope	:	3 - 4	%
Length/Diameter	:	11	- 16		Spec.heat consumption	:	3200 - 3600	kJ/kg cli
Kiln load (inside lining)	:	1.5	- 2.2	t/m3d				
BZ load (inside lining)	:	80	- 150	t/m2d				
Grate load	:	80	- 12	t/m2d				

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6. DRY PROCESS

6.1 Long Dry Kilns

Without internal heat exchange equipment

The simplest kind of dry process installation is the long dry kiln without any internal heat exchange equipment (empty tube). With a heat consumption of 5100 kJ/kg cli (1200 kcal/kg cli) or about 90% of the wet process it must be considered very uneconomical. Advantages might be its simplicity and insensitivity to heavy circulation problems.

This kiln type is suitable to be used in combination with waste heat recovery steam boilers for power generation. In that case, the waste heat contained in the hot kiln exhaust gases is further used to produce valuable energy.

Characteristic kiln data:

Heat consumption q	4500 ... 6000 kJ/kg cli	(1075 ... 1430 kcal/kg cli)
Kiln gas exit temperature	450° ... 500°C	
System pressure drop	0,5 ... 1,0 kPa	

With internal heat exchange equipment

Long dry kilns with internal heat exchange equipment (chains or crosses from steel or ceramic material) represent a more economical solution. Heat consumption of 4200 kJ/kg or even less can be achieved. Other typical operating figures are contained in annex 10.

Characteristic kiln data:

Heat consumption q	3800 ... 4500 kJ/kg cli	(910 ... 1075 kcal/kg cli)
Kiln gas exit temperature	400° ... 450°C	
System pressure drop	1,0 ... 1,5 kPa	

6.2 Raw Meal Suspension Preheater Kilns

6.2.1 General

During the last thirty years, the suspension preheater kiln became the dominant clinker manufacturing system. This system is fed by dry raw meal that is preferably prepared in a grinding and drying plant, using the kiln waste gases as a drying medium. This ground and dried raw meal is homogenised and then fed to the preheater where it is suspended in the kiln gas flow, where an extremely effective heat transfer takes place. More information is contained in the special section "Suspension Preheaters".

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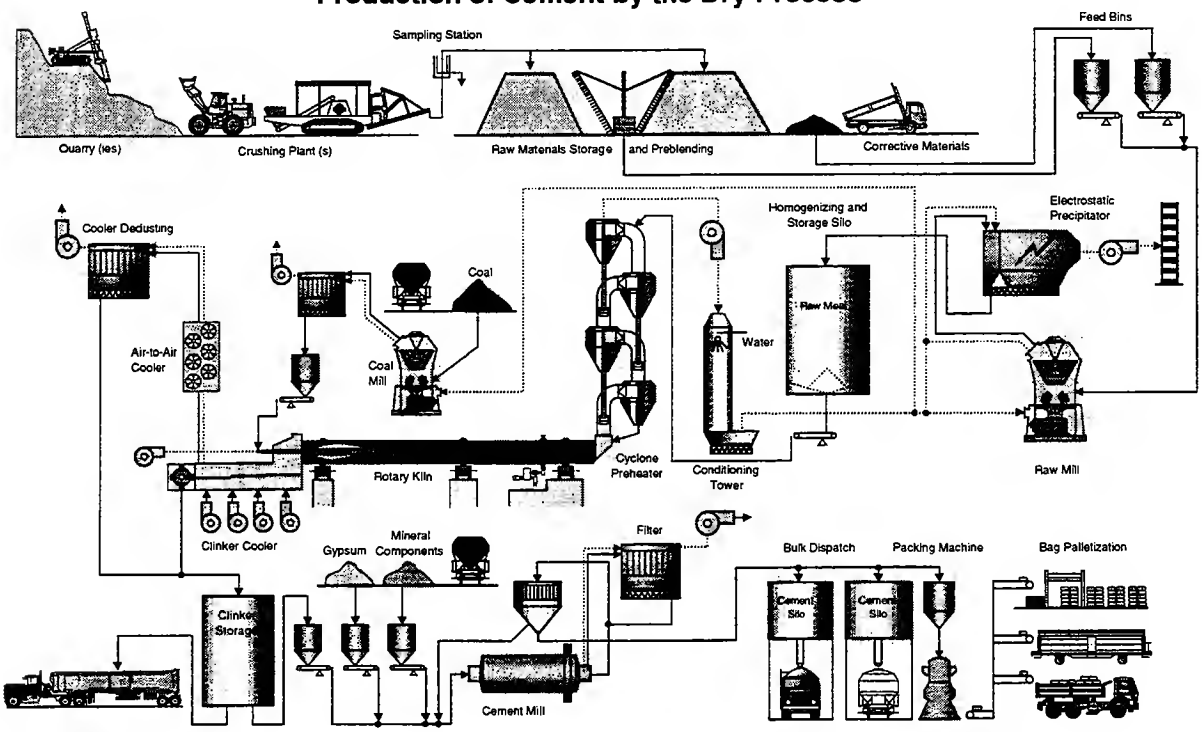
6.2.2 One and two Stage Cyclone Pre-heater Kilns

Characteristic kiln data:

one stage:	Heat consumption q	3750 ... 4000 kJ/kg cli	(900 ... 950 kcal/kg cli)
	Kiln gas exit temperature	400° ... 500°C	
	System pressure drop	1,5 ... 2,5 kPa	

two stages:	Heat consumption q	3500 ... 3750 kJ/kg cli	(850 ... 900 kcal/kg cli)
	Kiln gas exit temperature	400° ... 450°C	
	System pressure drop	1,5 ... 2,5 kPa	

Production of Cement by the Dry Process



6.2.3 Four Stage Cyclone Pre-heater Kilns

Until the mid 1980s, this arrangement belong to the systems with the lowest fuel consumption. It was offered in several configurations with capacities up to 4500 t/d, most of them being combinations of single or twin cyclone stages.

The kiln exit gas includes still enough heat to dry raw material up to moisture content of 8% if the mill is running during all the kiln operation time. From this point of view, the remaining relatively high exit gas temperature cannot be considered fully as a loss, because it can substitute an auxiliary firing for raw material drying.

The preheater system is installed in a steel or concrete tower with a height of about 60 to 120 m (6 stages) above the kiln inlet, depending on capacity and concept.

The four to six stages preheater is most susceptible to circulation problems at presence of excessive concentration of circulation compounds causing clogging problems in the pre-heater system.

The sketch shows a conventional four stage cyclone preheater system. In the 1970's, production lines with more than approx. 2000 t/d had to be built with two parallel preheater strings. Today, one-string installations are possible for up to 4000 t/d.

Characteristic operating figures of 4-stage pre-heater kilns:

Heat consumption q	
small units	3350 ... 3550 kJ/kg cli (= 800 ... 850 kcal/kg cli)
large units	3150 ... 3350 kJ/kg cli (= 750 ... 800 kcal/kg cli)
kiln exit gas temperature	320° ... 350°C
kiln exit gas volume	approx. 1,5 Nm3/kg cli
System pressure drop	4 ... 6 kPa
Dust loss relative to clinker	8 ... 15%
Transition chamber	
kiln gas temperature	approx. 1100°C
Material temperature	approx. 800°C

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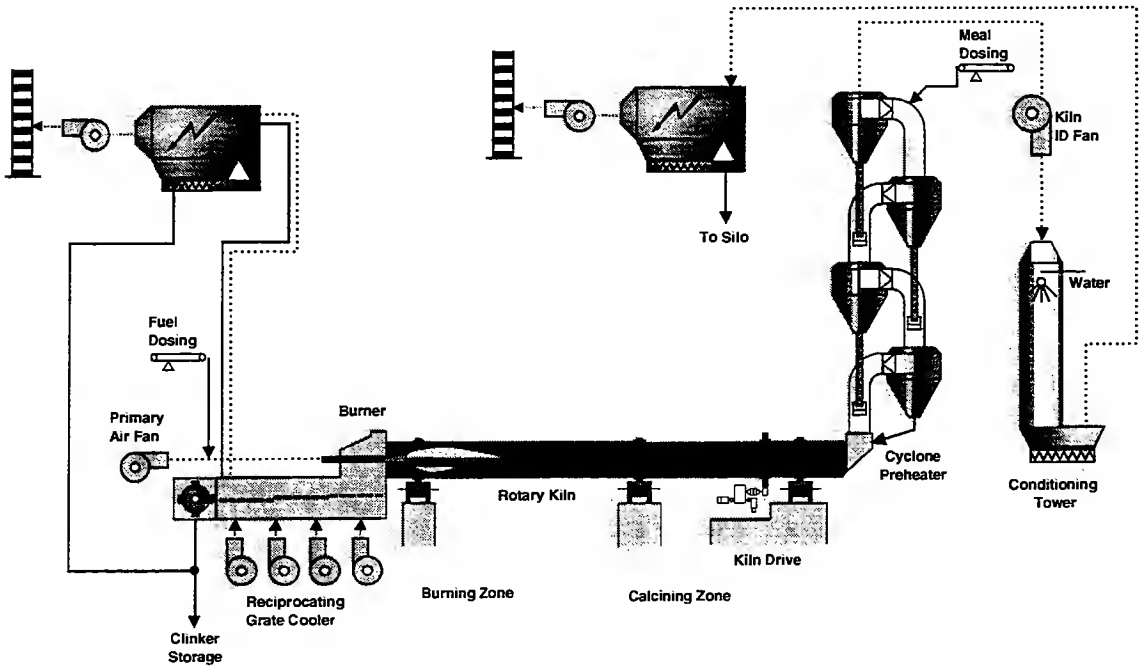
Dry process kiln (with 4-stage SP)



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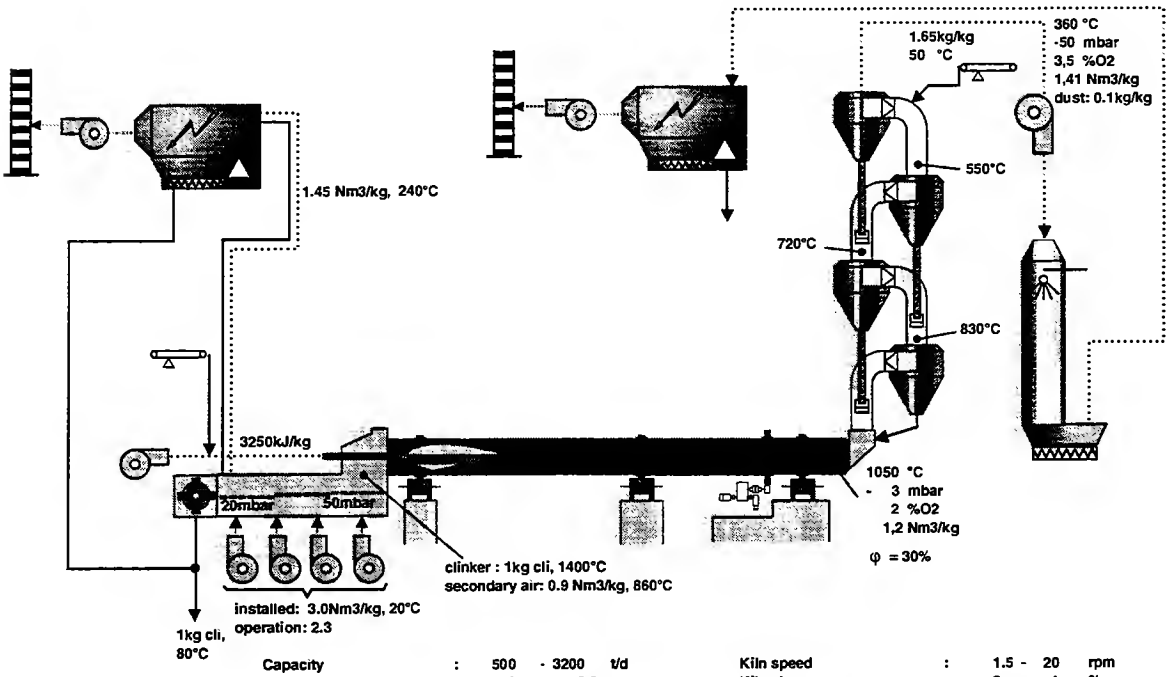
Process Technology

Dry process kiln (with 4-stage SP)



Capacity	:	500	- 3200	t/d	Kiln speed	:	1.5 - 20	rpm
Diameter	:	3.5	- 5.5	m	Kiln slope	:	3 - 4	%
Length	:	50	- 90	m	Spec.heat consumption	:	3150 - 3500	kJ/kg cli
Length/Diameter	:	14	- 16					
Kiln load (inside lining)	:	1.5	- 2.2	t/m3d				
BZ load (inside lining)	:	100	- 180	t/m2d				

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6.2.4 Precalciner Kilns

For larger production capacities, a larger portion of the pyroprocess had to be relocated out of the rotary kiln in order to maintain reasonable kiln diameters without excessive thermal load of the burning zone.

The process of dissociation of CO2 (calcination) is suitable to take place in a static reactor outside of the rotary kiln. Of the total heat consumption, 60 to 65% are required to achieve about 90% of calcination. 100% calcination must be avoided because clogging problems will seriously disturb kiln operation (beginning of clinker formation).

The development of this reactor started with a secondary firing in the kiln riser duct sufficient for 35 to 40% calcination of the meal, combustion air still pulled through the kiln tube (=air through). It was therefore referred to as precalciner (PC) type AT. Only when hot cooler air (= tertiary air) for the PC fuel (= secondary fuel) was taken to the calciner in a separate duct, the so called tertiary air duct, the full benefit of this technology could be used. Today, only this type called PC-AS (=air separate) is considered a real precalciner. The elements of a precalciner kiln system are explained in the sketch.

The strongest boost of calciner development was in the seventies in Japan, initiated by the demand for very large units exceeding the potential of conventional kilns with suspension preheaters. Only precalciner technology makes today's largest units of 10'000 t/d possible.

Two process alternatives of precalciner are used:

- in-line calciner (calciner installed in kiln gas flow)
- separate-line calciner (calciner not passed by kiln gases)

More details on calciner technology are contained in a separate section.

The operating data are very close to the ones of the corresponding preheater kiln system. In-line calciners have a tendency to higher gas exit temperature and system pressure drop; however, modern units are equipped with 5 or 6 preheater stages to compensate for this.

Characteristic operating data of 4 to 6 stage precalciner kilns:

Heat consumption q	
small units, 4 stage SP	3350 ... 3550 kJ/kg cli (= 800 ... 850 kcal/kg cli)
large units, 5 stage SP	2900 ... 3200 kJ/kg cli (= 700 ... 800 kcal/kg cli)
SP exit gas temp. 6 to 4 st. SP	290° ... 370°C
SP exit gas volume	approx. 1.3 to 1.5 Nm3/kg cli
System pressure drop	4 ... 6 kPa
Dust loss relative to clinker	8 ... 15%
Transition chamber:	
kiln gas temperature	approx. 1100°C
Material temperature	approx. 800°C

More data of precalciner kiln systems are shown in the section "Precalciners".

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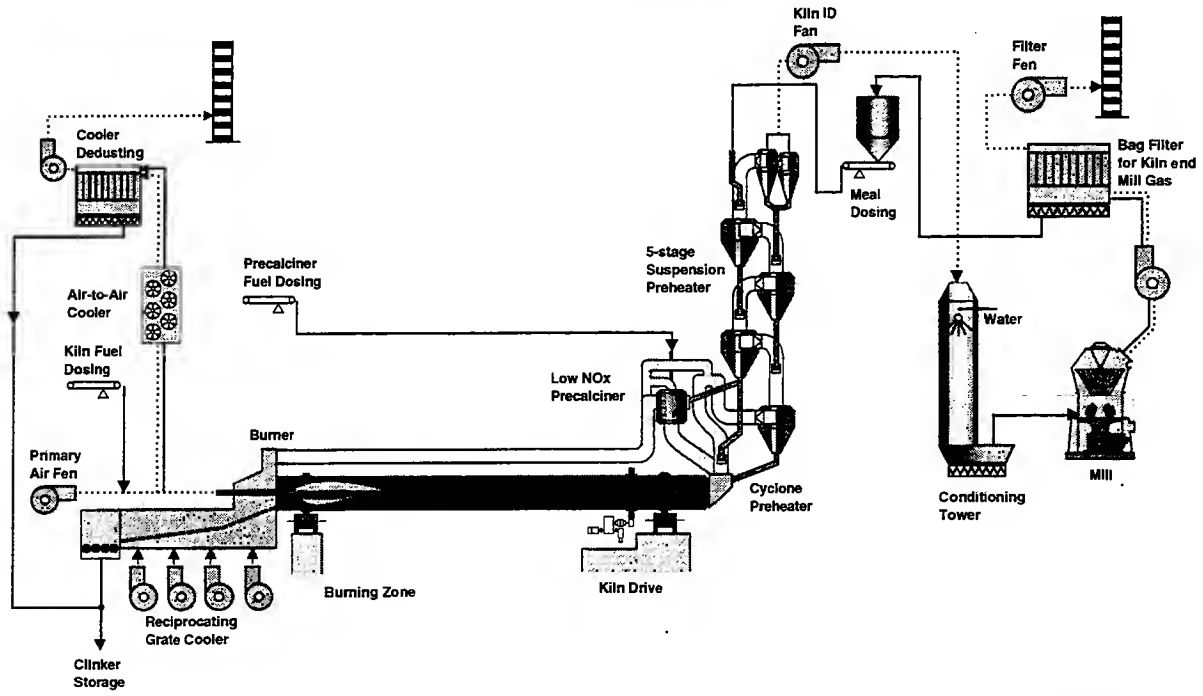
Dry process kiln (with 5-stage SP and PC)



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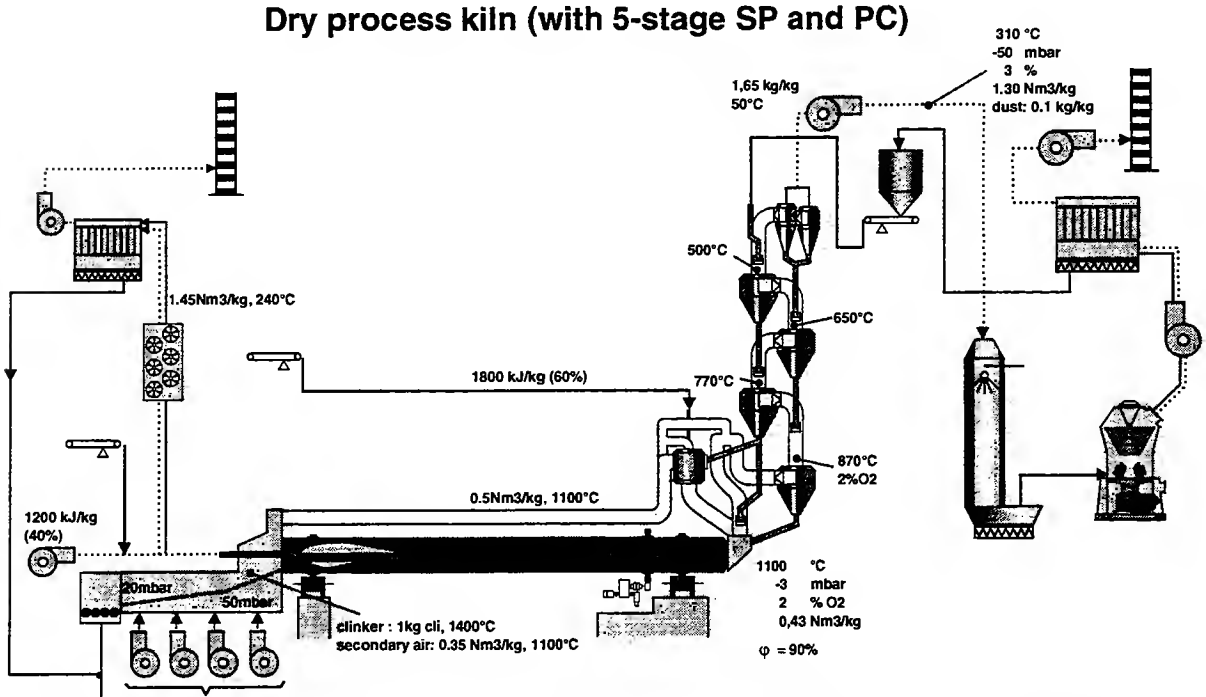
Process Technology

Dry process kiln (with 5-stage SP and PC)



Capacity	:	1500	- 10'000	t/d	Kiln speed	:	2.5 - 4.5 rpm
Diameter	:	3.5	- 6.2	m	Kiln slope	:	3.5 - 4 %
Length	:	40	- 105	m	Spec.heat consumption	:	2900 - 3500 kJ/kg cli
Length/Diameter	:	10	- 16				
Kiln load (inside lining)	:	3	- 5.5	t/m3d			
BZ load (inside lining)	:	185	- 350	t/m2d			

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HEAT BALANCE

WET / SEMI-DRY / 4-ST. PREHEATER / 5-ST. PREHEATER-PRECALCINER

	WET PROCESS		SEMI-DRY LEPOL		4-STAGE SP		6-STAGE SP-PC	
Input	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%
Fuel kiln combustion	**5560**	96.7%	**3343**	97.6%	**3150**	97.7%	**1180**	39.2%
sensible heat	25	0.4%	15	0.4%	13	0.4%	5	0.2%
Fuel PC combustion	0	0.0%	0	0.0%	0	0.0%	**1775**	58.9%
sensible heat	0	0.0%	0	0.0%	0	0.0%	8	0.3%
Kiln feed sensible heat	25	0.4%	30	0.9%	54	1.7%	45	1.5%
sensible heat of water	73	1.3%	17	0.5%	0	0.0%	0	0.0%
Insufflated air (PA, cooler)	67	1.2%	20	0.6%	6	0.2%	0	0.0%
Total inputs	**5750**	100%	**3425**	100%	**3223**	100%	**3013**	100%

Output	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%
Heat of formation	1750	30.4%	1750	51.1%	1750	54.3%	1750	58.1%
Water evaporation	2370	41.2%	506	14.8%	13	0.4%	8	0.3%
Exhaust gas sens. heat	754	13.1%	314	9.2%	636	19.7%	553	18.4%
Exhaust gas dust sens. heat	25	0.4%	21	0.6%	18	0.6%	29	1.0%
Clinker	59	1.0%	50	1.5%	63	2.0%	83	2.8%
Cooler waste air	100	1.7%	276	8.1%	423	13.1%	288	9.6%
Radiation and convection :								
- Preheater	0	0.0%	160	4.7%	77	2.4%	60	2.0%
- Precalciner (or bottom stage)	0	0.0%	0	0.0%	20	0.6%	20	0.7%
- Kiln (+tertiary air duct)	530	9.2%	200	5.8%	200	6.2%	200	6.6%
- Cooler	10	0.2%	92	2.7%	10	0.3%	10	0.3%
Water cooling	0	0.0%	42	1.2%	0	0.0%	0	0.0%
Other outputs	0	0.0%	0	0.0%	0	0.0%	0	0.0%
Rest	152	2.6%	14	0.4%	13	0.4%	12	0.4%
Total outputs	**5750**	100%	**3425**	100%	**3223**	107%	**3013**	100%

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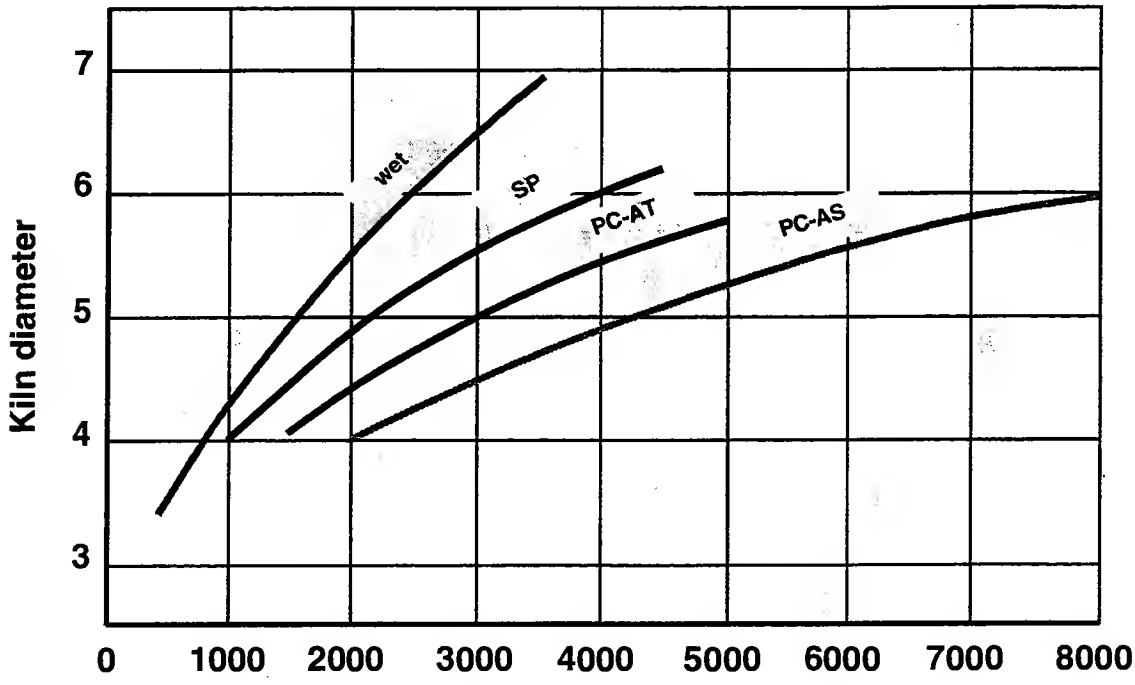

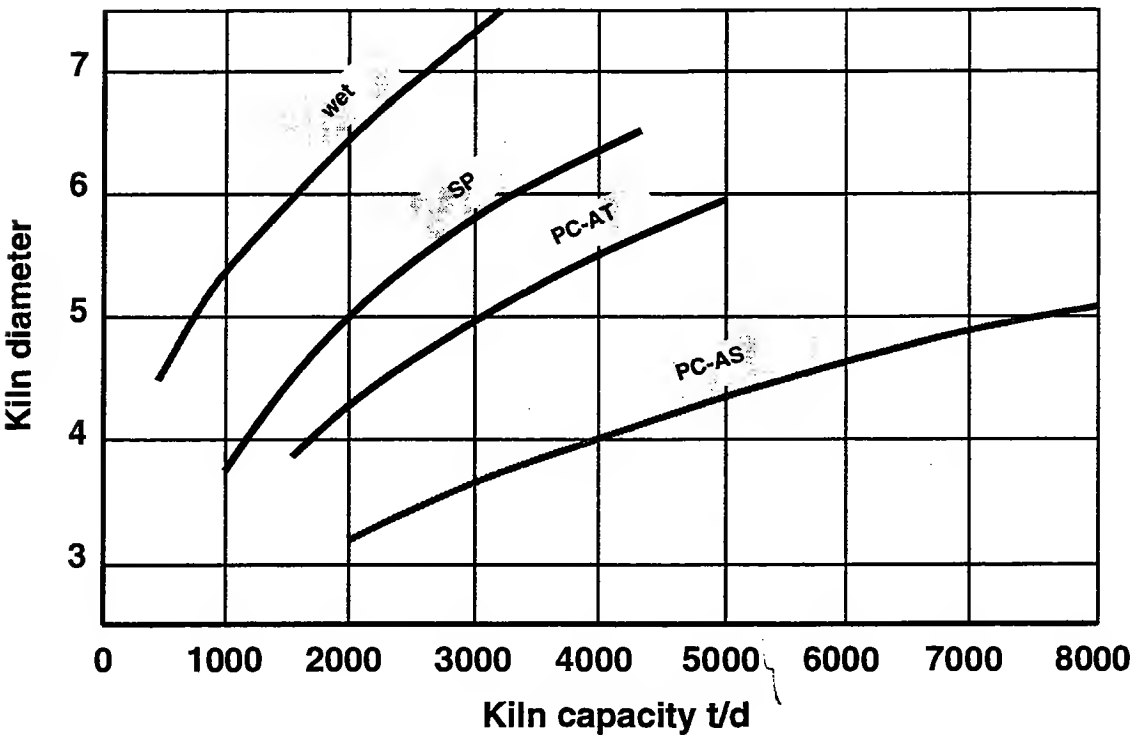

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HISTORICAL DEVELOPMENT

Annex 1

The word cement is more than 2000 years old, but impure lime has been used much longer as a building material. It is historically established, that the Phoenicians used a pozzolanic lime about 700 B.C. and also the Romans produced some sort of cement or hard burned lime. From the medieval ages, it is known that in Holland a type of hydraulic cement was formed out of lime and tuff in dome shaped kilns.

Our cement, as we know it today, is now more than **200 years** old, "invented" by the Englishman **John Smeaton in 1756.** It was burned in bottle kilns. The better known inventor of Portland cement was **Joseph Aspdin,** who patented his burning process in **1824.** He also used dome kilns of approx. 36 ft height and 17 ft diameter with a production of 90 bbl (= 15 t) per charge, each of which took several days to produce. Fuel consumption was 50% of clinker weight in coal which corresponds to 15'500 kJ/kg cli (= 3'700 kcal/kg cli).

In **1880** an important step forward was made with the development of the **continuously** working shaft kiln, which had a much better heat economy. An example of such a kiln was the "Dietzsche Etagenofen" which is shown in Annex 1.

From **1877** experiments have been conducted with rotary kilns. In **1897** Hurry and Seaman developed the first successfully operating unit of this type in America.

These first rotary kilns were wet process kilns with a daily capacity of 50 to 100 tons. Their heat consumption was again very high (about 30% of clinker in coal = 9'500 kJ/kg cli) and they had an incredible dust emission (usually more than one third of the whole production). In order to decrease heat consumption, chain systems were installed in wet kilns to improve heat transfer during drying. Behind long dry kilns, waste heat steam boilers were arranged for the same purpose.

It took almost another 30 years, before a further substantial reduction of heat consumption could be achieved by reducing the water content of the feed and by a better heat exchange in the preheating a calcining zone. In **1930** an officer of the army of the **tsar, Dr. Lellep,** took an important step in this direction. He developed the travelling grate pre-heater, which is fed with moist nodules. This invention was taken over by Polysius and got the name **LEPOL** kiln. Some years later, there was a Czech patent of a cyclone raw meal pre-heater, and in **1953 Kloeckner-Humboldt-Deutz AG in Germany installed the first suspension pre-heater system for raw meal.** This type of kiln now became dominant because of its heat economy and nowadays other systems are only chosen in special cases. In former years, the main reason for the selection of the wet process was, that effective homogenisation of ground raw material was not possible except in the form of slurry. In developing special techniques for dry material homogenisation such as mix beds, mixing chamber silos etc., this factor could be eliminated.

Utilising a rather old idea, since about **1966** especially Japanese cement machine manufacturers have designed several successfully working precalcining kiln systems. Calcination is already done in a stationary calciner system, where secondary firing is installed. By this means, it is possible to design kiln systems with a comparatively small rotary part diameter but a very large capacity up to more than 10'000 t/d.

Kiln systems built after 1990 include 6-stage preheaters with up to 4000 t/d per string, pure air calciners, designed for a variety of fuels and emission control. Using modern low primary air burners, low pressure drop cyclone designs and high recuperation efficiency coolers allow further reduction of heat and power consumption.

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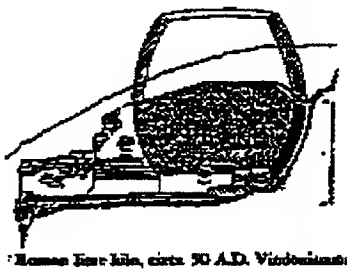
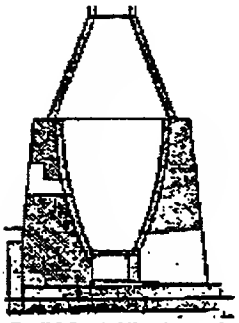
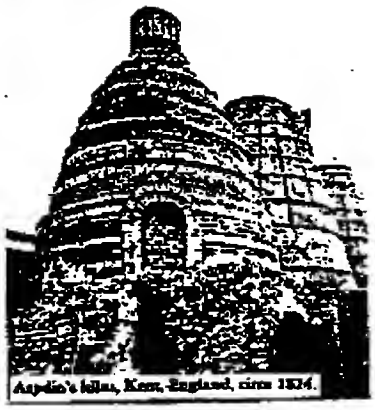


Roman lime kiln, circa 50 A.D. Vindonissa museum Switzerland.

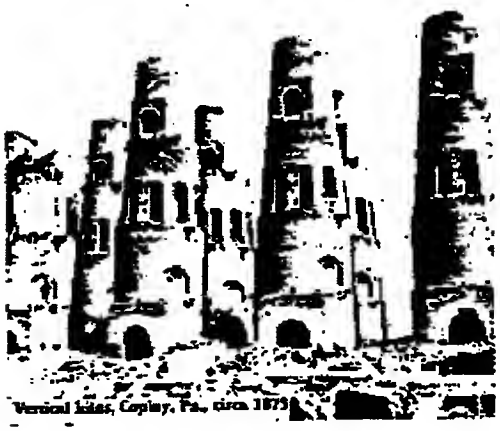
Thirteenth century burning.



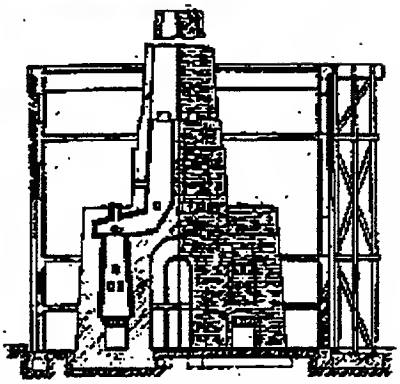
English bottle kiln, circa 1760



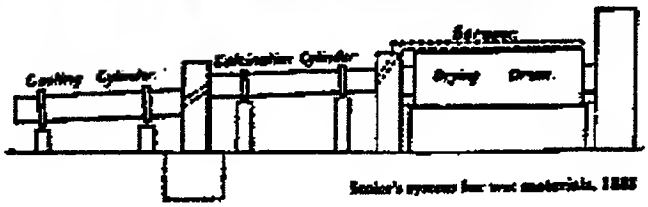
Aspdin's kilns, Kent, England, circa 1824.

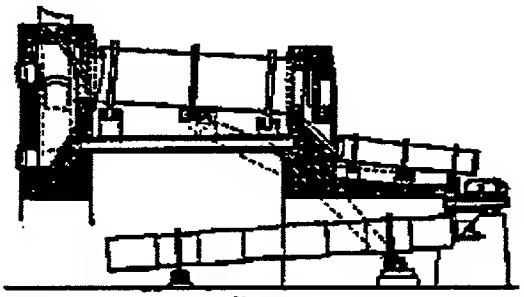


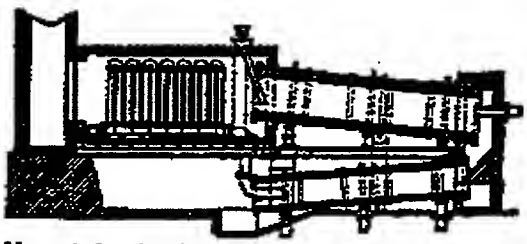
Vertical kilns, Copley, Pa., circa 1875

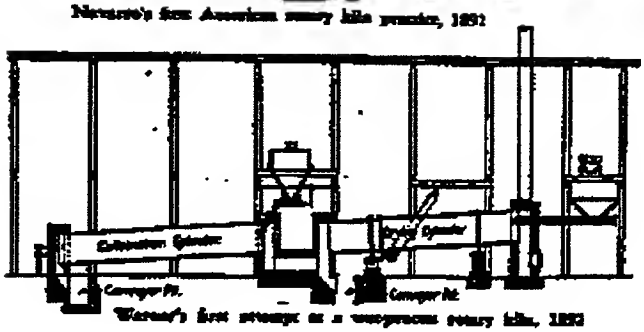


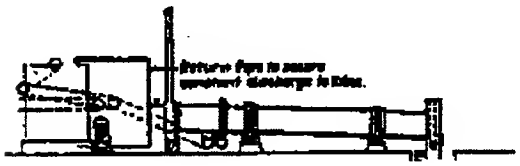
Dietzsch' multi stage shaft kiln











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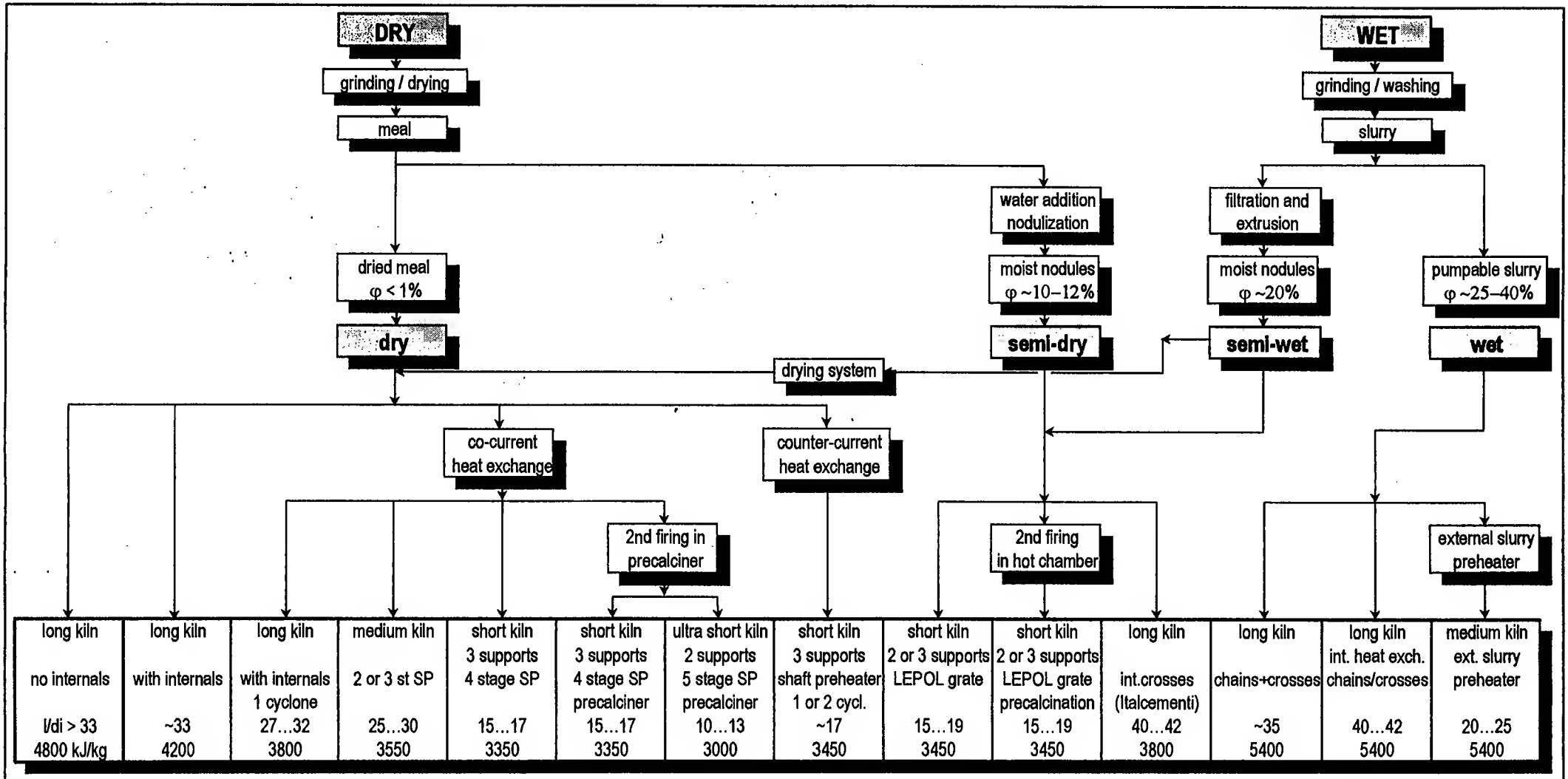

General Overview of Process Types and Rotary Kiln Systems

"HOLDERBANK"
PROCESS TECHNOLOGY

HEAT BALANCE

WET / SEMI-DRY / 4-ST. PREHEATER / 5-ST. PREHEATER-PRECALCINER

		WET PROCESS		SEMI-DRY LEPOL		4-STAGE SP		6-STAGE SP-PC	
Input		kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%
Fuel kiln	combustion	**5560**	96.7%	**3343**	97.6%	**3150**	97.7%	**1180**	39.2%
	sensible heat	25	0.4%	15	0.4%	13	0.4%	5	0.2%
Fuel PC	combustion	0	0.0%	0	0.0%	0	0.0%	**1775**	58.9%
	sensible heat	0	0.0%	0	0.0%	0	0.0%	8	0.3%
Kiln feed	sensible heat	25	0.4%	30	0.9%	54	1.7%	45	1.5%
	sensible heat of water	73	1.3%	17	0.5%	0	0.0%	0	0.0%
Insufflated air (primary, cooler)		67	1.2%	20	0.6%	6	0.2%	0	0.0%
Total inputs		**5750**	100%	**3425**	100%	**3223**	100%	**3013**	100%

Output	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%
Heat of formation	1750	30.4%	1750	51.1%	1750	54.3%	1750	58.1%
Water evaporation	2370	41.2%	506	14.8%	13	0.4%	8	0.3%
Exhaust gas sens.heat	754	13.1%	314	9.2%	636	19.7%	553	18.4%
Exhaust gas dust sens. heat	25	0.4%	21	0.6%	18	0.6%	29	1.0%
Clinker	59	1.0%	50	1.5%	63	2.0%	83	2.8%
Cooler waste air	100	1.7%	276	8.1%	423	13.1%	288	9.6%
Radiation and convection :								
- Preheater	0	0.0%	160	4.7%	77	2.4%	60	2.0%
- Precalciner (or bottom stage)	0	0.0%	0	0.0%	20	0.6%	20	0.7%
- Kiln (+tert.air duct)	530	9.2%	200	5.8%	200	6.2%	200	6.6%
- Cooler	10	0.2%	92	2.7%	10	0.3%	10	0.3%
Water cooling	0	0.0%	42	1.2%	0	0.0%	0	0.0%
Other outputs	0	0.0%	0	0.0%	0	0.0%	0	0.0%
Rest	152	2.6%	14	0.4%	13	0.4%	12	0.4%
Total outputs	**5750**	100%	**3425**	100%	**3223**	107%	**3013**	100%

Rotary Kilns

U. Gasser
PT 98/14362/E

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SUMMARY

After over 100 years, the rotary kiln is used in all cement plants for clinker production.

The following properties made it superior to other principles:

- suitable to cope with high temperatures
- easy to be lined with refractory bricks due to its shape
- material transport behaviour
- tight to ambient
- mechanically relatively simple
- large units possible

The rotary kiln must be designed for process, combustion and mechanical requirements.

Characteristic figures:	Length L [m] , diameter D [m] and their ratio L/D [-]	
	Slope [°], speed range [min^{-1}] and drive [kWh]	
Dimensioning criteria:	Volume load	[t/(d m3)]
	Burning zone load	[t/(d m2)]
	Thermal burning zone load	[MW/m2]

Important mechanical features are:

- riding ring fixation
- roller station / alignment
- seals at inlet and outlet
- drive

With modern precalciner technology, outputs exceeding 10'000 t/d per kiln are possible with diameters still below the 6.5 m of the largest wet kilns.

There is a trend towards short L/D kilns with only two piers mainly because of lower investment.

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1. GENERAL

Today, all clinker producing installations of industrial size use a rotary kiln. The rotary kiln is still the only feasible way to manage this high temperature process with process material of varying behaviour.

One exception is the vertical shaft kiln still used in some parts of the world, e.g. China, however, for small unit capacities only. The other exceptions are few pilot installations based on sintering in a fluidized bed reactor.

Like many other great ideas, the rotary kiln was invented towards the end of the 19th century and has found application in many different industries. In 1987, Hurry and Seaman in the USA developed the first successfully working rotary kiln to produce cement clinker.

The first rotary cement kilns were using the wet process with one very long kiln tube, making it the dominating single piece of equipment of a plant. With technological progress, the kiln sections used for for drying, heating-up and calcining have gradually been replaced by other types of equipment, the rotary kiln remains to be the most suitable type of machine for the clinkerization process.

The rotary kiln has to satisfy three types of requirements:

Combustion:	as a combustion chamber for burning zone fuel	
Process:	as a reactor for the clinker burning process	(→ retention time)
	as a material conveyor	(→ slope, speed)
Mechanical:	stability of shape, carrying load, thermal flexibility, tightness	

Remarks:

- *Even though the rotary kiln is a relatively simple piece of equipment, nobody has developed a complete theoretical/mathematical model of its behaviour and process which would allow correct process simulation and equipment design.*
- *The rotary kiln is still the "heart" of the entire production line. Its OEE (overall equipment efficiency) depending mainly on hourly output and availability, is decisive for the success of a plant.*
- *The rotary kiln is designed to operate 24 hours a day, and the rest of the equipment upstream and downstream has to follow.*
- *Being a major cause for production cost (mechanical maintenance, refractories), a well managed kiln is vital for a successful plant.*

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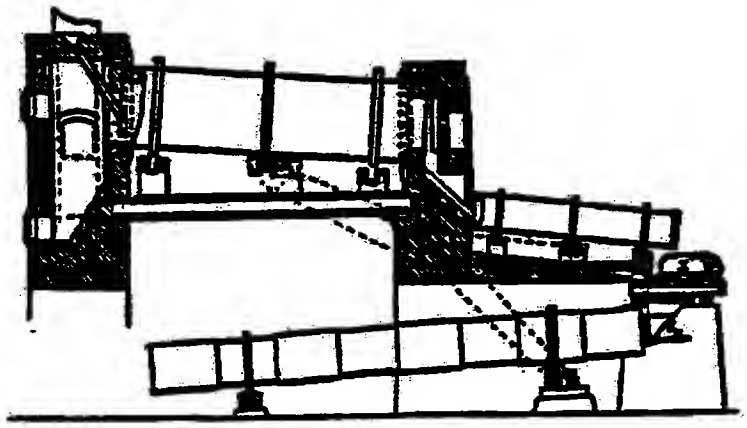
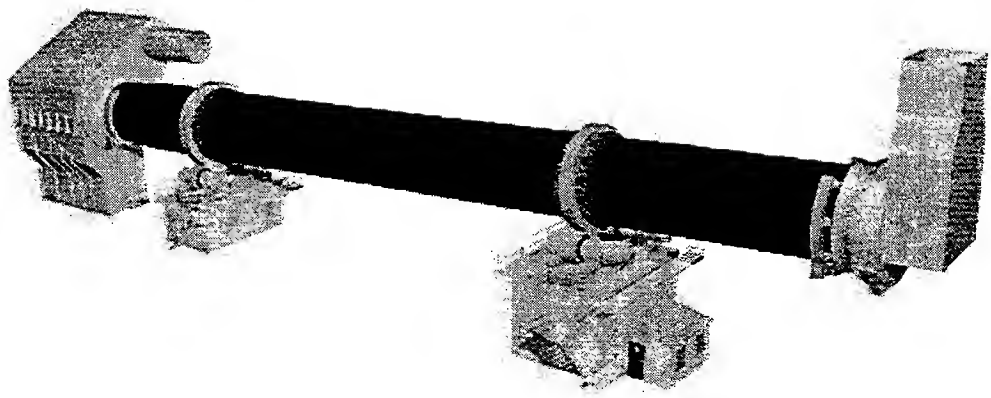
Figure 1: Old and new kiln





Latest kiln technology : short L/D kiln with 2 piecs and friction drive (FLS)

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2. KILN DIMENSIONING

The kiln dimensions are defined with diameter D (for kilns with different diameter: burning zone D) and length L:

L [m] and D [m] resp. L/D [m]

- For cement kilns, the actual L/D ratio range is:
  from **40** (for long wet kilns) to **11** (for modern short kilns with precalciner)
- The diameter D is the inner diameter D_i of the kiln (steel-) shell.
- Process technological dimensioning of a kiln is based on empirical figures and experience from existing installations

One limiting factor for the diameter is the mechanical stability of the 'arch' of the brick lining. Maximum diameters which can be safely realised with standard size bricks are about 6,5 m. The largest kiln in the "Holderbank" group is 232 m (wet process, 3750 t/d).

The following process technological dimensioning criteria are mostly used:

Specific Volume Load	$\frac{\text{Clinker Production}}{\text{Net Kiln Volume}}$	[t/(d m^3)] $< 5,5$
Specific Zone Load	$\frac{\text{Clinker Production}}{\text{Net Burning Zone Cross Section}}$	[t/(d m^2)] < 350
Thermal Burning Zone Load	$\frac{\text{Burning Zone Heat Input}}{\text{Net Bruning Zone Cross Section}}$	[MW/m^2)] $< 6,0$

Specific volume load and thermal burning zone (BZ) load have no physical significance. They are merely defined to make existing installations comparable.

The specific load is indirectly a gas velocity, because generating a certain amount of thermal energy by fuel combustion results in a proportional gas flow which can be calculated.

The thermal BZ load per cross section is considered the limiting factor for a modern kiln system. For a certain length/diameter ratio, which is typical for each kiln type, the thermal BZ load it is proportional to the heat load on the inside of the lining surface which is one of the main influencing factor on brick life. The limit usually respected is:

Max. Thermal BZ Load = 6 MW/m^2 (=5.16 x 10^6 kcal/m^2 h)

Other absolute limiting values of all the three factors are not known. Each supplier seems to have his own rules of kiln dimensioning. Since no theoretical formulas have been derived to calculate the kiln size on an analytical basis, it is possible, that the present limits of the dimensioning criteria may be surpassed even for the conventional processes.

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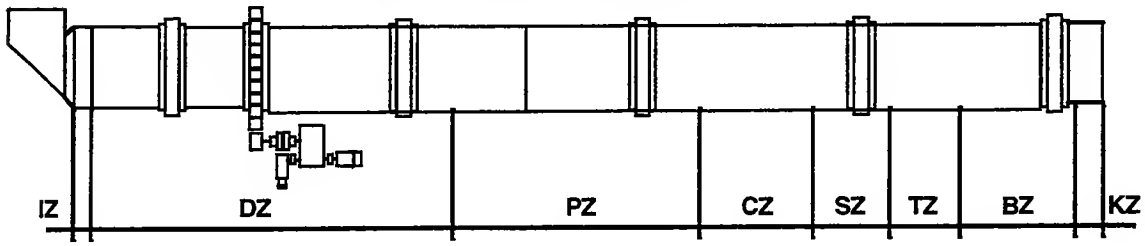
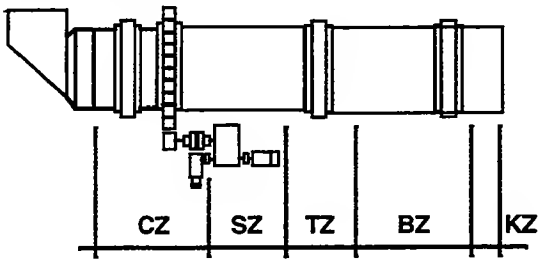
Figure 2: Long and short L/D kilns

Rotary kiln zones

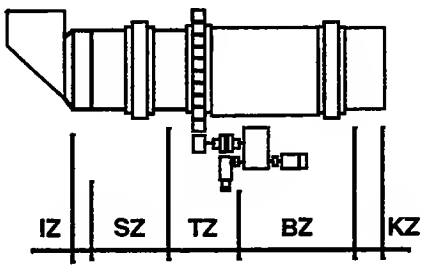
L/D = 30 (long wet kiln)



L/D = 15 (dry preheater kiln)



L/D = 11 (short dry PC kiln)



DZ = Drying zone
PZ = Preheater zone
CZ = Calcining zone
SZ = Safety zone
TZ = Transition zone
BZ = Burning (sintering) zone
KZ = Kiln internal cooling zone

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3. MECHANICAL ASPECTS OF ROTARY KILNS

The following aspects of kiln mechanical design are relevant for the process:

- Riding ring fixation, kiln shell ovality
- Kiln seals
- kiln drive
- refractory lining (separate paper)
- nose ring (covered in "refractory lining")

3.1 Riding Ring Fixation, Kiln Shell Ovality

A rotary kiln should be designed as cheaply as possible, yet it must still be rigid to guarantee minimum wear of the lining. This requirement can be met, if the deformation of the kiln shell is reduced to a tolerable limit.

The parameter expressing shell deformation at a certain point is the kiln shell ovality ω:

Definition of ω: $\omega=2\ (a - b)$ with 2a and 2b as the main axis of an ellipse

Investigations have shown, that generally a maximum relative ovality ω of 0,3% is allowed
This ovality may be subdivided into two amounts:

a) Ovality of the riding ring 3 cm due to external forces allowed value:

$$\frac{\omega}{dr} <= 0.2\%$$

a) Ovality of the kiln shell due to deformations by its own weight in loose riding rings and due to increased temperature.

The following two requirements must be met to keep the kiln ovality within the tolerable limits:

- The riding rings must be rigid enough
- The clearance between the ring shoes and the riding ring should be minimum during operation. The following table shows some practical values:

Riding Ring No.	1	2	3	4
Clearance during operation [mm] maximum [mm]	3-4	3-4	4-6 10-15	5-6

Riding rings with splined fixation provide much better support of the kiln shell. Because the kiln shell is laterally suspended in adequately designed carrying bars, ovality is much reduced resulting in noticeably better brick life.

Such systems are currently available from Polysius and FLS, the latter one is Iso offered as retrofit. Splined tire fixations are integral part of gearless kiln drive systems.

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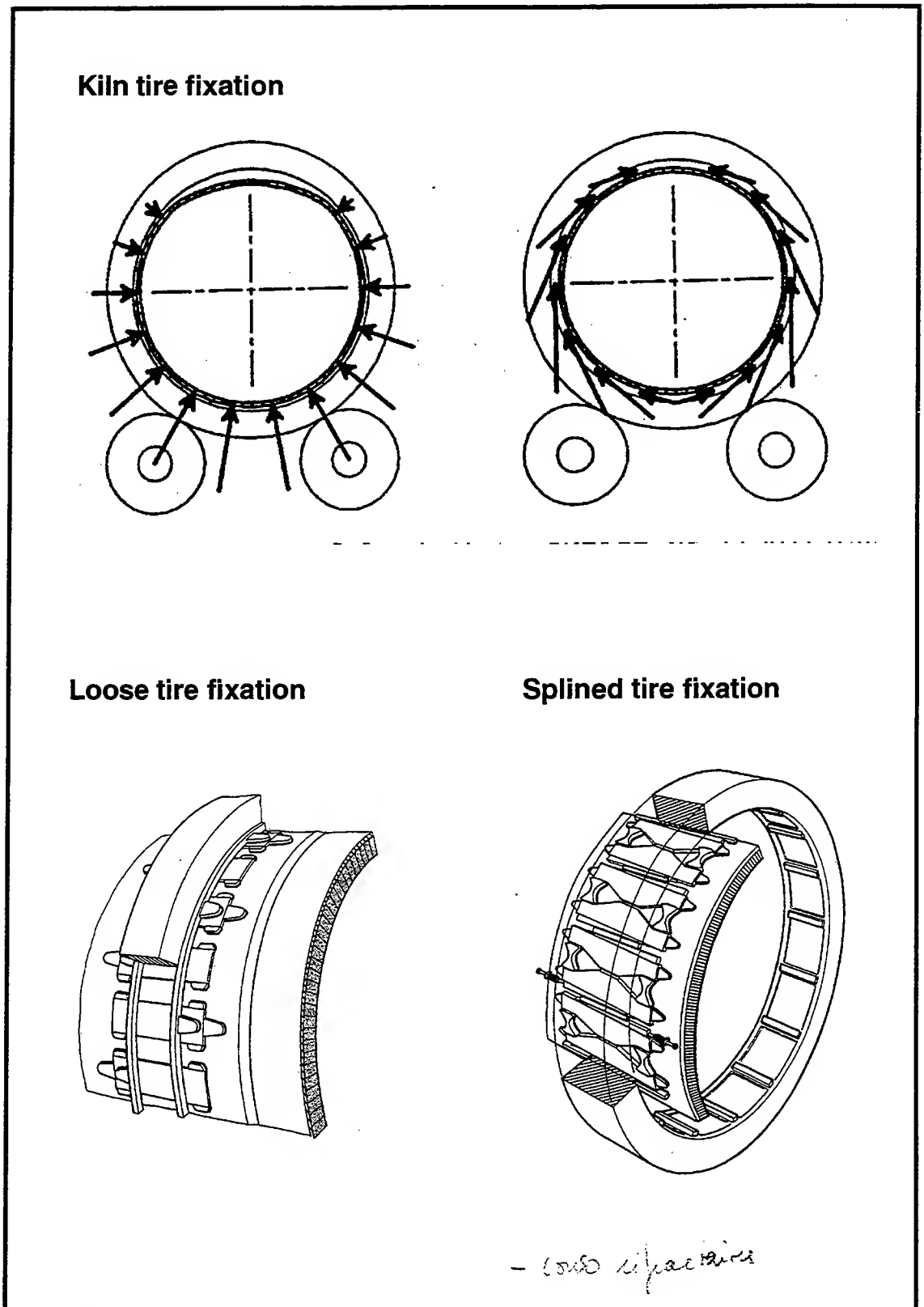
Figure 3: Tire fixations



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3.2 Kiln Seals

In order to avoid the danger of hot gases and dust leaking into the atmosphere, the entire kiln system is operated at negative pressure. The pressure profile starts at ambient (grate cooler: above first grate, satellite and tube cooler: fresh air inlet) and becomes increasingly negative towards the kiln induced draft (ID) fan.

Instead of leaking out from within the process, there is now a problem with ambient air being sucked into the system, called false air. Depending on the point of entry, false air has different undesired effects. That is why a lot of effort is made to keep process systems tight.

3.2.1 Kiln Inlet Seal

The kiln inlet seal (inlet: referring to material flow) is at point with negative pressure of less than 10 mmWG (modern 2-support kilns) up to 100 mmWG (long wet kilns with chains).

Modern kilns with low suction have high temperatures (up to 1300°C) instead.

False air entering the system causes

- Additional gas to be handled by kiln ID fan and dedusting system
- Unnecessary cooling of hot process gases reducing value of heat

Kiln inlet seals:

- Sealing force by pneumatic cylinders (pneumatic); sealing-rings
- Sealing force by coil springs/levers or weights (mechanical); sealing-segments
- Sealing force by leaf springs and rope with weight; lamella (fish scale)

Kiln inlet seals must be equipped with a dust return scoop ring to avoid spillage of kiln feed.

Note:

The inlet seal is designed to seal against cold fresh air from outside, but it can be damaged if it must seal hot gas from inside to ambient in case of system overpressure! (this happens sometimes during the heating-up phase)

3.2.2 Kiln Outlet Seal

With grate and tube coolers, the kiln outlet seal is installed between kiln head and rotary kiln where pressure should be slightly negative. Kiln outlet seals used with grate coolers must be designed to cope with pressure pulsation with occasional positive pressure. Outlet seal and nosering (brick retainer) with cooling air fan can be considered one system.

Here, the loss generated by false air reduces recuperation from the clinker cooler. Cold ambient air replaces hot secondary air from the cooler which has to be vented.

Outlet seals designed specifically for this application of the following type are available:

- Pneumatic
- Mechanical
- Lamella (fish scale)
- Labyrinth (outdated)

With planetary coolers, false air reduces the amount of cooling air resulting in higher clinker temperatures. The outlet seal is smaller, at lower temperature and negative pressure only.

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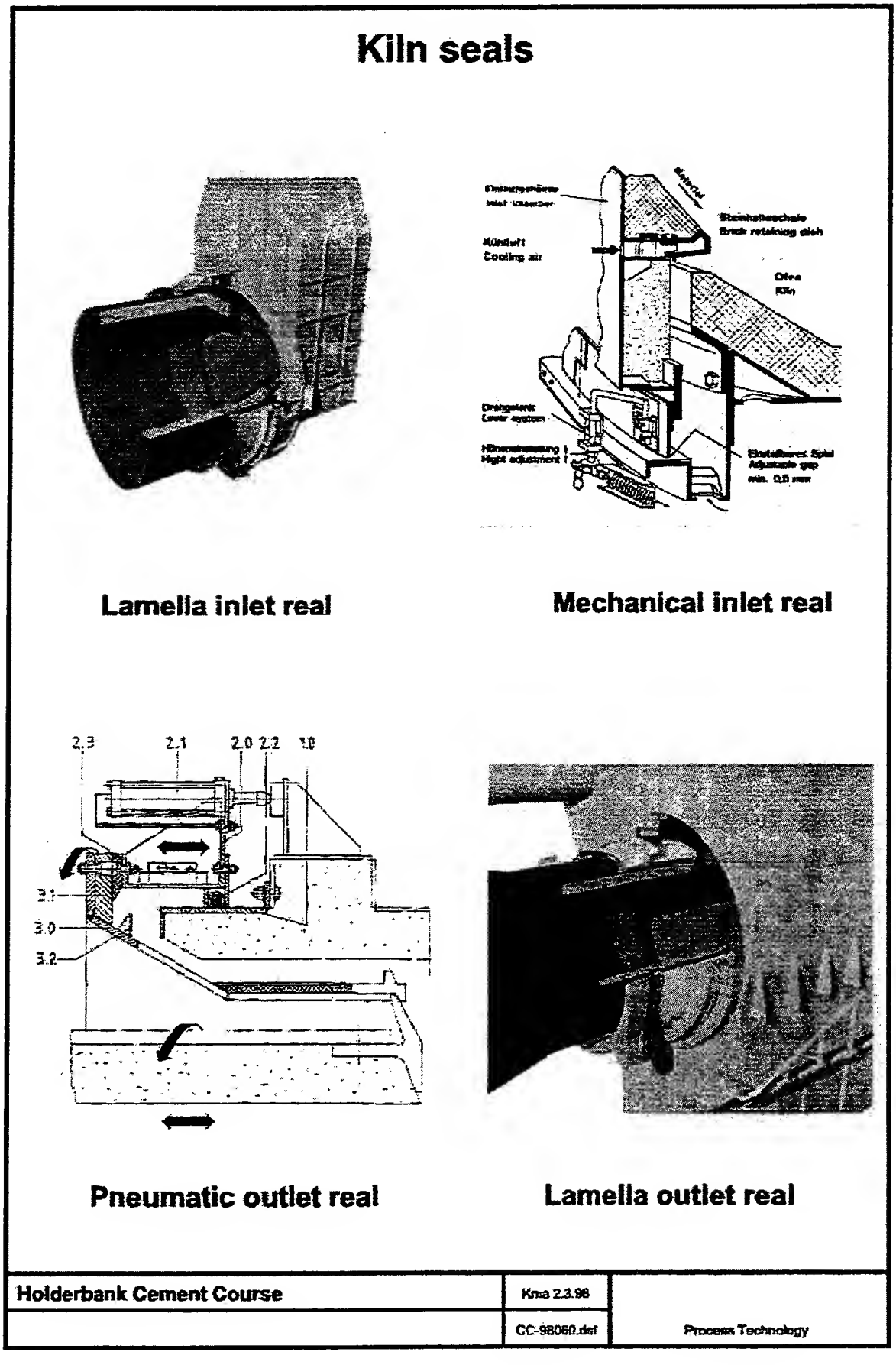
Figure 4: Kiln seals



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3.3 Kiln Drive

Kiln drives are designed for speeds between 1.0 and 4.0 min^{-1}, depending on slope, process and kiln dimensions. Long wet kilns are typically operated at the low end of this speed range where some new high performance kilns (short L/D with precalciners) are running at the upper end.

For over 10 years, rotary kilns have been driven by girth and pinion type drives. Decisive for their performance are:

- Correct dimensioning
- Correct alignment (even load distribution on the flanks of the teeth; no peaks)
- Adequate lubrication system and lubricant quality

With the new two support short kilns (L/D < 13) with long overhangs, kiln shell deformation and burning zone much closer to the drive, it became more difficult to ascertain correct alignment. Because of the determined load distribution on two piers, it became possible to avoid the girth drive by using the kiln rollers to transfer the torque to the riding ring: the gearless drive (=friction drive) was introduced. It is currently available from Polysius (POLRO) and FLS-Fuller (ROTAX).

The following elements are part of this system:

• Two supports	for defined load on the driven tire
• Splined tire fixation	for safe torque transmission to the shell
• Self-aligning roller station	for linear load pattern between roller and tire (friction)

Today, there are only few kilns with friction in operation; the first one was Lägerdorf 11 by Polysius. Detail optimization and long term experience are yet to be awaited.

Most systems have hydraulic drives for two rollers. This provides smooth operation, but is expensive, rather complex (hydraulic unit) and has higher power consumption. Electric direct drive of only one roller has been installed in one case.

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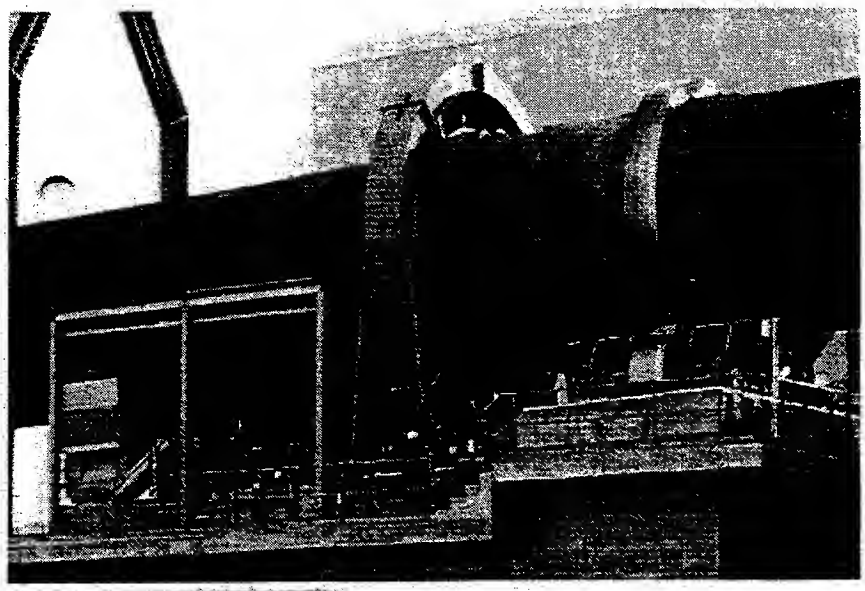
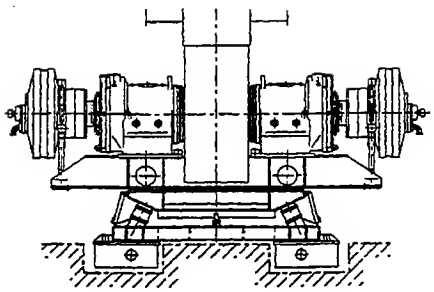
Figure 5: Kiln drives

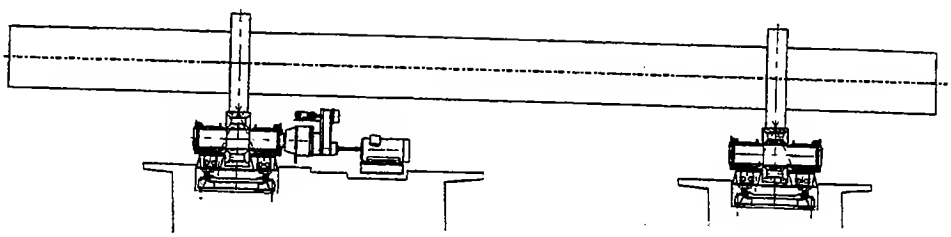
Kiln drives



Girth and pinion drive



Gearless (friction) drive



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Suspension Preheater

U. Gasser
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SUMMARY

Practically all modern kiln systems are equipped with a cyclone suspension preheater.

New installations include a precalciner with tertiary air duct, so that the preheater and precalciner have become one unit. However, the preheater has a specific task and is not principally connected to the precalciner.

Modern low pressure drop cyclones are the result of a development which started in 1932.

- Shaft Preheaters:
  - Counter-current heat exchange
  - Limited production (around 1000 t/d)
  - Disappointing heat exchange mainly due to poor meal distribution
- Hybrid preheaters:
  - Combination of shaft and cyclone stages
  - Bühler-Miag, Polysius, Prerov, Humboldt
  - Shaft stages often replaced by cyclone stages
- Cyclone preheaters:
  - Co-current heat exchange
  - Successful concept, predominantly used
- Industrial installations of cyclone preheaters:
  - Precalciner kiln with 4 to 6 cyclone stages (contemporary technology):
    * Large capacities possible > 10000 t/d
    * Up to 4000 t/d in 1 string
    * Heat consumption < 3000 kJ/kg possible (6 stages)
    * Sensitive to circulation phenomena (-> kiln gas bypass!)
  - 4-stage cyclone pre-heater kiln (standard technology 1970 to 1980):
    * Cyclone for raw meal preheating
    * Large application world wide
    * Capacities of up to 4500 t/d technically possible
    * Heat consumption: 3150 to 3350 kJ/kg cli
    * Sensitive to circulation phenomena (-> kiln gas bypass!)
  - 2-stage cyclone pre-heater kiln:
    * Less sensitive to circulating elements than 4-stage pre-heater
    * Higher heat consumption than pre-heater with more stages
- Most recent innovations:
  - Horizontal cyclone for "low profile" preheaters (Polysius)
  - Dip tube add-on RTS for 30% lower cyclone pressure drop

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1. GENERAL

1.1 History

With dry process, the heat exchange for heating up and calcination takes place between hot kiln gas and dry powder. Since the high dust losses from long dry kilns made it almost impossible to achieve acceptable heat consumption: other heat exchange principles had to be applied.

Since the temperature range to be covered is below 1000°C, where the meal behaves normally like dry powder, stationary reactors where the meal is in suspension with the hot gas can be used.

The first patent for a suspension preheater using four co-current cyclone stages was applied for in 1932 and issued in 1934 by the patent office in Prague to a Danish engineer employed by FLS. Even though the concept was entirely described in the patent, it took another 20 years for industrial application in 1951 by the company Humboldt, now KHD.

Other developments using shaft stages have been abandoned and today, a suspension preheater is actually a cyclone preheater.

1.2 Trend

All new kiln systems and the majority of the ones with start-up date after 1970 are equipped with cyclone pre-heaters. Gradually, older plants with wet kilns or long dry kilns are shut down for good due to their age as well as their high specific production cost

The portion of world's cement produced with kilns using suspension pre-heaters is still growing, as can be seen by the development of the "Holderbank" plants. It looks as if it will exceed 95% one day because no feasible alternative solution changing this development is in sight.

In combination with pre-calciners, units of 10'000 t/d have been built using up to four strings, five stages. Typically, 3500 t/d can be handled in one string, in a recent project even 4000 t/d have been proposed.

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Figure 1:

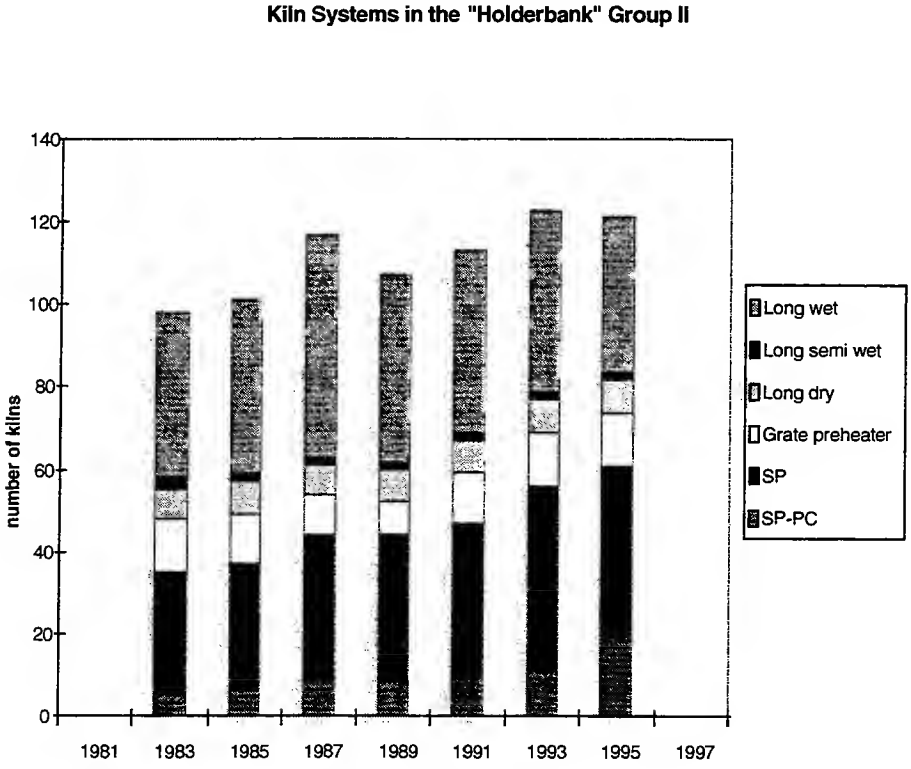

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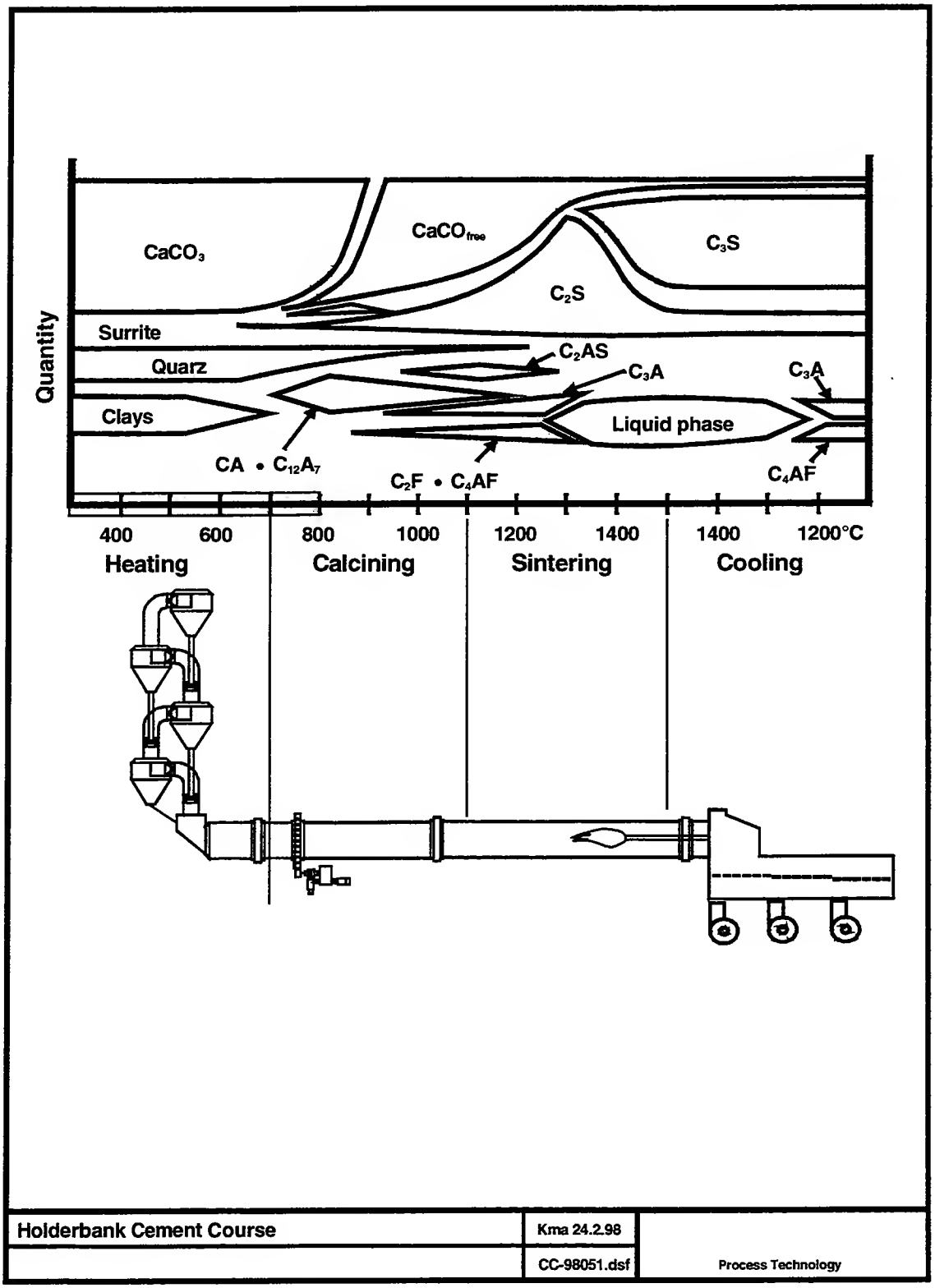
Figure 2:


Quantity
CaCO3
CaCOfree
C3S
C2S
Surrite
Quarz
C2AS
C3A
C3A
Clays
Liquid phase
CA • C12A7
C2F • C4AF
C4AF
400
600
800
1000
1200
1400
1400
1200°C
Heating
Calcining
Sintering
Cooling
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2. HEAT EXCHANGE IN A SUSPENSION PREHEATER

2.1 Counter-Current Heat Exchange (Shaft Stage)

The most efficient type of heat exchange is the counter-current principle. The flows of the heat releasing media and the heat absorbing media are in opposite directions. This provides optimum the temperature difference (=temperature gradient, in theory allowing almost complete heat exchange.

In case of a suspension preheater, where powder is suspended in a gas, the heat exchange takes place in a "reactor" vessel where the hot gas enters from below and leaves at the top. The meal to be preheated is fed at the top. The meal retention time depends on distribution across the gas flow and the retention time, which is determined by the gas velocity.

In industrial installations, the heat exchange proved to be far below expected, because even distribution of the meal was not achieved, particularly not with large units.

2.2 Co-Current Heat Exchange (Cyclone Stage)

Co-current heat exchange takes place if both heat exchanging media flow in the same direction. Because of the rapidly decreasing temperature difference, the meal can never reach gas inlet temperature.

Good and reproducible results in industrial installations with this type lead to the predominance of this principle in the cement industry. The heat exchanger is a gas duct with velocities from 10 to 20 m/s, equipped with good meal dispersion devices. The purpose of the cyclone is primarily to separate meal from gas, and not to exchange heat!

2.3 Thermodynamic Limits

Regardless of the type of heat exchange, there is always a thermodynamic imbalance between hot gases from kiln and calciner and cold raw meal. The heat contained available in the hot gas leaving the rotary kiln exceeds the heat required for heating the meal to the temperature levels required for calcination.

Another limit must be observed: Because the temperature gradient between gas and meal (T gas > T meal) must always be maintained, a higher calcination degree than 30% cannot be achieved without additional heat input.

The following heat balance estimate shall illustrate this:

Heat contained in the gas:	*2300 kJ/kg cli*	
(1100°C; 1.3 Nm3/kg cli)		
Heat to preheat meal to 850°C:	*1300 kJ/kg cli*	
(1.6 kg meal /kg cli)		
Heat required for 30% calcination:	*650 kJ/kg cli*	
Rest (ideal heat exchange):	*350 kJ/kg*	*(corresponding to 200 °C)*

This shows that even if the heat of the gas above 850°C is used for partial calcination (about 30%), there is still excessive heat in the gas which would correspond to 200°C gas temperature.

It is obvious that even with a very large number of stages (with accordingly small temperature gradients), there will always be excess heat! This waste heat is lost only for the kiln system, but not for the plant, since it can be used for raw material drying in the mill.

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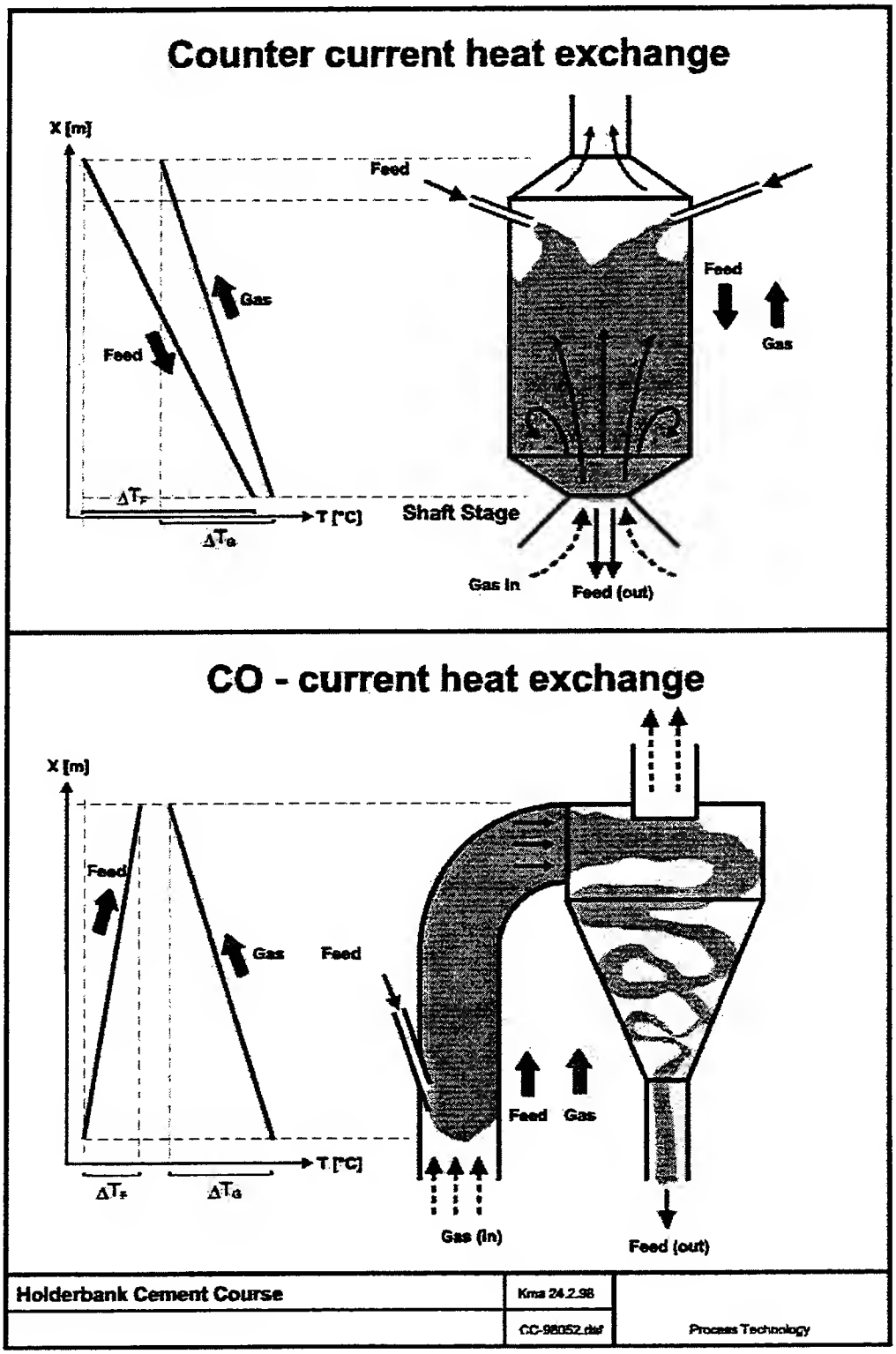
Figure 3:


Counter current heat exchange
X [m]
Feed
Gas
Feed
ΔTp
ΔTg
T [°C]
Shaft Stage
Feed
Gas
Gas in
Feed (out)
CO - current heat exchange
X [m]
Feed
Gas
Feed
ΔTp
ΔTg
T [°C]
Feed
Gas
Gas (in)
Feed (out)
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3. PREHEATER TYPES

3.1 Preheaters with Shaft Stages

The rather disappointing performance of the shaft stage made it virtually disappear from the market. Many hybrid preheaters were equipped with one or two cyclone stages replacing the shaft stage.

Shaft stages at the kiln inlet have the advantage to be less sensitive to build-ups. This could be an advantage in cases where elevated sulfur input in the kiln system must be expected.

Several Suppliers built preheaters using shaft stages. Two groups can be distinguished:

3.1.1 Pure shaft preheaters:

Polysius:
- GEPOL
- Self-supporting structure (no tower required)
- Vertical tube with restrictions
- For small capacities (up to ca. 1000 t/d)

ZAB Dessau:
- Some applications in Eastern Europe
- Similar to GEPOL, but not self-supporting
- The Deuna plant had originally 4 ZAB shaft preheaters

3.1.2 Hybrid preheaters:

Several suppliers used a combination of shaft and cyclone stages:

Polysius:
- DOPOL preheater (first generation)
- The central swirl-pot (second lowest stage) was shaft stage
- Replaced by DOPOL 90 from 1990
- Gradually developed into a cyclone preheater
- Up to ca. 3000 t/d

Bühler-Miag:
- Lowest stage was shaft stage
- Later often replaced by cyclone stage

Prerov:
- One large shaft stage with dedusting cyclone
- Shaft stage selfsupporting
- Additional cyclone stage possible
- Separate meal duct to kiln
- As sensitive to circulation phenomena as a cyclone type

MBM:
- Bottom shaft stage with 4 cyclone stages
- Only hybrid design still on the market

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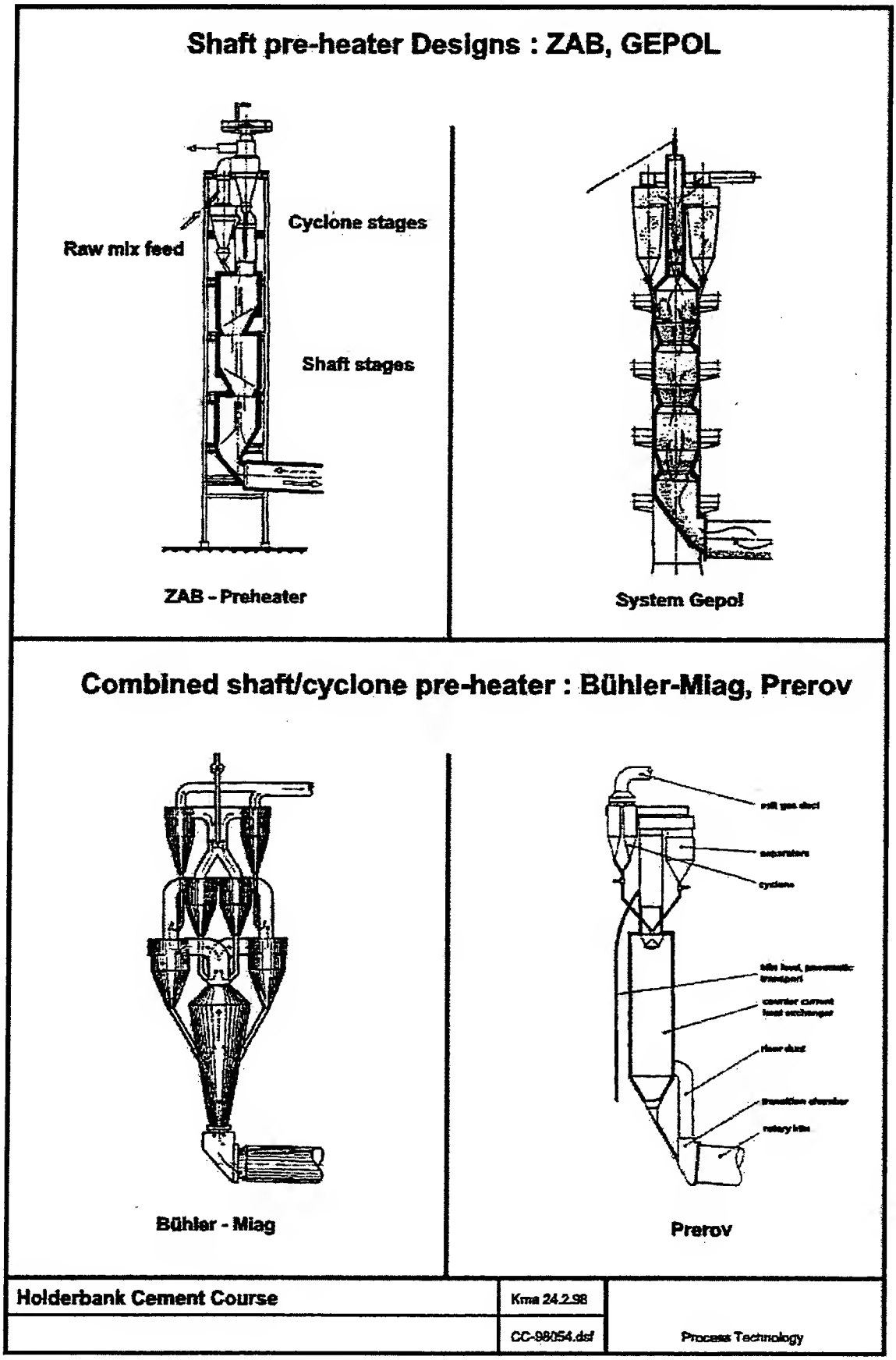
Figure 4:


Shaft pre-heater Designs : ZAB, GEPOL
Raw mix feed
Cyclone stages
Shaft stages
ZAB - Preheater
System Gepol
Combined shaft/cyclone pre-heater : Bühler-Miag, Prerov
Bühler - Miag
Prerov
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3.2 Preheaters with Cyclone Stages

A quasi counter-current heat exchange can be achieved by serial installation of several co-current stages. The result is the multi-stage cyclone preheater as it is generally applied in modern cement plants.

In the early years, one and two stage systems have been installed with long kilns, often to avoid problems caused by circulating phenomena. A large number of plants are equipped with four stages; the majority of them were built before 1990.

Today, five stage preheaters represent the economical optimum. High raw material moisture leads occasionally to fewer stages, in combination with low temperature dedusting systems, or in areas with high fuel cost, six stages can be more economical. Number of stages depends thus on:

- Raw material moisture (i.e. drying heat requirement)
- Cost of thermal energy
- Cost of electrical energy
- Gas handling system (temperature limit, dew point)
- Soil conditions (foundations, earthquake zone -> height of structure)

If raw material moisture shows significant seasonal variations, it can be economical to equip preheaters with "variable stages". This is achieved by feeding all or part of the meal to the second highest stage or by skipping a stage.

Note: *Numbering of stages is always from top to bottom:* ***top stage = stage 1.***
Exception: Polysius: bottom stage = stage 1

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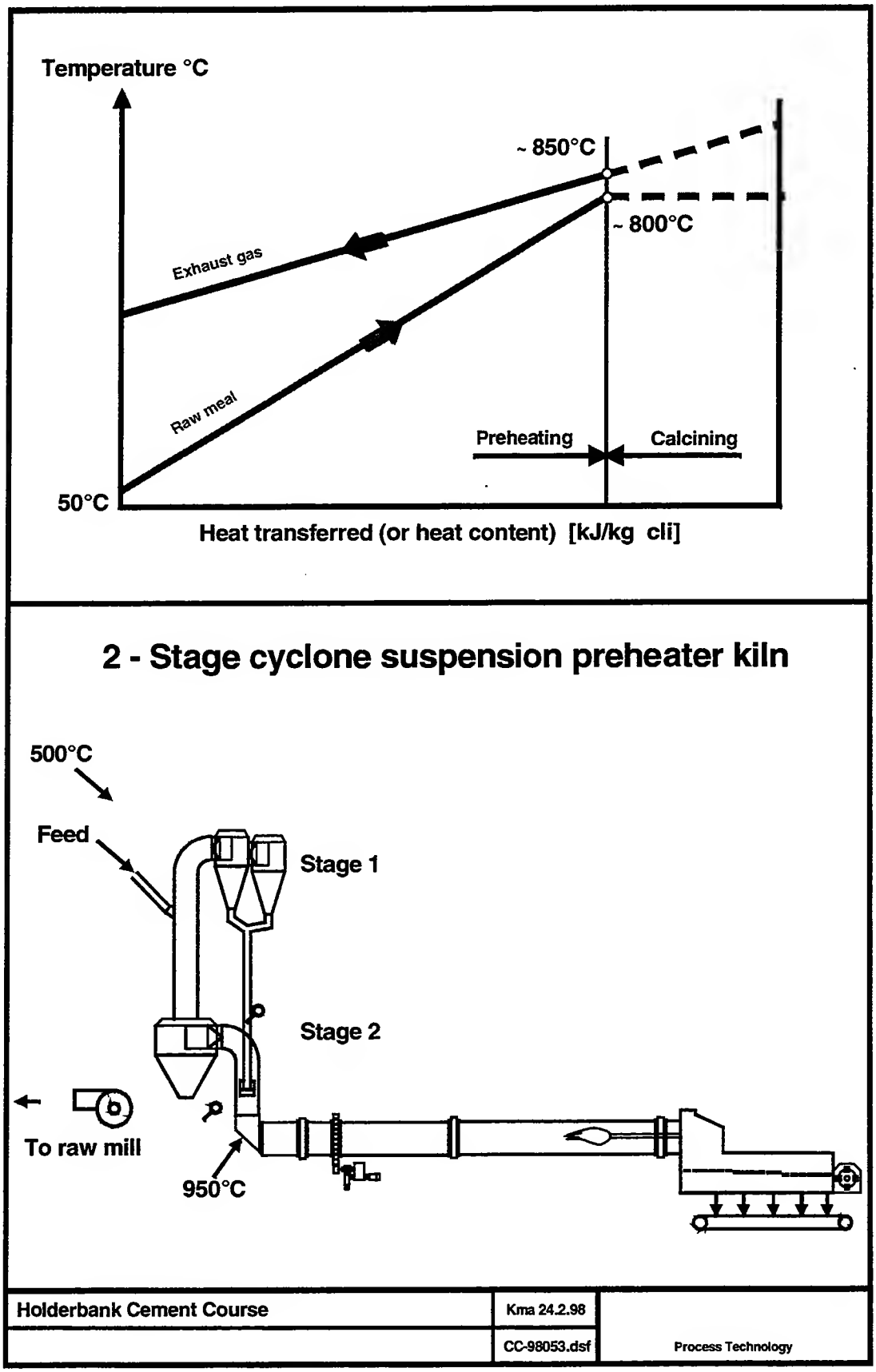
Figure 5:


Temperature °C
~ 850°C
~ 800°C
Exhaust gas
Raw meal
Preheating
Calcining
50°C
Heat transferred (or heat content) [kJ/kg cli]
2 - Stage cyclone suspension preheater kiln
500°C
Feed
Stage 1
Stage 2
To raw mill
950°C
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3.3 Economical Number of Stages for Cyclone Preheaters

For many years, the pressure drop across one preheater stage was up to 1,5 kPa (15 mbar). The reason for the 4-stage pre-heater being so widely used is, that it represented an optimum between investment cost (structure height, foundation), pressure drop and heat consumption. . The performance of comparable systems built in about the same period are within a relatively narrow range.

About two third of the pressure drop of a stage occurs in the cyclone and depends on its shape/design and the size, the latter being the determining cost factor.

New cyclone designs are now on the market with only 0,5 to 1,0 kPa (5-10 mbar) pressure drop per stage. Considering increasing energy cost, it is justified to install 5 or 6 pre-heater stages for new or modified kiln systems.

The following table indicated the estimated effect of a 5th and a 6th cyclone stage:

		4 to 5st	5 to 6st
Heat consumption	kJ/kg cli	- 80	- 50
Exhaust gas temperature	°C	- 40 to -50	- 20 to -30
Exhaust gas quantity	Nm^3/kg cli	- 0,03	- 0,015
Drying capacity in RM	% H_2O	from 8 - 6,5	From 6,5 - 5,5

3.4 Minimum Gas Velocity

Dimensioning of a cyclone preheater is a careful consideration of the importance of separation efficiency, pressure drop, part load operation capability, size of the preheater and cost of the project.

It must be mentioned that there is a lowest gas velocity in a cyclone preheater. If operation results in lower figures, the meal will not be lifted by the gas anymore, resulting in poor heat exchange and consequently high heat consumption, but also excessive temperatures .

Large dimensions give lower velocities with low pressure drop, but also limit the lowest possible economical production.

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Figure 6: Polysius

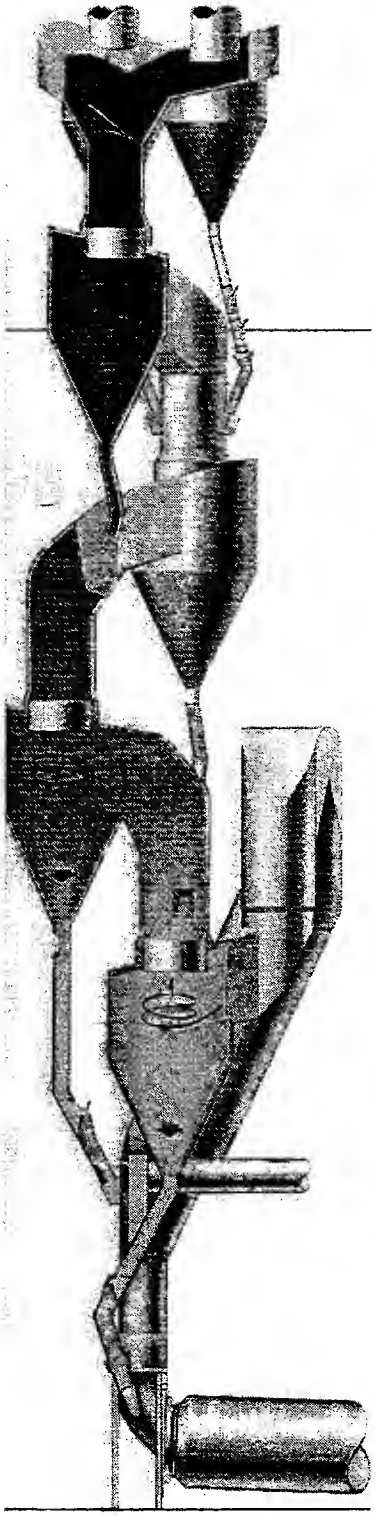

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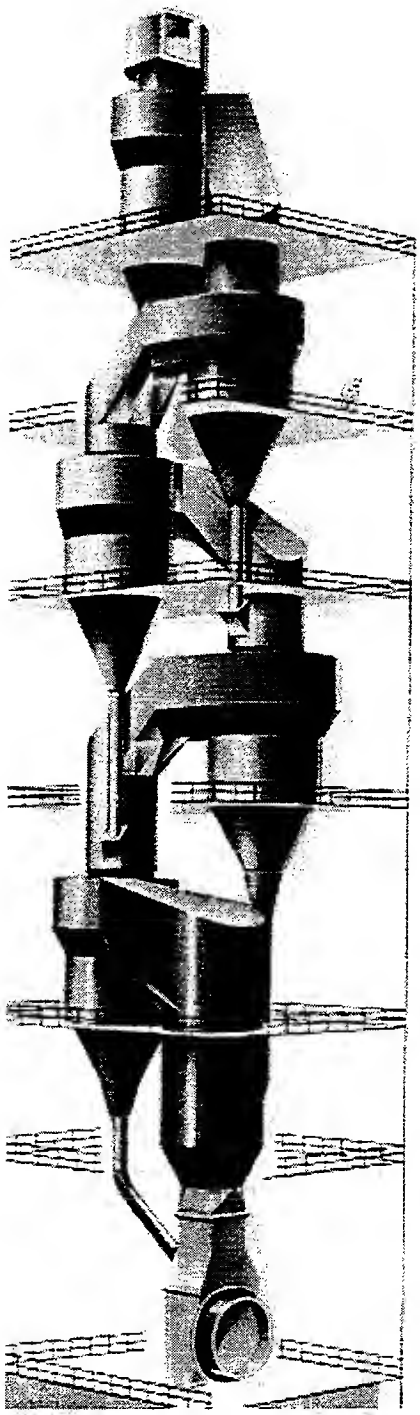
Figure 7: FLS



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4. DESIGN FEATURES OF PREHEATER-CYCLONES

4.1 General

Modern preheaters are designed for low pressure drop using the new cyclone design which must still provide good separation efficiency, particularly in the top and the bottom stage. Cyclone inlet velocities are designed in the range of 10 to 15 m/s.

It has been found that the total pressure drop of one cyclone stage is caused by about 1/3 by the gas duct (i.e. lifting of the meal) and 2/3 by the cyclone. Since not much can be done regarding lifting of the meal, efforts have been made to improve the cyclone design in order to reduce total pressure drop: the low pressure (drop) cyclone was designed.

Cyclone design means to optimize between high separation efficiency, low pressure drop and low cost (i.e. small size).

Other than having the correct design parameters, all stages should be equipped with

- Dip Tubes (also called 'immersion tubes', 'thimbles' or 'vortex finders')
- Meal flaps
- Splash boxes (or splash plates).

4.2 Dust Cycles

The entire kiln system is subject to dust cycles. Precondition is gas flow in opposite direction of pulverized process materials. This causes wear, unnecessary material transport and heat losses due to heat exchange in the wrong direction.

In the preheater, internal dust cycles due to poor separation efficiency of the cyclones result in less than optimum preheating of meal. Unfortunately, it is almost impossible to measure dust loss from lower cyclones in normal operation. The only indicator is the temperature profiles of gas and meal, but even the meal temperature is not always easy tp measure.

4.3 Features

4.3.1 Splash Box

Early cyclone preheater designs had no splash boxes. Instead, the meal was fed into the gas at a higher point against the gas flow, creating some turbulence with a certain distribution effect.

Modern cyclone preheaters must be equipped with correctly designed splash boxes for optimum meal distribution across the gas duct cross section. The principle is based on impact on a plate. In some installations, the bottom plate of the splash box can be adjusted.

Note: *No splash box must be installed at the kiln inlet! The hot meal from the bottom cyclone must enter the rotary kiln as smoothly as possible. Meal is easily picked up by the kiln gas and will create a dusty transition chamber.*

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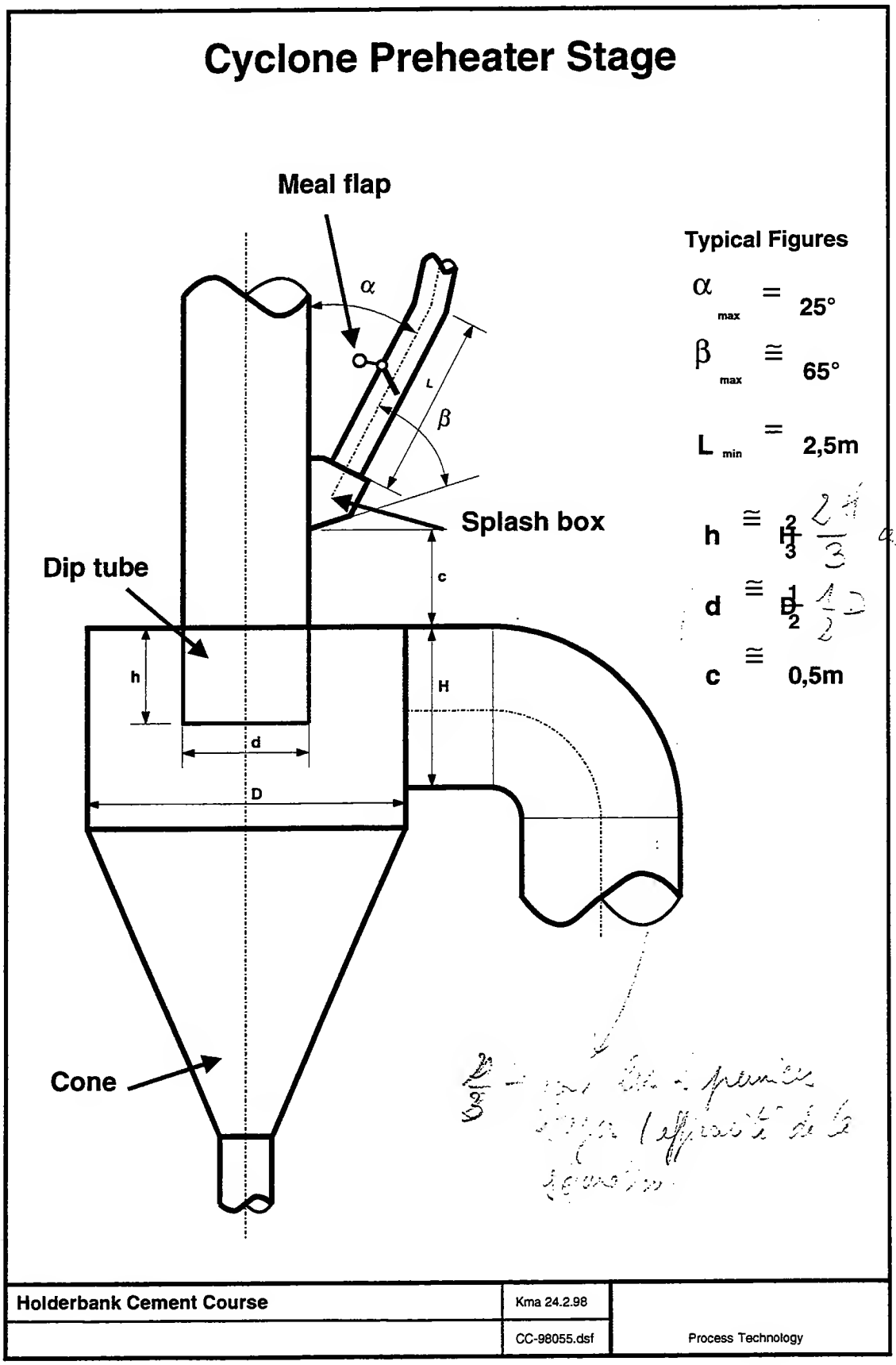
Figure 8:



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4.3.2 Dip Tube (Immersion Tube, Vortex Finder, Thimble)

This integral element of the cyclone has a decisive influence on separation and pressure drop. It makes the gas to follow a 180 to 360° rotation thus creating the desired centrifugal force for the separation effect.

In the colder upper stages (stage 1 to 3) it can be designed as simple extension of the outlet gas duct, made from steel plate. These upper stage dip tubes create usually no problems except when the preheater gets overheated, e.g. during start-up. Then, the dip tube can collapse, causing excessive pressure drop.

In the hotter lower stages, mild steel ducts from one piece are not suitable. Several segmented designs made from heat resistant steel or ceramic material (Hasle) are available on the market. It is standard today that all stages are equipped with dip tubes.

Note: It appears that some designs of segmented dip tubes have a tendency to unhook enabling individual elements to drop and to block the cyclone outlet!

For older plants, installing a segmented dip tube in the lower stages is a optimization possibility which is often applied.

4.3.3 Meal Flap

In order to understand the purpose of the meal flap, the following two aspects must be mentioned:

- There is a pressure difference across a cyclone stage, i.e. between two subsequent cyclone gas outlets (maintained by the ID fan).
- Without meal, there are two ways the gas can flow from one stage to the next: through gas duct and through meal duct

If there was an ideal kiln system, i.e. a system with 100% constant meal flow and never changing operation parameters, the meal duct diameter could be designed for just the meal. The meal would then fill the entire cross section, leaving no opening for the gas. In reality, there are fluctuations of meal and dropping build-ups, requiring oversized meal ducts.

It is the purpose of the meal flap to close the free cross section not used by the meal, to avoid gas bypass. There are designs that open only when a certain weight pushes them open, thus creating meal fluctuations. Other designs (see figure) are adjustable so that they move only in case of meal peaks or lumps.

Not operational meal flaps cause heat loss and allow build-up formation in meal ducts (circulating elements)!

4.3.4 Cyclone Shapes

The separation efficiency of a cyclone gets better with longer dip tube and increasing distance between swirl (cylinder) and dust collecting cone, i.e. with high and slim shapes.

The top stage of preheaters is designed for high separation efficiency. In order to save height, most suppliers install twin cyclones with the drawback that meal and gas have to be split. There are a few plants from FLS with only one top cyclone, avoiding this drawback.

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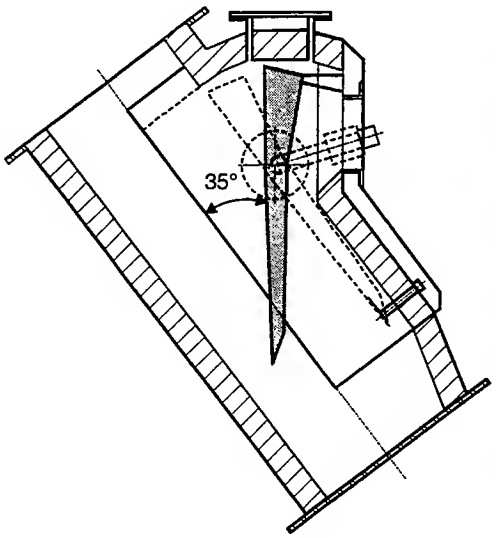
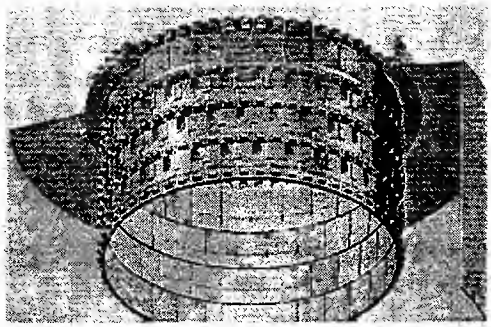
Figure 9:

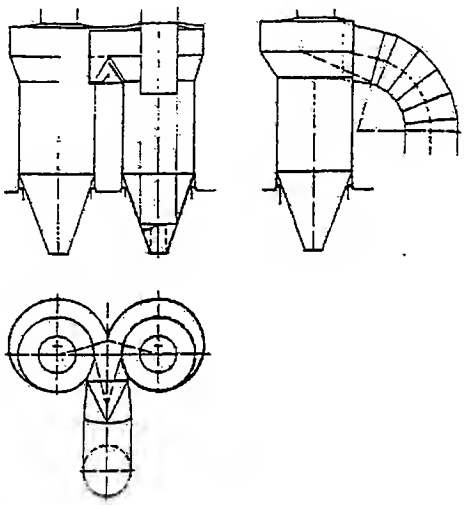


Meal flap

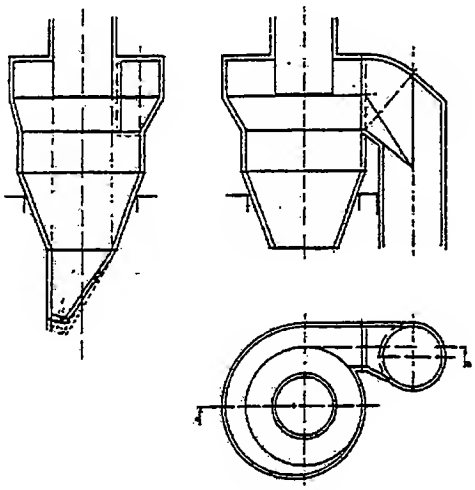


FLS

Dip tube
segmented for hot stages (4 to 6)



Top stage (double cyclone)



Cyclone for lower stage

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5. PREHEATER OPERATION

The performance of a preheater is assessed based on the criteria:

- Temperature profile (first indicator: exit gas temperature)
- Pressure profile
- Oxygen profile

Table Typical Gas Temperature Profiles

		4 stages		5 stages		6 stages	
		SP	PC	SP	PC	SP	PC
Stage 1	°C	350	360	300	310	270	280
Stage 2	°C	540	570	490	500	440	460
Stage 3	°C	710	740	630	650	580	600
Stage 4	°C	840	870	750	770	690	710
Stage 5	°C	-	-	840	870	770	800
Stage 6	°C	-	-	-	-	840	870

5.1 Operating Problems of Suspension Preheaters

Some reasons for poor preheater performance frequently experienced:

- Worn out or non existing immersion tubes (often in bottom stage)
- Open inspection doors, leaky gaskets or holes in the pre-heater (cold false air leaks in, can be detected by hissing sound)
- Blocked or non existing meal flaps
- No splash boxes (specially older preheaters), combined with not optimum position of meal feed point (e.g. old DOPOL)
- Excessive dust circulation due to poor separation efficiency of cyclones

5.1.1 Circulation Phenomena.

Cyclone preheaters are sensitive to circulation phenomena. Cyclone blockages cause kiln stops resulting in production loss and dangerous cleaning actions. Possible causes are:

- Excessive input via feed or fuel (Cl, S, 1 Na, K)
- Chemical unbalance (sulphur, alkali ratio)
- Unfavourable kiln/burner operation
- Unfavourable design geometry of bottom stage and kiln gas riser duct area

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Countermeasures known today allow to solve the problems are:

- Change feed composition or fuel quality
- Improve burning conditions
- Install automatic cleaning (air cannon, big blasters) at critical locations
- Change temperature profile by installing a small secondary burner
- Install a kiln gas bypass* system

**A bypass system is not desirable since it is expensive and causes loss of heat and material. It is therefore the last solution left and should be only considered if all other measures are not sufficient.*

The paper 'circulating phenomena' contains more details on this rather complex subject.

6. NEW DEVELOPMENTS

6.1 Horizontal Cyclone

Polysius have developed a "horizontal cyclone". (not to be mixed up with earlier designs of Kawasaki!)

This cyclone is a modified version of the conventional cyclone with the major difference that the gas outlet is also at the bottom, encircling the meal outlet.

The heat exchanger duct is still from bottom to top, but the stages can be arranged next to each other instead on top of each other. This allows a significantly lower height of the preheater structure.

It is expected that savings in civil cost can be achieved. Additional benefit is possible in cases where the maximum height is restricted (earthquake zones, scenery protection).

Only top cyclones on conventional preheaters are in industrial operation, however. Any other experience is yet to be made.

6.2 TRS

The Austrian company Zyklontechnik have introduced a dip tube add-on device which will reduce pressure drop across the cyclone (not the entire stage!) by 30% at otherwise unchanged performance.

The principle is to avoid the flow around the edge of the dip tube. Instead, the horizontal swirl from the gas inlet is maintained and can continue into the dip tube through an accurately shaped slot in the TRS.

Prerequisite is aerodynamically correct cyclone design and very accurate manufacturing of the TRS, which cannot be made locally.

The device can be mounted to the bottom of a shortened dip tube. If the inspection opening is large enough, the whole unit can be installed in one piece, otherwise it comes in pieces.

Several TRS are in operation (not in preheaters, however) with performance equal to or exceeding the predicted figures.

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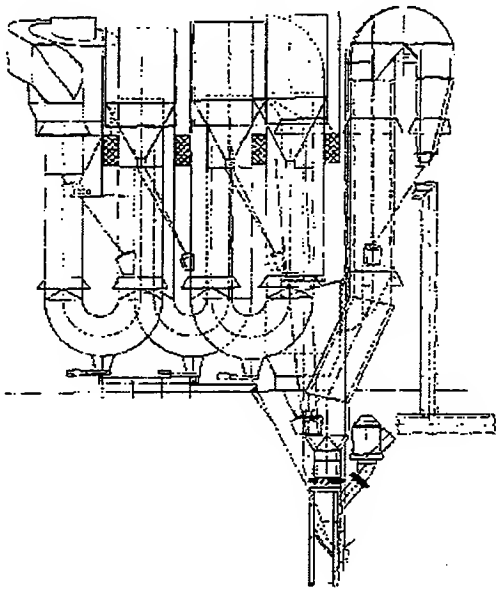
Figure 10:

Suspension preheater
with horizontal cyclones (Polysius)



Dip tube extension TRS
(Zyklontechnik)

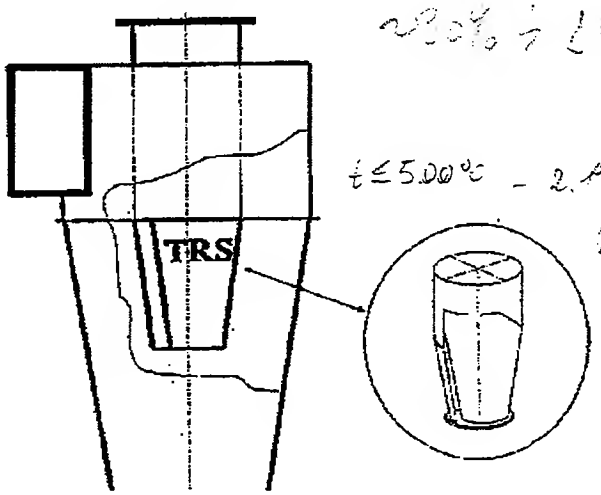


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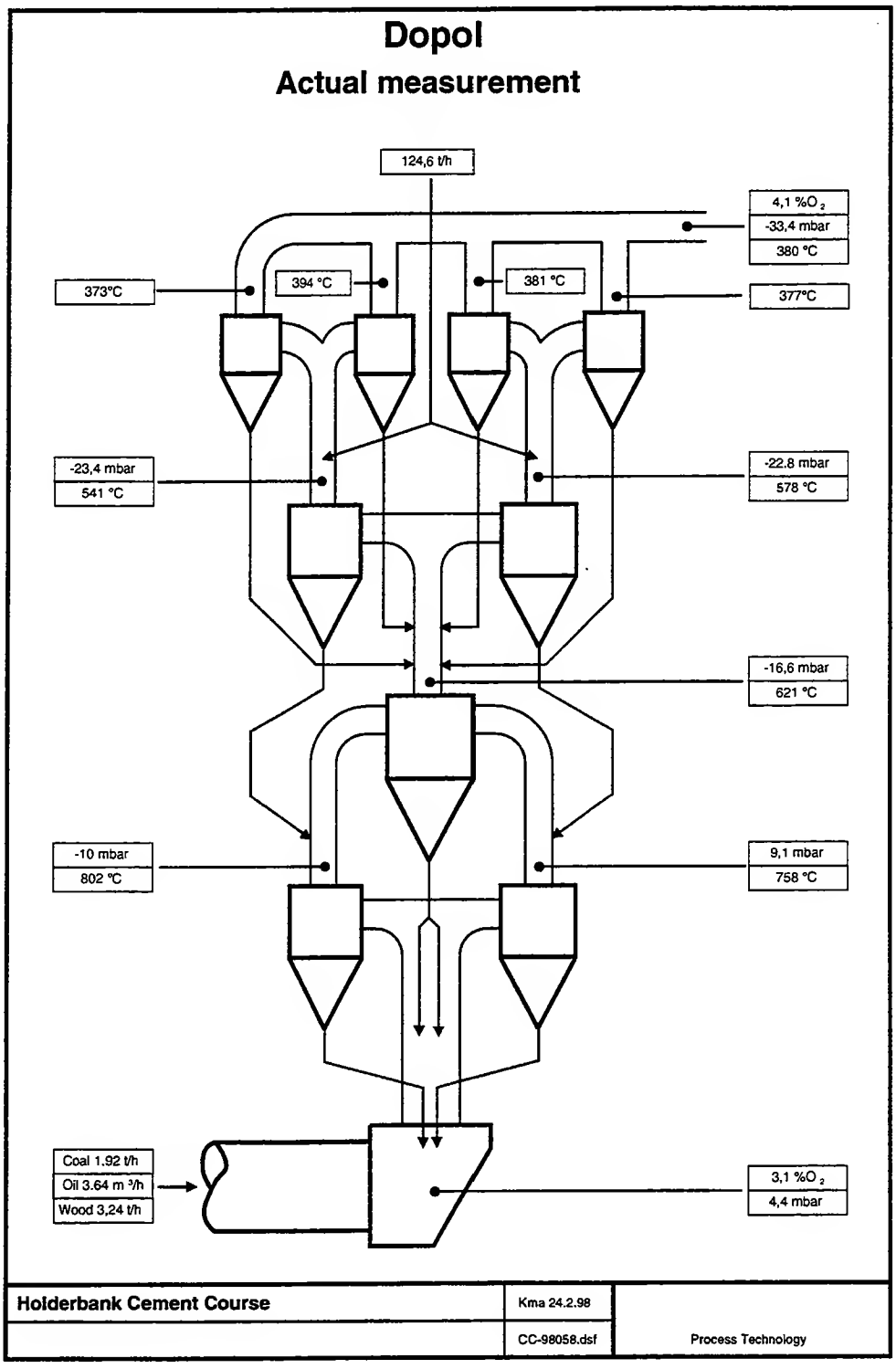
Figure 11:



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Precalcining Systems

U. Gasser
VA 93/4055/E

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SUMMARY

When burning cement clinker in a suspension preheater kiln, about 2/3 of the total heat consumed or about 2000 kJ/kg are required for the dissociation of $CaCO_3$ also known as calcination.

The idea of precalcination is, to let this process take place before the meal enters the rotary kiln by introducing that part of the fuel, i.e. up to 65%, into a stationary reactor.

Because the combustion air (tertiary air) is drawn through a separate duct parallel to the kiln directly from the cooler, the rotary kiln operates at significantly lower specific thermal load and gas flow.

The main advantages of precalcination are:

- More stable kiln operation due to better kiln control via two separate fuel feed/control points
- More stable kiln operation due to controlled meal conditions at kiln inlet
- Reduced thermal load of burning zone
- Higher kiln availability
- Longer life of burning zone refractories
- Larger capacity with given kiln dimensions, resp. smaller kiln for given capacity
- Possibility of increasing capacity of existing kilns
- More favorable conditions regarding circulating element problems
- Allows shorter kilns (L/D <12, 2 supports)
- Lower NOx emissions

The drawbacks of higher gas exit temperature after the bottom cyclone and the preheater higher pressure drop can be compensated by five preheater stages and modern low pressure drop cyclones.

There are three basic precalciner arrangements available from several suppliers: in-line, off-line and separate line, all with separate tertiary air duct.

Being the key for complete combustion, the main design criteria is gas retention time: 2 to 3.5 sec minimum, depending on fuel reactivity, 0.5 to 1 sec more for in-line calciners.

Systems where 10 to 20% of the fuel is introduced to the riser duct are considered secondary firings (SF) and not precalciners.

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1. INTRODUCTION

The idea of separating the calcining process from the burning process was already described in a patent as early as 1912.

However, the first industrial precalciner was built by Humboldt-Wedag (KHD) only in 1966 (Fig. 1). It was the Polysius kiln in Dotternhausen (Germany) which was equipped with a special 5-stage suspension preheater with extended riser duct. This riser duct had a larger diameter and the shape of a gooseneck to provide more length thus more gas retention time enabling combustion of oilshale, a locally available fuel containing raw material. The combustion air (tertiary air) was still drawn through the rotary kiln. Additional burners were installed later at the bottom of the precalciner chamber.

Tube type calciners using the gooseneck design are still being used by KHD (Pyroclon) and Polysius (Prepol).

So it is obvious that the precalciner (PC) kiln was developed from a straight suspension preheater (SP) kiln. The process characteristics (heat balance etc.) of both SP and PC kiln systems are quite similar, the main difference being the fact that in case of the PC kiln, 50 to 60% of the fuel (heat) is introduced via a chamber between kiln inlet and bottom cyclone. This allows to match the process heat requirements more evenly leading to significant improvements.

Since true precalciners with 50 to 60% PC fuel ratio require a separate tertiary air duct, almost all PC kilns feature a grate cooler.

The demand for larger and larger capacities which started back in the 1970ies led to a rapid development of the new precalciner technology. The fastest growing market asking for the largest units was in Japan where most of the clinker is produced in PC kilns.

During that period, 12 competing suppliers developed their own precalciners, 8 of them were Japanese (see para „synopsis of precalciners").

After the home market for cement plants started to stagnate, the Japanese suppliers exported their know-how via licenses as well as entire plants. During the late 1980ies, where only few new plants have been constructed world-wide, the Japanese activities came to a stop.

The latest development of precalciner technology was aimed at

- Complete combustion, also for low reactivity fuels
- Suitability for a wide range of fuels
- Low emissions of NOx

Since the Japanese competitors have virtually disappeared on the international market, the variety of precalciner systems is reduced. Five European suppliers (FCB, FLS-Fuller, KHD, Polysius and Prerov) offer precalciners, some even a choice of alternative solutions.

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2. THEORETICAL ASPECTS OF PRECALCINING

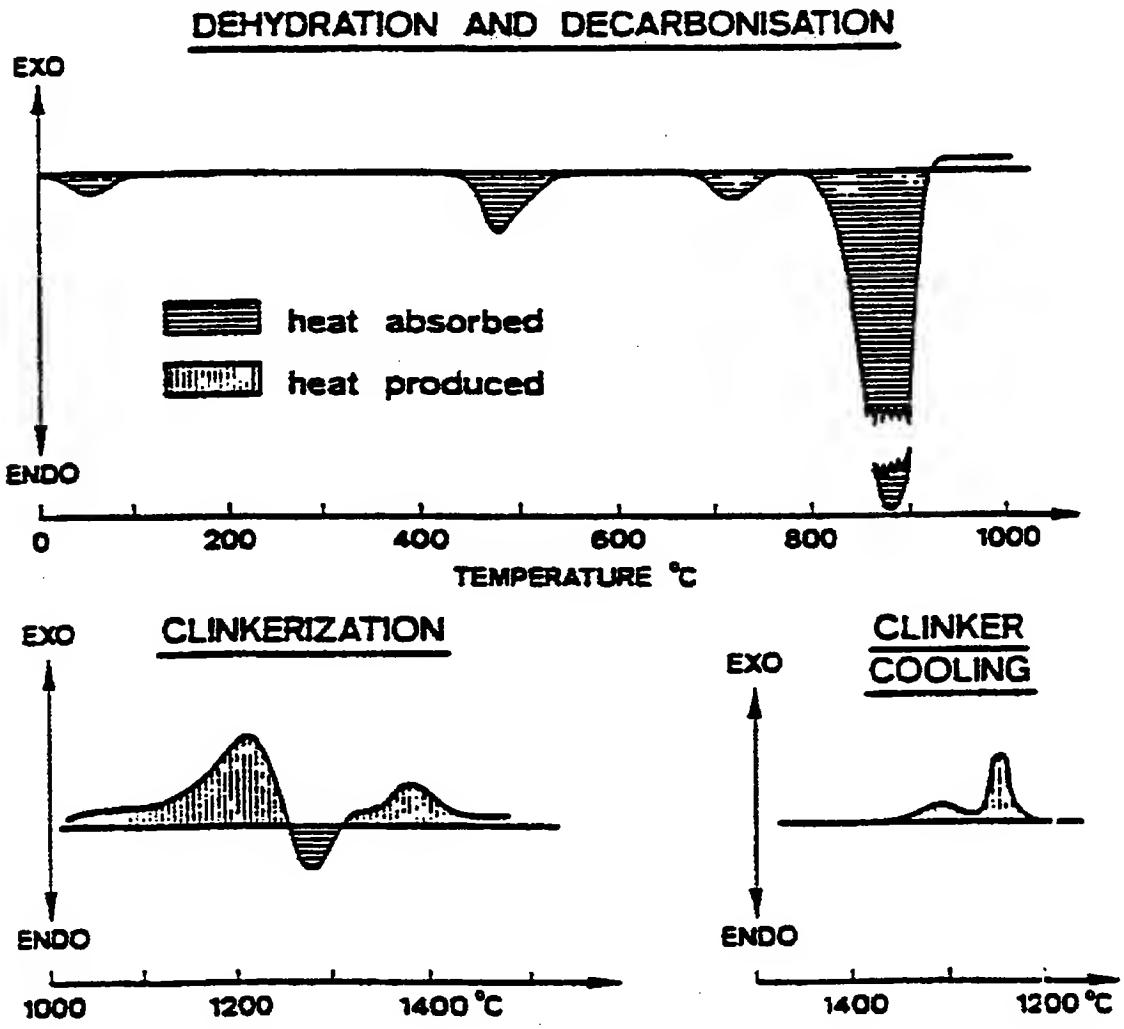
2.1 2.1 Calcining of Raw Meal

Among all reactions taking place when burning clinker, the calcining - also called decarbonisation - requires the highest amount of energy: the dissociation of carbonates, primarily calciumcarbonate according to the reaction

$$CaCO_3 + heat \rightarrow CaO + CO_2$$

in the raw meal requires approx. 1.3 MJ/kg raw meal corresponding to 2.0 MJ/kg cli. The DTA-curves (Fig. 2) illustrate very well the importance of calcining within the clinker burning process.

Fig. 2 Differential Thermo-Analysis (DTA)-curves of a typical cement raw meal



During the process of heating up a raw meal, the calcining does not happen suddenly at a well defined temperature, but starts at about 600 - 700°C and ends between 900 and 1000°C, following a so-called „S curve" (Fig. 3). Exact shape and position of this curve vary from raw meal to raw meal.

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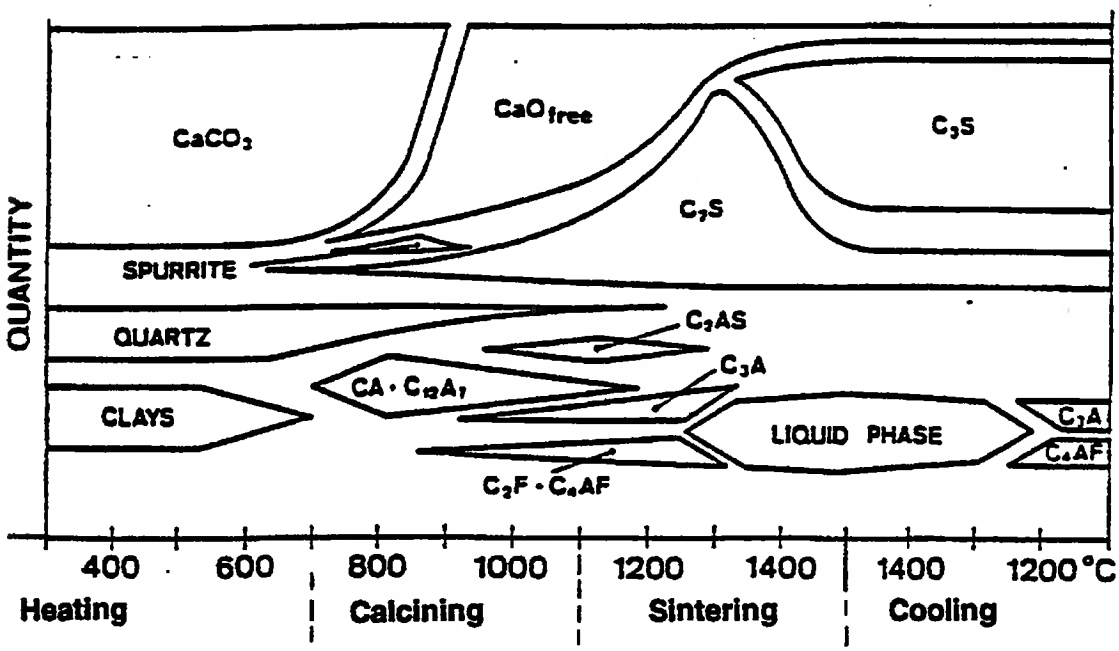
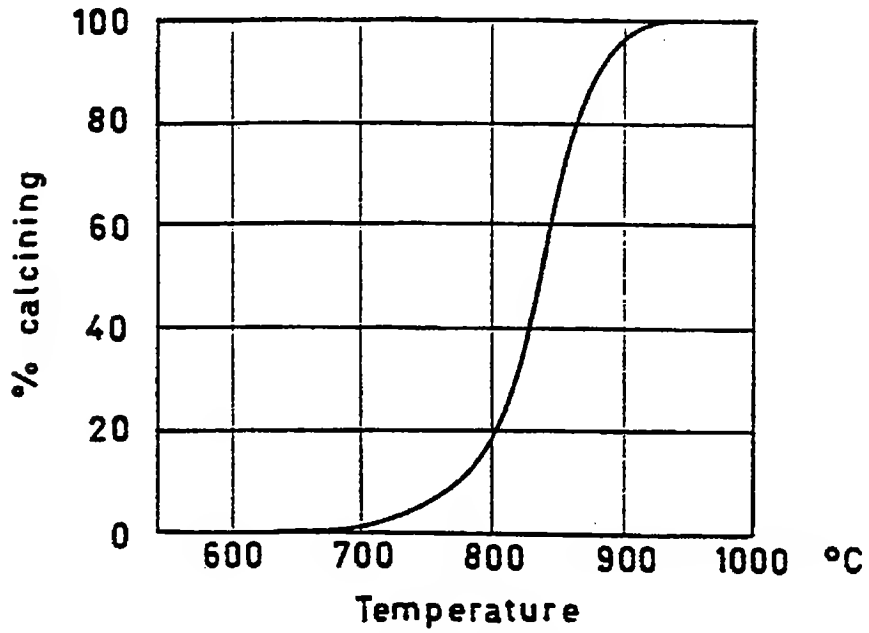
Fig. 3 General aspect of the calcining curve of a cement raw meal





Not only the temperature, but also the retention time of the raw meal is an important parameter of calcining. While the heat transfer from gas to suspended raw meal in a preheater stage is achieved a fraction of a second, the complete calcination at a temperature of about 900°C in suspension requires a reaction time in the range of 2 to 12 seconds. However, as only 90 to 95% of the calcining should take place in the precalciner in order to avoid clogging problems, a residence time of about 1 to 3 seconds has proven to be sufficient.

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To perform both above mentioned tasks, i.e. to keep raw meal in suspension for a few seconds at 850 to 900°C in a stationary vessel without clogging, is the common process target of all PC systems.

2.2 Combustion in Precalciner

The combustion in the precalciner takes place under quite different conditions compared to the main firing because:

- The temperature of the combustion environment is in the order of 850 to 900°C (flame temperature of the main firing: around 2000°C).
- Some PC systems (in-line systems) use an air-gas mixture for combustion (main firing: pure primary and secondary air) while others use pure air (off-line and separate line systems).
- In all PC systems preheated raw meal is suspended in the combustion air or air-gas mixture respectively in order to absorb the heat released thereby maintaining the temperature at a comparatively low level. By all means must Sintering of material avoided, as this would lead to clogging in the precalciner stage.

On the account of the less favorable combustion conditions complete combustion is not always readily obtained, it requires a certain experience to achieve optimum performance. Of the various parameters influencing the combustion performance, the following are perhaps the more important ones:

- Good mixing of the fuel with the available oxygen. (This is particularly difficult to achieve with in-line calciners.) Optimum fuel dispersion into the gas flow (liquid fuel: atomization) is essential.
- Retention time for combustion has to be sufficient. The combustion must be completed in the PC stage. Otherwise, it will continue in the next stage (post-combustion) where the temperature level is lower and therefore less favorable for the calcination (see S-curve). This results in not optimum utilization of the heat which leads eventually to higher fuel consumption.
- The flow pattern of the air/gas mixture (resp. tertiary air) has to be favorable for the combustion.
- The meal distribution in the combustion zone has to be optimum, i.e. causing minimum distortion of the combustion. ($CaCO_3$ as well as CO_2 can also react with C - carbon from the fuel - to produce CO!).

It is known from experience that too high concentrations of raw meal can seriously impede the complete combustion.

With the introduction of the separate air duct for the combustion air for the calciner, the new term of tertiary air had to be introduced:

Primary air:	Air introduced via kiln burner
Secondary air:	Air from cooler to kiln burning zone
Tertiary air:	Air from cooler to PC for combustion

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Introduction of fuel between kiln inlet and bottom cyclone - as secondary firing or via precalciner - necessarily increases the temperature level. The gas exit temperatures from the lowest stage of a straight preheater kiln is only 790 to 820°C as compared to precalciner kilns where this temperature increases by some ten degrees to 840 to 870°C. Therefore, the preheater exit temperature is also somewhat higher entailing an increased heat loss, which is more pronounced with 4-stage preheaters.

The performance of PC systems can primarily be judged on two characteristic values:

- The temperature difference between gas and material ex precalcining stage should be as low as possible, so as to minimize the heat losses of the exit gas. The reaction temperature in the precalciner depends of course on the raw meal and the required precalcining degree as well as tolerated NOx level.
- Complete combustion must be achieved as this directly influences the overall heat consumption of the system. It must be mentioned that this is strongly influenced by the excess of air.
  - Note: Stating the amount of unburned matter in the gas is therefore only meaningful to assess a calciner system, if the amount of oxygen in the gas is indicated as well.

Solid, liquid and gaseous fuels are successfully fired in PC kiln systems. However, the location and position of the burners in the precalciners have to be adapted to the fuel particularities. This is specially important for gaseous fuels, which seem to be more difficult to burn in the PC chamber than other fuels.

2.3 Specific Heat Consumption

From the above mentioned it can be concluded that PC systems have a tendency to slightly increased heat consumption, unless countermeasures are taken such as:

- Although equipping existing preheater kilns with precalcination usually results in a slight increase of the heat consumption, the average (annual) heat consumption may be equal or even lower on account of a more regular kiln performance.
- Also for new installations the heat consumption is about 50 - 100 kJ/kg cli higher than for conventional preheater kilns with 4 stages.

Where the somewhat higher exit gas temperature cannot be fully used, say for raw material drying - then it has become standard to install one or two additional preheater stages to reduce the heat consumption to a figure slightly, for 6 stages noticeably, below that of a conventional 4st SP kiln. The first PC kiln in Dotternhausen was in fact equipped with a 5-stage preheater.

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2.4 True and Apparent Calcination Degree

An important parameter for controlling the precalciner operation is the calcination degree. It is important to know the meaning of the true and the apparent calcination degree.

True calcination degree:

Degree to which the calcination is completed, i.e. extent to which the CO_2 is dissociated from the $CaCO_3$.

Extremes:	Raw meal	0% (LOI = 35%)
	Clinker	100% (LOI = 0%)

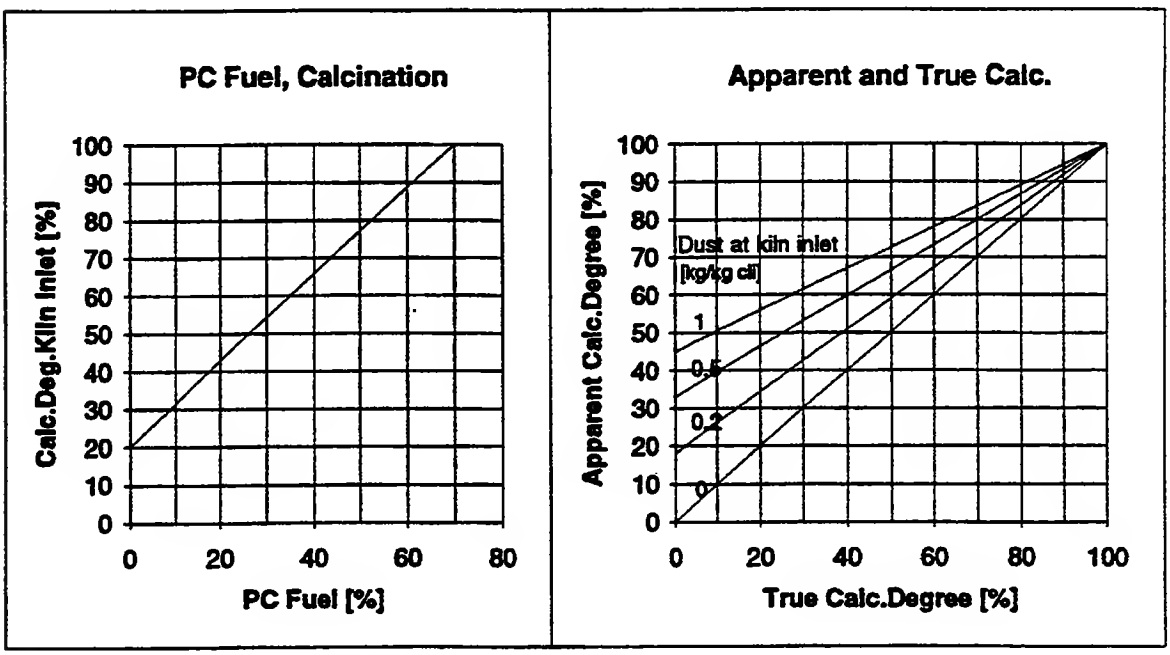
In reality, the calcination degree is determined using a hot meal sample taken from the meal duct of the bottom cyclone. Because of always present dust cycles between kiln / kiln inlet / kiln riser / bottom cyclone, this sample contains a certain amount of dust which was already in the kiln calcining zone and is higher or even fully calcined. This sample is therefore a mixture consisting of „fresh" meal and dust circulated back and has a higher calcination degree than the pure „fresh" hot meal.

This means: The higher the dust concentration near the kiln inlet resp. the dust cycle, the higher the apparent calcination degree.

Apparent calcination degree:

The calcination degree determined from a hot meal sample taken from the meal duct of the bottom cyclone.

Fig. 5 True and Apparent Calcination Degree, PC Fuel, Dust



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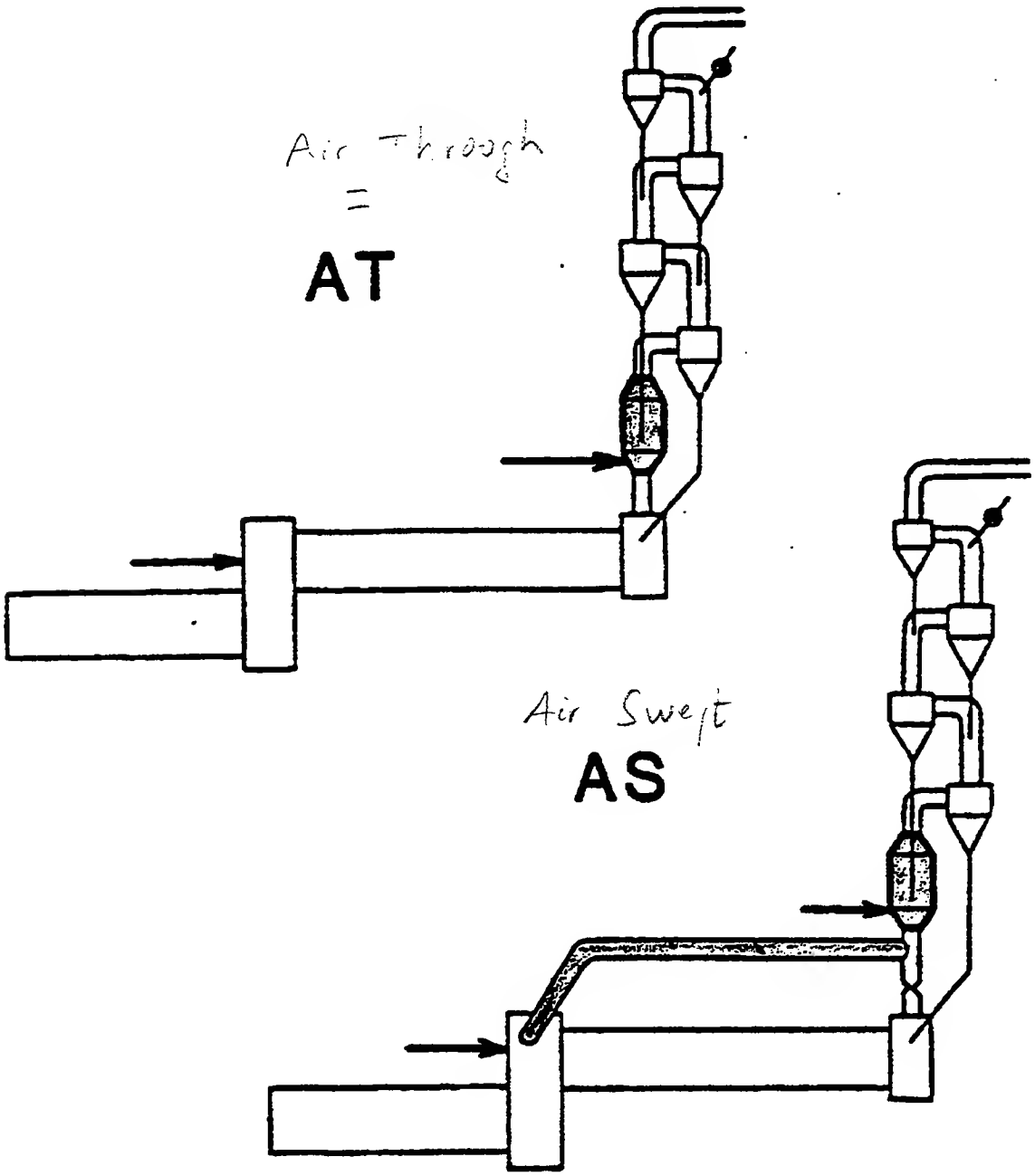
3. BASIC ARRANGEMENTS OF PRECALCINING SYSTEMS

3.1 AS and AT Systems

The first precalciner in Dotternhausen used combustion air which was drawn through the kiln as excess air. This technology was maintained for several years and is known as AT System. However, in reality only up to 35% fuel could be introduced to the precalciner thus limiting its benefits. The AT type is thus no longer considered a precalciner; it is rather used for secondary firings where a high calcination degree at the kiln inlet is not the main target.

Fig. 7 AS and AT Systems



Today, all precalciners are AS Systems using tertiary air which is extracted from the kiln hood or from the cooler roof and drawn via a separate tertiary air duct parallel to the kiln to the precalciner. This means that planetary coolers are not compatible with precalcination technology (i.e. AS systems).

Table 1 Comparison of AS and AT System

Item	AS	AT
Portion of fuel to the precalciner	up to 65%	max. 35%
Largest kiln in operation	8500 t/d, ϕ 6.2 x 105 m	4700 t/d, ϕ 5.2 x 80 m
Kiln ϕ for given capacity (st SP = 100%)	approx. 75-80%	approx. 85-90%
Suitable type of cooler	**only grate or rotary**	all types
Suitable for extension of existing SP kiln	poor (cooler, tertiary air duct)	very good for low PC rates
Burning conditions in rotary kiln	normal flame temperature (normal excess air)	lower flame temperature and stable operation due to **high** excess air
Thermal load in burning zone (4st SP = 100%)	approx. 60-70% at 60% PC	approx. 85-90% at 30% PC
Behavior regarding circulating elements	like 4st SP kiln	due to the high O2-content of the kiln atmosphere, **reduced volatility** of sulfur and therefore decrease of encrustation in transition zone and riser pipe
Heat loss at 10% bypass (4st SP = 100%)	approx. **40%** (bypass will be smaller than in 4st SP kiln)	approx. **90%** (bypass will have same size as in 4st SP kiln)
Exhaust gas temperature (4st preheater)	higher than 4st SP	higher than 4st SP
Heat consumption	slightly higher than 4st SP	slightly higher than 4st SP
Pressure loss over preheater	higher than 4st SP	slightly higher than 4st SP

$T_{max\ gas} = 260\,°C \approx 290\,°C$

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3.2 In-Line, Off-Line and Separate Line Calciners

This criteria refers to the position of the precalciner in the kiln system installation and is illustrated with Fig. 8 below.

- In-Line Calciners are installed in the kiln exhaust gas flow which means that the combustion takes place in an air/kiln gas mix. This precalciner can be considered an enlarged kiln riser duct.
- Off-Line Calciners are installed off the kiln exhaust gas flow. The combustion takes place in pure (tertiary) air which is also responsible for lifting up the meal.
- Separate Line Calciners are off-line calciners with a separate preheater string.

Fig. 8 Precalciner Arrangements

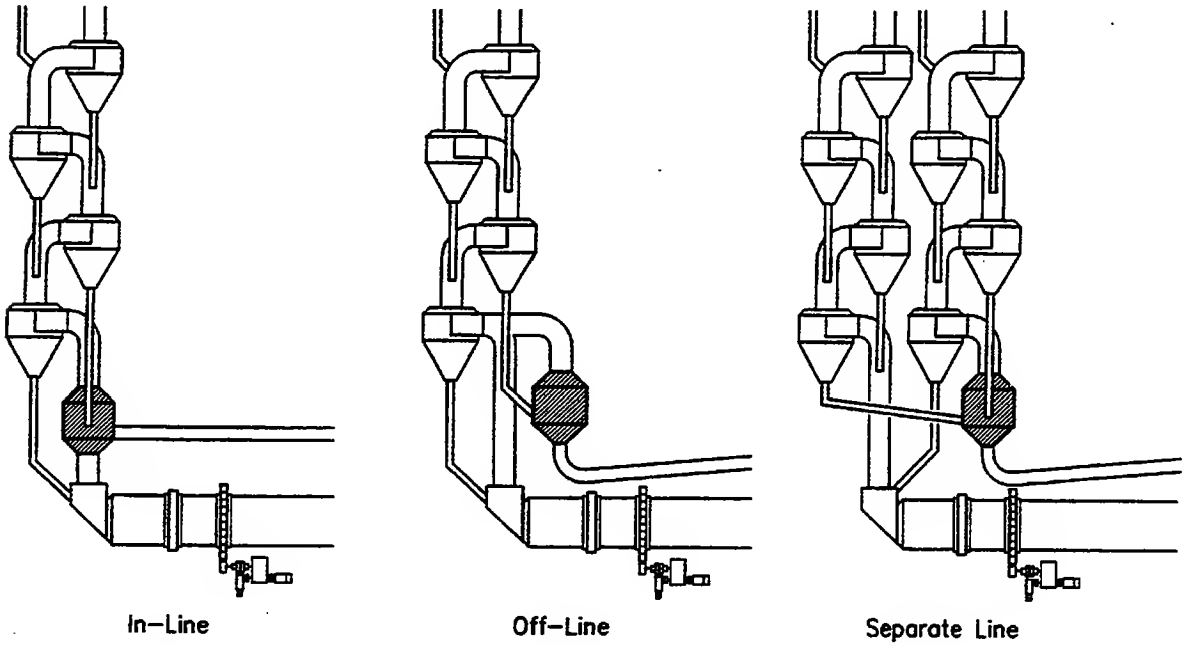

Table 2 Comparison of Calciner Arrangements

	In-Line	Off-Line	Separate Line
PC arrangement	Extended riser duct	Parallel to riser duct	Parallel to riser duct
Combustion atmosphere	Kiln gas and air mix	Pure air	Pure air
Preheater string	1 to 4 of same size	1 to 4 of same size	2 to 4, 2 different sizes
Advantages	Low NOx version (reducing kiln NOx)	Suitable for modification	Two independent combustions → Easy combustion control
	Excess air used for combustion	Good combustion	Good combustion
	Suitable for lump fuel	Suitable for modifications	Suitable for modifications
Weak points	Mixing of air with gas	Higher peak temperature (NOx!)	Higher peak temperature (NOx!)
	Larger volume required	PC drop-out can fill TAD	PC drop-out can fill TAD
	Incomplete combustion		Asymmetry regarding circulating elements
	Height requirement (depending on type/design)		Requires 2 strings (not feasible for <3000 t/d)
			Strings of different sizes (problem >7000 t/d)

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4. FEATURES OF PRECALCINERS

4.1 Main Benefits of Precalciner Technology

There are many advantages of precalciner technology which made it state of the art today. Some of them are listed here:

1) More stable kiln operation due to better kiln control via two separate fuel feed/control points.
2) More stable kiln operation due to controlled meal conditions at kiln inlet.
3) Reduced thermal load of burning zone.
4) Lower brick consumption as a result of 1. and 3.
5) More than double capacities possible with given kiln (10'000 t/d with 6 m x 95 m kiln).
6) Possibility of increasing capacity of existing kilns.
7) Reduced volatilization of circulating elements.
8) Reduction of cycles (S, Cl, Na20, K2O) with smaller bypass rate, i.e. lower losses.
9) Makes short kilns possible with 2 stations, L/D < 12
10) Possibilities of NOx reduction.
11) Lump fuel utilization in some cases.

4.2 Limitations and Restrictions

Even though the advantages of precalciner systems are doubtlessly convincing, not all types can be used in all cases. Limitations are:

- Additional installation (fuel dosing, calciner, tertiary air duct) as well as the relatively smaller rotary kiln sets a lower economical limit to PC systems for new plants at around 1200 t/d.
- Alternative fuels containing hazardous components can only be used in the main firing due to the high temperature level there. The potential to use such fuels is then lower for PC kilns.
- Higher exhaust gas temperature and higher pressure drop can be a drawback in specific cases.
- Separate line calciners for new installations are only feasible if a 2-string arrangement is required for the capacity, i.e. above 3500 t/d.

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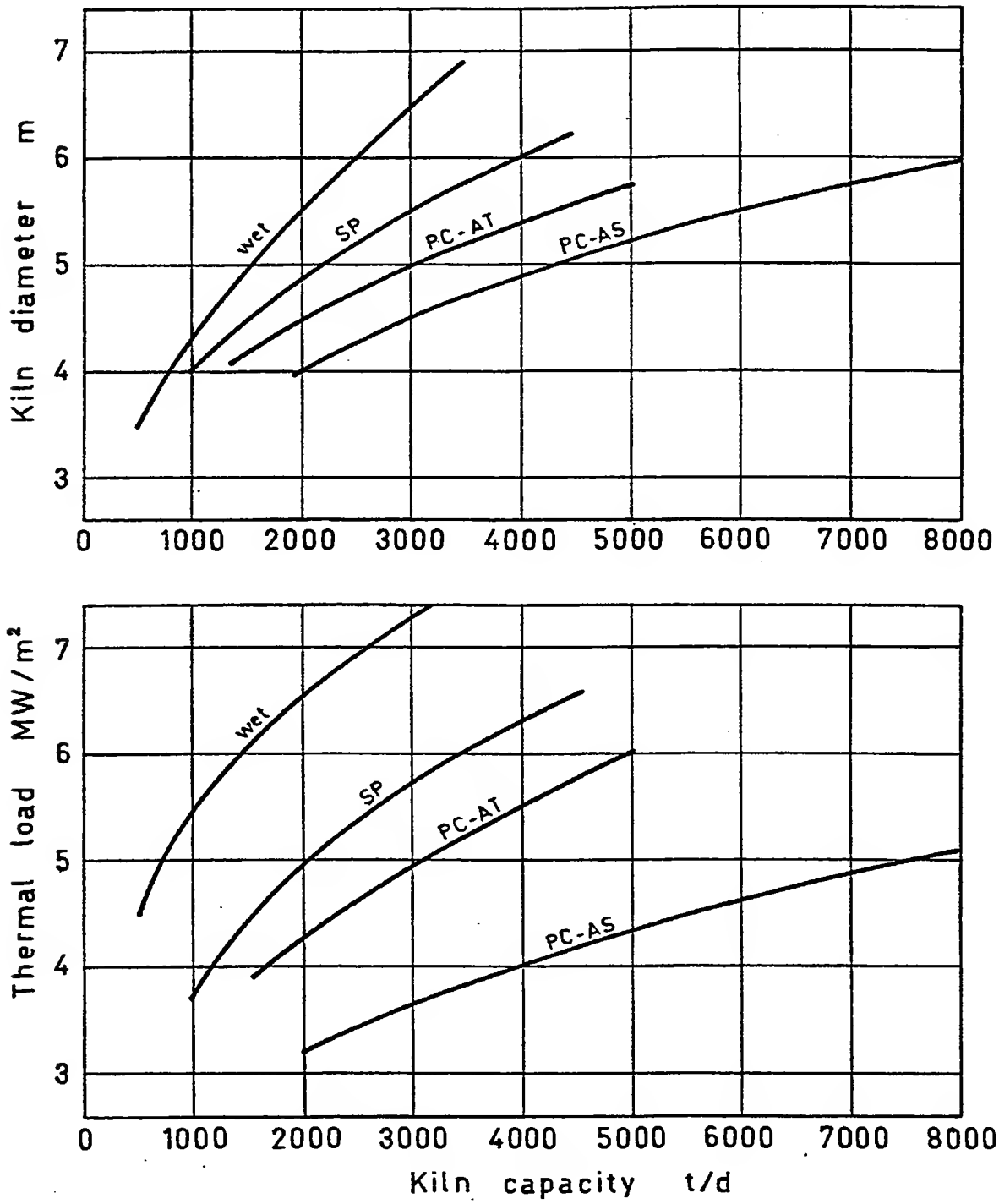
Fig. 9 Comparison of wet, SP and PC Kilns (average curves)



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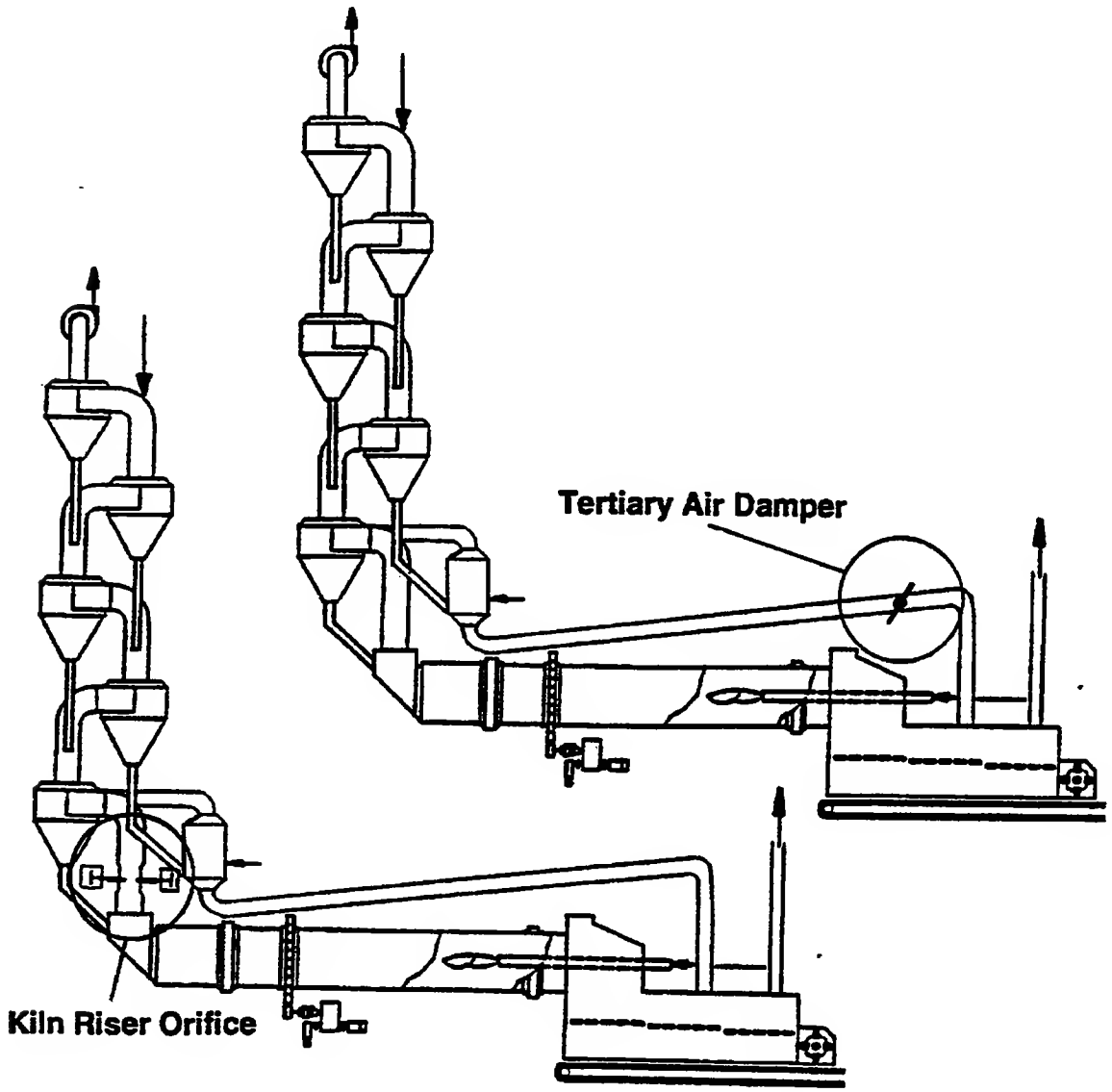
4.3 Tertiary Air Damper and Kiln Riser Orifice

Off-line calciners as well as in-line calciners are usually equipped with one kiln ID fan. In order to allow control of the tertiary air/secondary air ratio, there must be a control device in at least one of the two gas paths (kiln resp. tertiary air duct).

For efficient warming up of the preheater, a damper is usually installed in the tertiary air duct to avoid fresh air to bypass the main flame. Very often, this damper is used also, for controlling the tertiary air flow (Fig. 10a). However, experience shows that high temperature and clinker dust require a quite refined design of this tertiary air damper. In many cases, this damper operates only for a short period without problems.

Another approach is to install the control device in the other path. Some suppliers (e.g. FLS and Kawasaki) have developed a kiln riser orifice which is successfully operating in several plants. This solution (Fig. 10b) is generally more expensive than the TA damper above, but performs well.

Fig. 10 TA Damper and Kiln Orifice



4.4 Circulation Problems and Bypass with PC Kilns

Precalciner kiln systems have two major advantages regarding circulation problems.

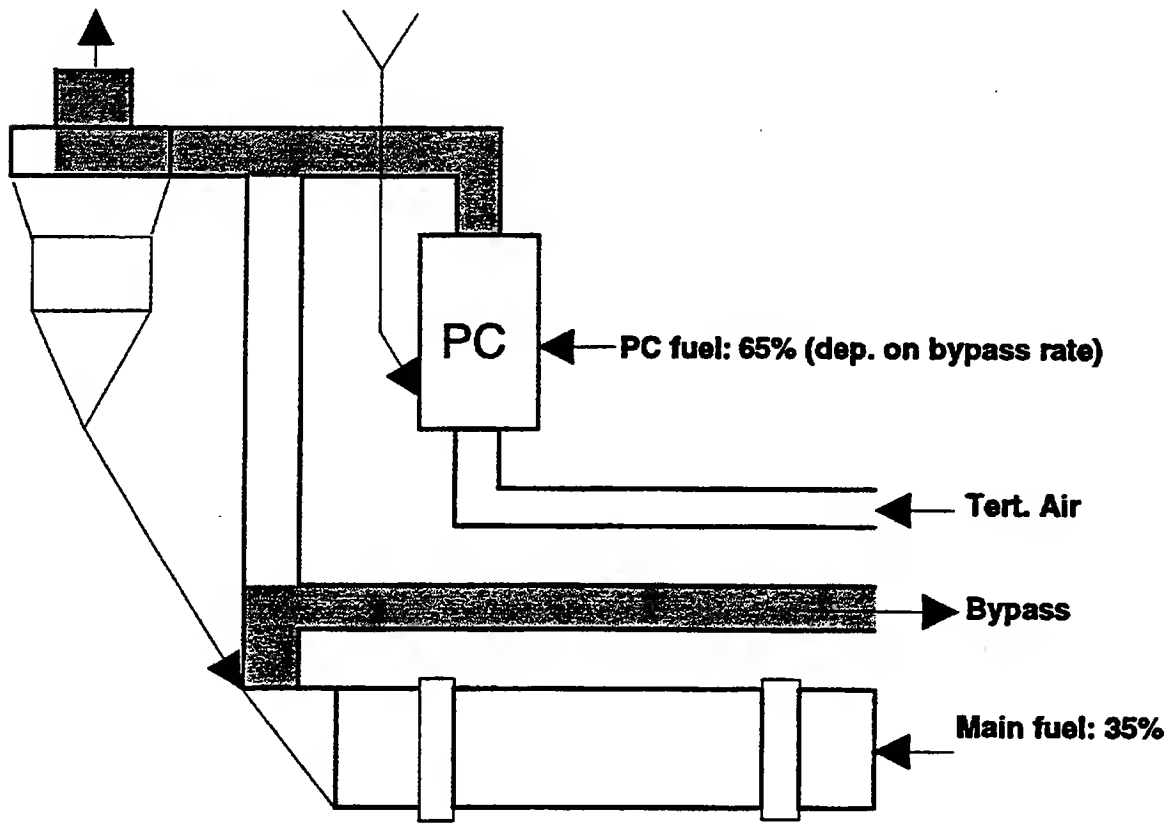
- Reduced volatilization in the rotary kiln because less than 50% of the heat is introduced in the burning zone.
- Less than 50% thermal and dust losses in case of a bypass compared to a straight SP kiln.

The volatilization of circulating elements occurs primarily in the rotary kiln. The percentage of the volatilized elements which can be extracted with a bypass depends on

- volatilization rate in the kiln, and
- amount of kiln gas extracted via bypass (= bypass rate) which is expressed by the ratio:
  bypass gas
  gas at kiln inlet

The highest possible reduction of circulating elements at a given volatilization rate would be if 100% of the gases at the kiln inlet could be extracted. this is only possible in the case of a precalciner but not with a straight preheater kiln. Accordingly are the heat losses approx. 50 to 60% lower at a given reduction because the concentration of volatilized circulating elements in the gas at the kiln inlet is much higher than for a SP kiln.

Fig. 11 Bypass for PC Kilns



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5. PRESENT STATE OF PRECALCINER DEVELOPMENT

5.1 Calciners from FCB

FCB have been IHI licensees since the mid seventies for in-line calciners resulting in 8 operating installations and 6 under construction. The highest capacity is 3300 t/d (Tourah, Egypt).

Together with Ciments Français FCB have designed a new type of calciner with low emissions suitable for low grade fuels called the FCB low NOx PC (Fig. 12).

In combination with a low NOx kiln burner, FCB expect to achieve 150 - 350 ppm NOx at the stack with their new calciner. The first industrial prototype is scheduled for 1992.

The FCB calciner looks like a vertical reactor with one three channel burner on the top. Tertiary air is introduced from the top as well as with the meal on two sides. Flow is vertical from top to bottom. Meal can be proportioned via three points on two levels. FCB claim to achieve:

- Hot spot
- Reducing atmosphere zone → NOx reduction
- Controlled flame
- No separation of coal and meal

5.2 Calciners from FLS - FULLER

The FLS range of calciners will be marketed by both FLS and FULLER.

Three basic air separate calciner systems are available: ILC, SLC-S and SLC (Fig. 13).

All these use a vessel type calciner which provides retention time by means of volume. Experiences with this system made no conceptual changes necessary.

The only modification to be mentioned is the new tangential tertiary air inlet for the ILC system which allows larger calciner volume without requiring more height.

Main features of the FLS calciner systems presently available are:

- Variable kiln orifice (Fig. 14) for the SLC-S calciner to control the ratio of secondary to tertiary air in place of the often troublesome damper in the tertiary air duct.
- Low NOx version by splitting the tertiary air creating a controlled area of reducing atmosphere in the lower part of the PC which is horizontally divided in two zones by an orifice.
- Variation of the calciner outlet temperature with the SLC-S system without changing the preheater temperature profile providing a „temperature window" for NH_3 injection.

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5.3 PYROCLON Calciners (KHD)

The calciner systems by KHD (and Polysius) are based on the 1965 Dottenhausen „goose neck" design, a tube type calciner. As PYROCLON-R, a whole range of versions has been developed (Fig. 15).

A low NOx version of the RP version is not available. KHD tackle the problem of CO from incomplete combustion with coal firing by focusing on improved coal dosification.

Incomplete mixture of waste gases from kiln and calciner is often found with tube type calciners. In order to achieve a good mixture, an essential prerequisite for low NOx systems using excess fuel zones in the precalciner, the 180°C elbow is substituted by a new reaction chamber, called PYROTOP (Fig. 16).A PYROCLON-R Low NOx with PYROTOP allows:

- Complete combustion of the calciner fuel
- Temperature controlled zones (NH_3 injection)
- Improved mixing of gases
- Reduction of NOx

5.4 PREPOL® Calciners (Polysius)

Polysius calciners are all of the air separate (AS) h-line-type. It is generally accepted today that the calcination process takes place within a few seconds making the fuel reactivity the decisive design criteria for the calciner size.

The „goose neck"-tube type calciner PREPOL by Polysius is presently available in three basic configurations (Fig. 17).

Several Polysius calciners have been modified by the company CLE who added an RSP type pre-combustion chamber. The same principle is now incorporated in the PREPOL AS-CC calciner by Polysius.

Polysius started in 1985 to develop their NOx reducing technology called MSC based on experience available from power stations with staged combustion. They have adapted this method to the requirements of the clinker burning process. Trial operation on cement plants have shown 35 - 45% reduction of NOx.

The idea is to create a limited zone of reducing atmosphere near the transition chamber by adding a small amount of fuel to the rotary kiln exhaust gas via a small burner in the riser duct. For the NOx from the calciner fuel, the same principle is applied resulting in a second reducing zone. Such a system would have the following fuel inputs:

- < 50% main burner
- < 10% via primary DeNOx burner
- > 30% via precalciner
- < 10% via secondary DeNOx burner

Experience on an industrial scale only will prove the capability of this system. One of the difficulties is how to control the kiln atmosphere without the gas analysis sampled near the kiln inlet.

5.5 Prerov-Calciner

The Czek company Prerov have developed a new precalciner (Fig. 18). It consists of a precombustion chamber (KKS) and a reaction chamber (KKN) with a vortex chamber and is comparable to Polysius' PREPOL-AS CC. During 1992, the first installation will be commissioned in Southern Italy.

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5.6 Conclusion

The development of tube type calciners and vessel type calciners has moved them closer to each other. The tube type calciners have received a swirl pot or a pre-combustion chamber for improved mixing and fuel burning and the vessel type calciners have become longer.

The calciner without separate air duct also known as „air through" actually operating only with 10 - 20% of the total fuel never fulfilled the expectations and has virtually disappeared, together with the planetary cooler.

Low NOx calciners have been developed based on the principle of locally reducing atmosphere by means of fuel excess zones. It can be expected that NOx from precalciner combustion can be reduced to around 700 - 800 ppm. Calciners can be designed to reduce NOx generated in the burning zone, or to keep NOx generated in the calciner low, or both.

Since further NOx reduction to lower levels require methods such as NH_3 injection, temperature control is very important.

A modern calciner can be described as follows:

Type:	in-line with pre-combustion chamber
Fuel ratio:	50 - 60% (include. low NOx fuel in case of staged combustion
Fuel dosing:	low fluctuation
Fuel types:	various, including alternative fuels
Combustion environment:	pure air or air/kiln gas mix
Calciner size criteria:	fuel reactivity gas retention time (up to 4 - 5 sec.)
Feature:	enhanced turbulence
Tertiary air:	staged for reducing zone

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6. SYNOPSIS OF PRECALCINERS

The different PC systems as well as their developers and suppliers are summarized in Table 3. During the 1970ies the cement manufacturers greatly contributed to the development of the Japanese PC systems:

Until 1985, ot 304 kilns with PC, 83 were located in Japan, totaling 35% of the capacity. This shows the explosive expansion of PC systems in Japan back than. Today, all new kilns have precalciner with tertiary air duct.

Table 3 Synopsis of PC Systems

Trade Name	Signification	Developer & Licenser	Plant Supplier & Licensee
PASEC		Voert Alpine / SKET	ACT
SLC	Separate Line Calciner	F.L. Smidth	F.L. Smidth
SLC-S	Separate Line Calciner Special		
ILC	In-Line Calciner		
ILC-D	In-Line Calciner Downdraft		
ILC-E*	In-Line Calciner, Excess Air		
Prepol AS	Air Separate	Krupp-Polysius	Krupp-Polysius
Prepol AS-CC	Controlled Combustion		
Prepol AS-MSC	Multi Stage Combustion		
Prepol AT*	Air Through		
Pyroclon R	Regular = Air Separate	KHD Humboldt Wedag	KHD Humboldt Wedag
Pyroclon RP	Regular Parallel		
Pyroclon R Low NOx			
Pyroclon R Low Nox with Pyrotop			
Pyroclon S*	Special = Air Through		
EVS-PC (only fuel - oil)	Echangeur à voie sèche avec précalcination	Fives-Cail Babcock	Fives-Cail Babcock
KKS-KKN	n.a.	Prero	Prerov
SF	Suspension Flash Calciner	Ishikawajima-Harima Heavy Ind. Chichibu Cement	Ishikawajima-Harima Heavy Ind. Fuller Company / Fives-Cail Babcock
NSF	New SF		
RSP	Reinforced Suspension Preheater	Onoda Cement	Onoda Engineering & Consulting Kawasaki Heavy Industries Allis-Chalmers CLE-Technip
KSV	Kawasaki Spouted Bed and Vortex Chamber	Kawasaki Heavy Industries	Kawasaki Heavy Industries
NKSV	New KSV		
MFC	Mitsubishi Fluidized Calciner	Mitsubishi Mining & Cement	Mitsubishi Heavy Industries
GG	Reduction Gas Generator	Mitsubishi Heavy Industries	
DDF	Dual Combustion and Denitration Furnance	Nihon Cement	Kobe Steel
CSF (CFF)	Chichibu Suspension Flash Calciner	Chichibu Cement	Chichibu Cement (own plants)
SCS	Sumitomo Cross Suspension Preheater and Spouted Furnace Process	Sumitomo Cement	Kawasaki Heavy Industries Ishikawajima-Harima Heavy Industries

*Air through: secondary firing systems

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7. TEST QUESTIONS

1) Which is the chemical reaction with the highest heat consumption within the clinker burning process? How much does it consume in absolute terms (kJ/kg clinker) and in percent of the total heat consumption of a modern kiln system?
2) Which are the three basic precalciner arrangements and what are their differences?
3) At what temperature does the calcination take place and how much CO_2 is totally dissociated from the $CaCO_3$?
4) Which are the benefits of precalciner technology?
5) Which is the most important design criteria for precalciner dimensioning?
6) Explain the term „apparent calcination degree". How can it be determined and what is its significance?
7) How do the effects of a bypass compare in case of a straight preheater kiln and a precalciner kiln?

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Fig. 1 **Sketch of Dotternhausen Kiln, the first Precalciner (KHD, 1966)**

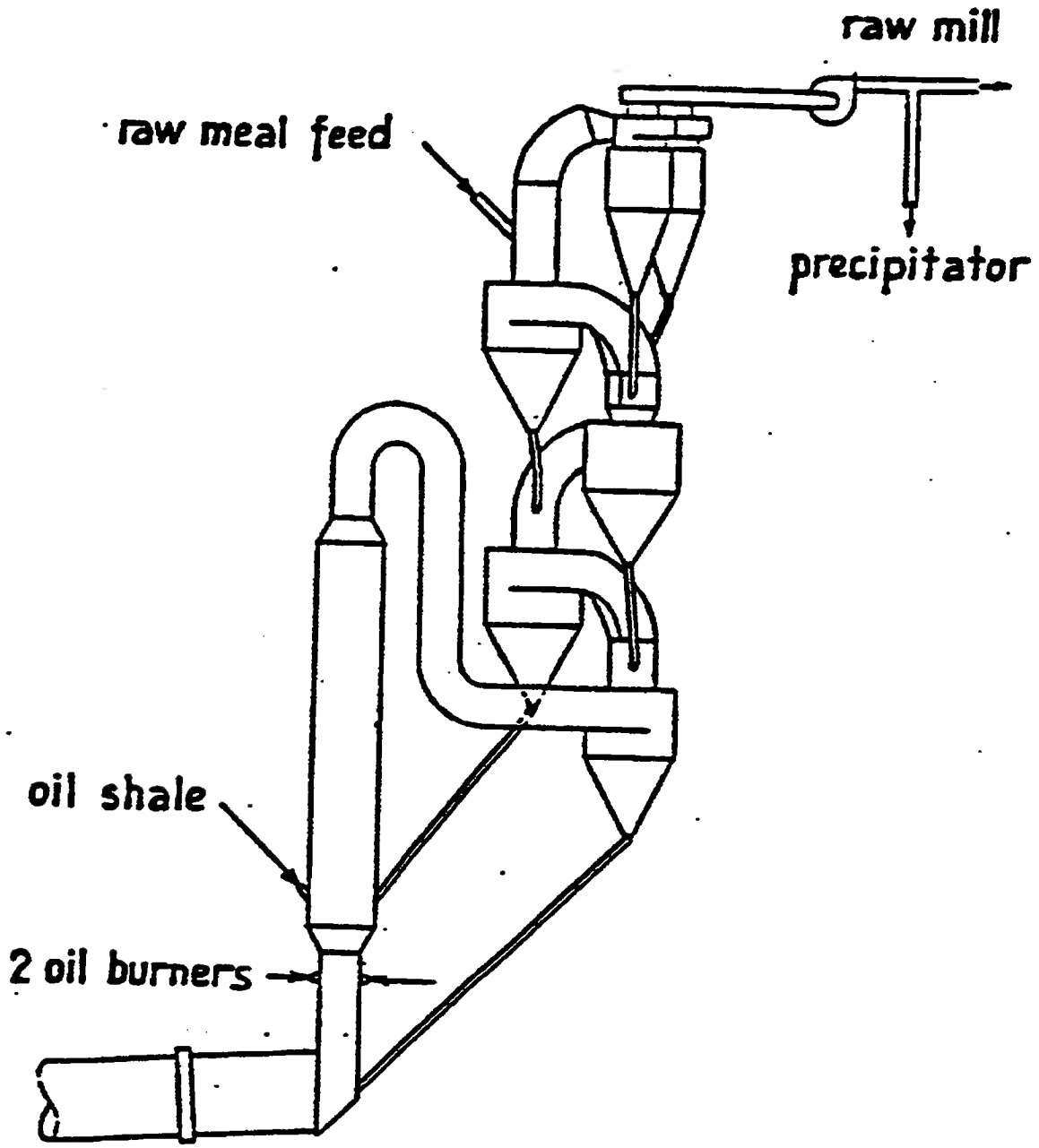

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Table 4: Temperatures and Process Steps for Clinker Burning

Temperature [°C]	Process Step, Type of Reaction	Heat
20 - 100	Evaporation of free H_2O	Endo
100 - 300	Loss of physically absorbed H_2O	Endo
400 - 900	Removal of structural water	Endo
> 500	Structural changes in silicate minerals	Exo
600 - 900	**Dissociation of CO_2 from $CaCO_3$**	**Endo**
> 800	Formation of intermediate products Belite, Aluminate and Ferrite	Exo
> 1250	Formation of liquid phase (aluminate and ferrite melt)	Endo
	Formation of alite	Exo
1300 - 1240	Crystallization of liquid phase into mainly aluminate and ferrite	Exo

For numerical calculations, an approximate quantity of CO_2 from the raw material (dissociated from the calcites) can be used, regardless of the exact chemical composition.

CO2 from raw mat = 0.28 Nm3/kg cli

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Table 5: Energy Balance of Process Steps for Clinker Burning

Endothermic Processes:	**kJ/kg cli**	**kcal/kg cli**
Dehydration of clays	165	40
Decarbonisation of calcite	**1990**	**475**
Heat of melting	105	25
Heating of raw materials (0 to 1450°C)	2050	490
Total endothermic	4310	1030
Exothermic Processes:	**kJ/kg cli**	**kcal/kg cli**
Recrystallization of dehydrated clay	40	10
Heat of formation of clinker minerals	420	100
Crystallization of melt	105	25
Cooling of clinker	1400	335
Cooling of CO_2 (ex calcite)	500	120
Cooling and condensation of H_2O	85	20
Total exothermic	2550	610
Net Theor. Heat of Clinker Formation:	**kJ/kg cli**	**kcal/kg cli**
Endothermic - exothermic	1760	420

Heat consumption of Kiln System:	**kJ/kg cli**	**kcal/kg cli**
Average 4-stage SP system	3300	790
Modern 6-stage SP system	3000	720
Rel. Heat Requirement of Calcination:		
Average 4-stage SP system	60%	
Modern 6-stage SP system	66%	

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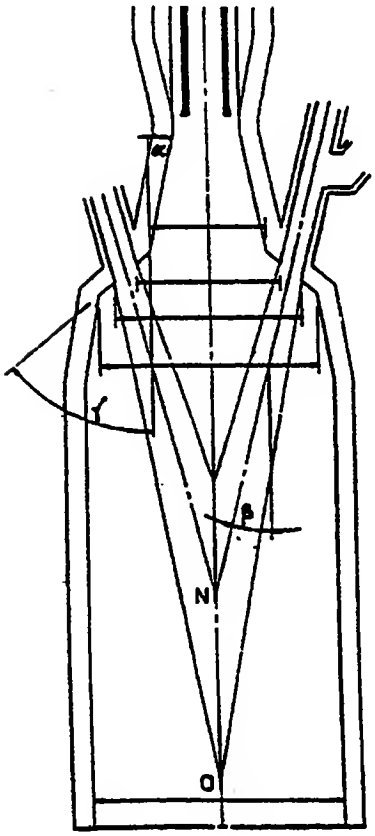
Fig. 12 FCB Low-NOx Precalciner



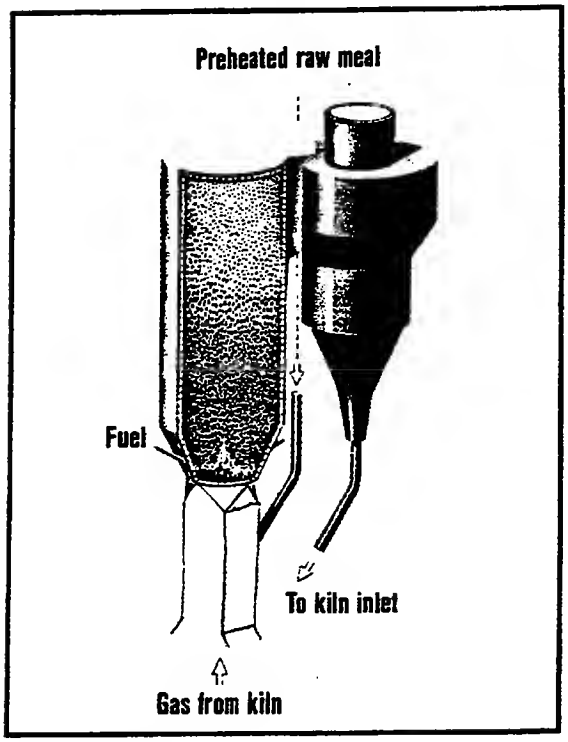
fcb

FCB LOW-NO_x PRECALCINER

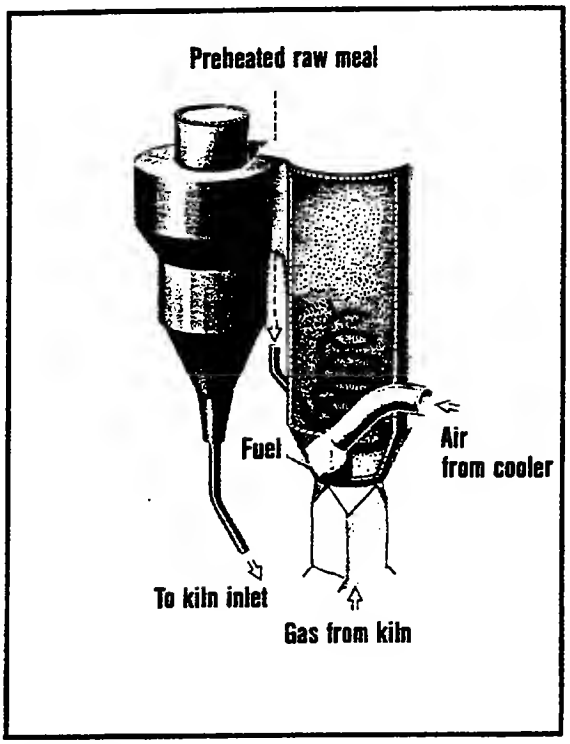
A High-performance burner	⇨	Ignition of all fuels
A reducing first zone	⇨	Low NO_x production
A controled mixing in the reactor	⇨	Better burn out

Fig. 13 **FLS**

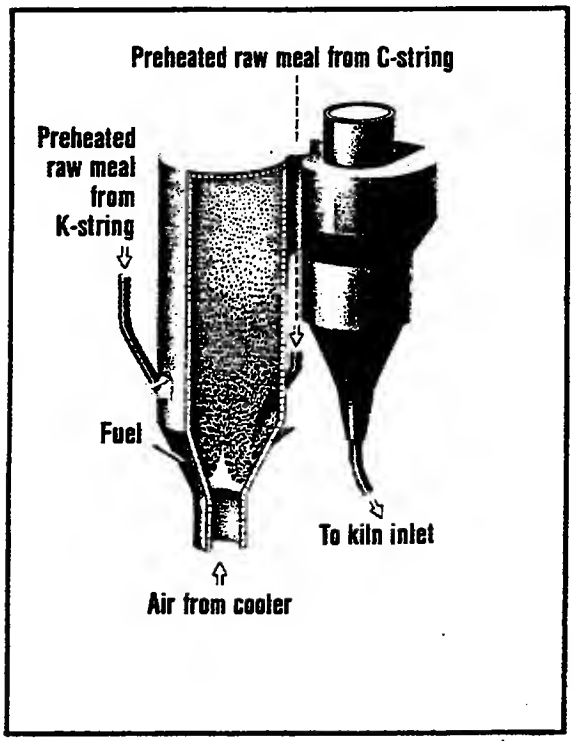
FLS F.L.SMIDTH



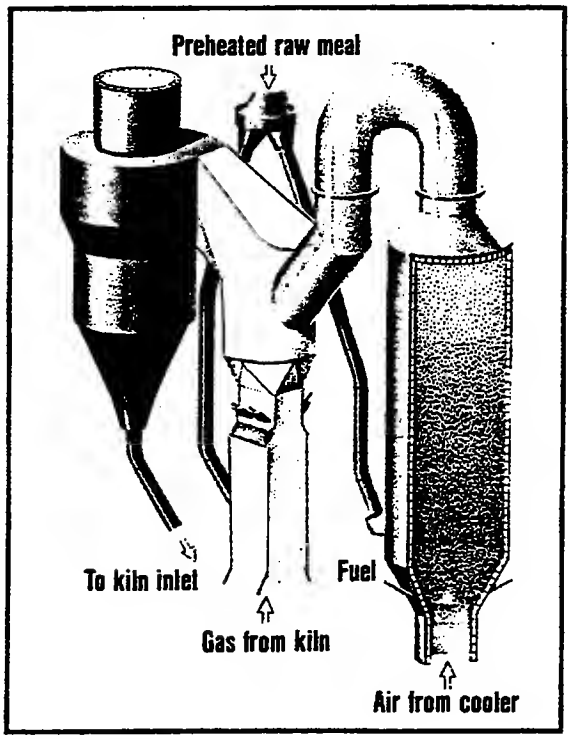
ILC-E calciner.



ILC calciner.



SLC calciner.



SLC-S calciner.

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Fig. 14 FLS Adjustable Kiln Orifice

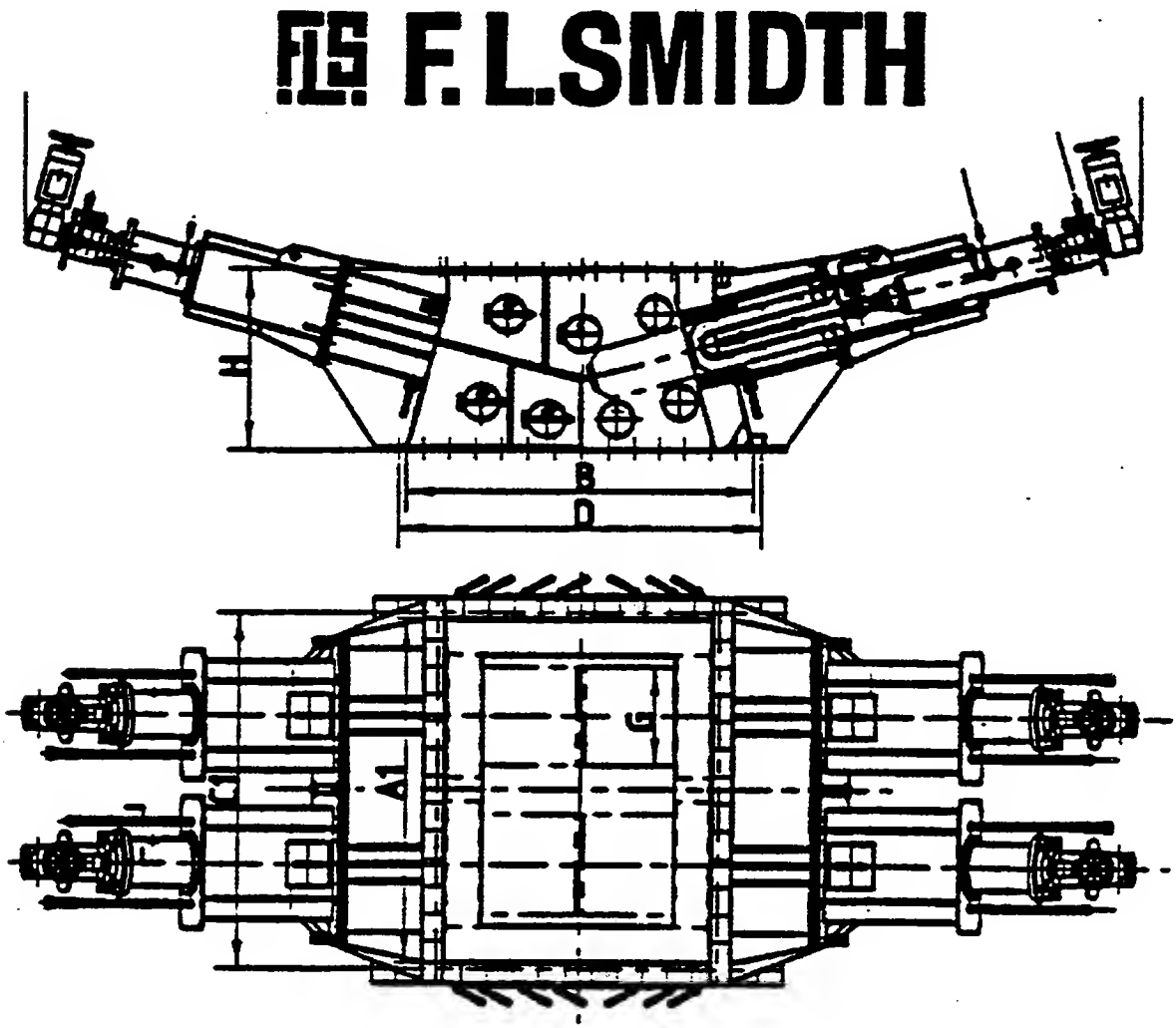

Fig. 15 Pyroclon

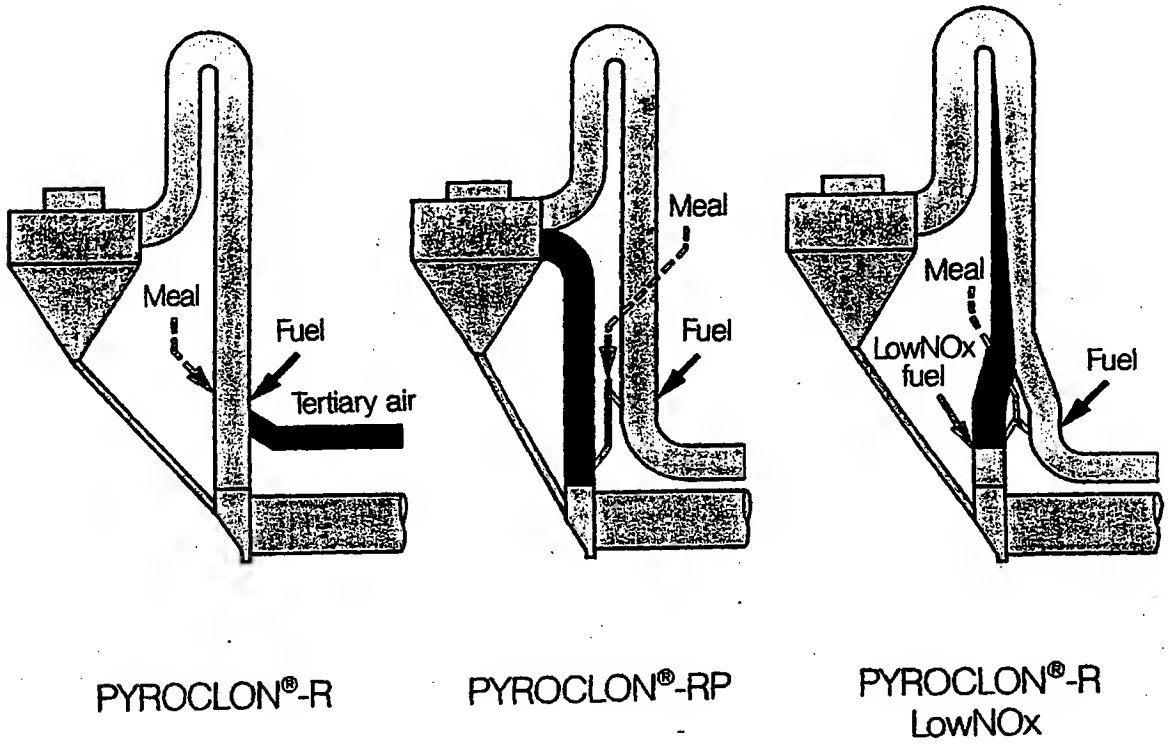

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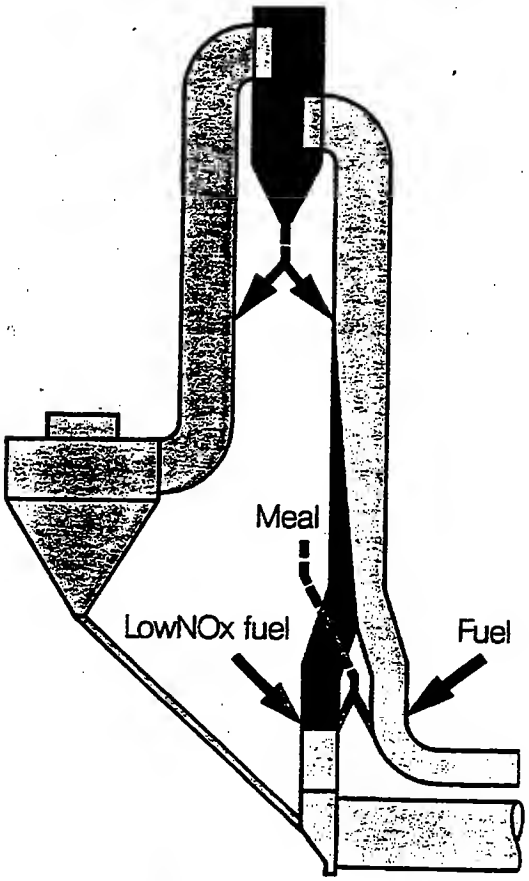
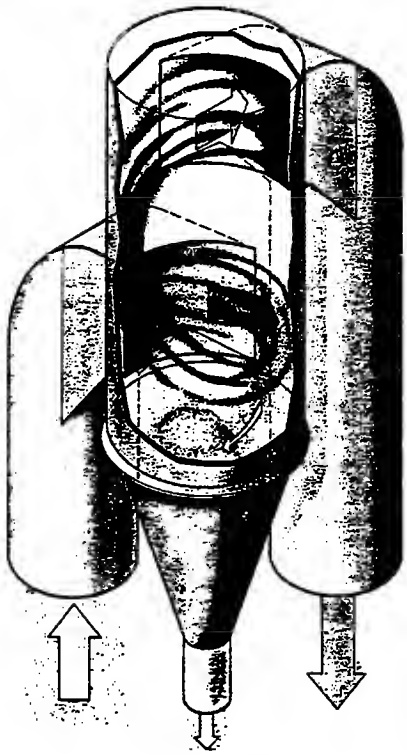
Fig. 16 Pyrotop

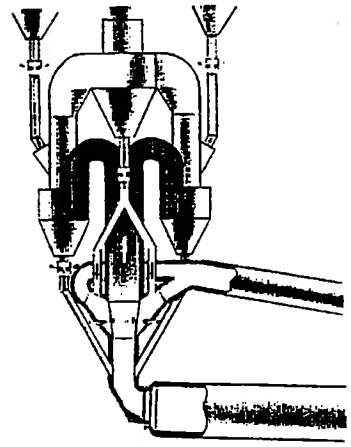


PYROTOP-calciner

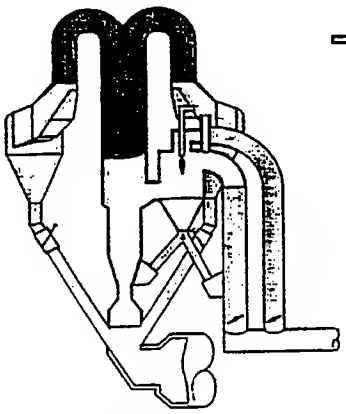


Flow pattern in PYROTOP-calciner

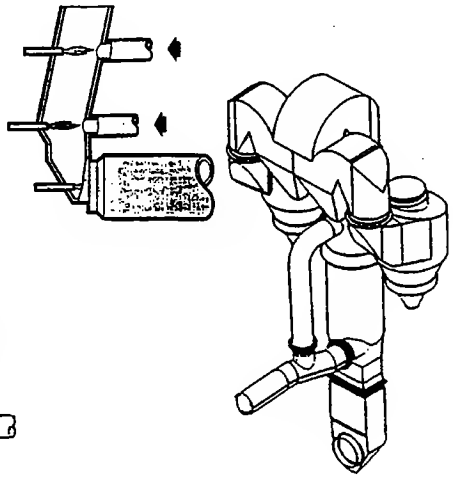
Fig. 17 Polysius



PREPOL®-AS



PREPOL®-AS-CC



PREPOL®-MSC
Multi-stage-combustion

Polysius calciners

POLYSIUS

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Fig. 18 Prerov

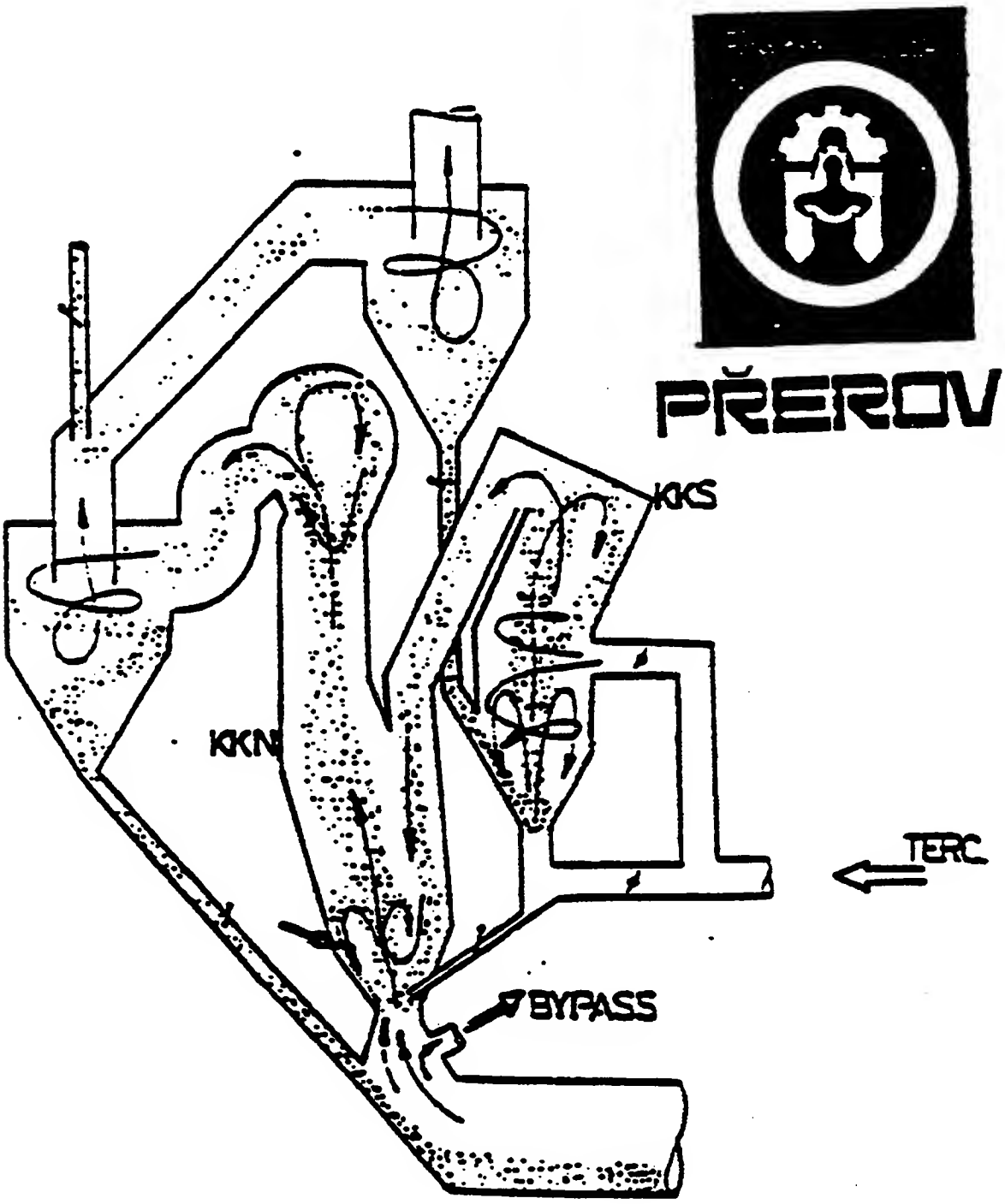

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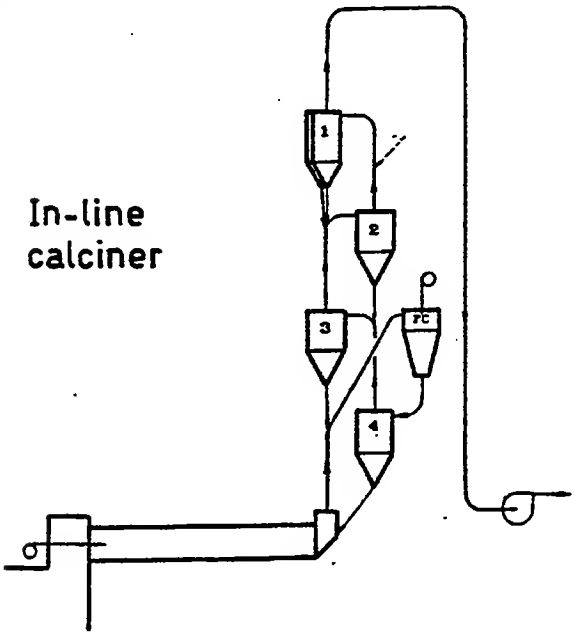
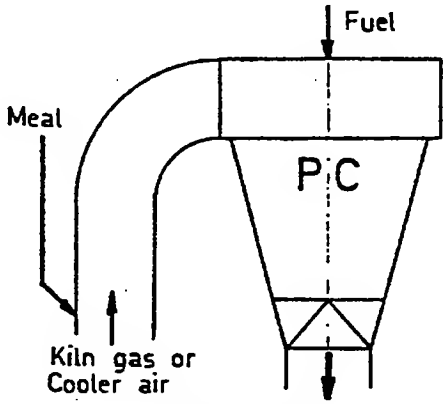
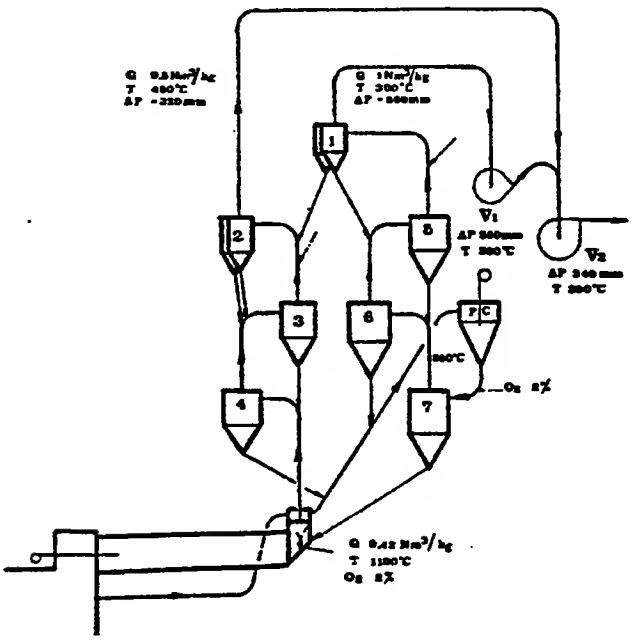
Fig. 19 EVS-PC

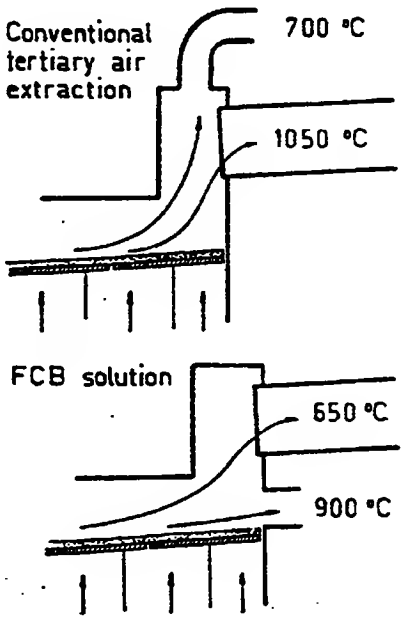




Precalciner



Separate-line calciner



Tertiary air extraction

Features of EVS-PC PC system

Supplier: Fives-Cail Babcock

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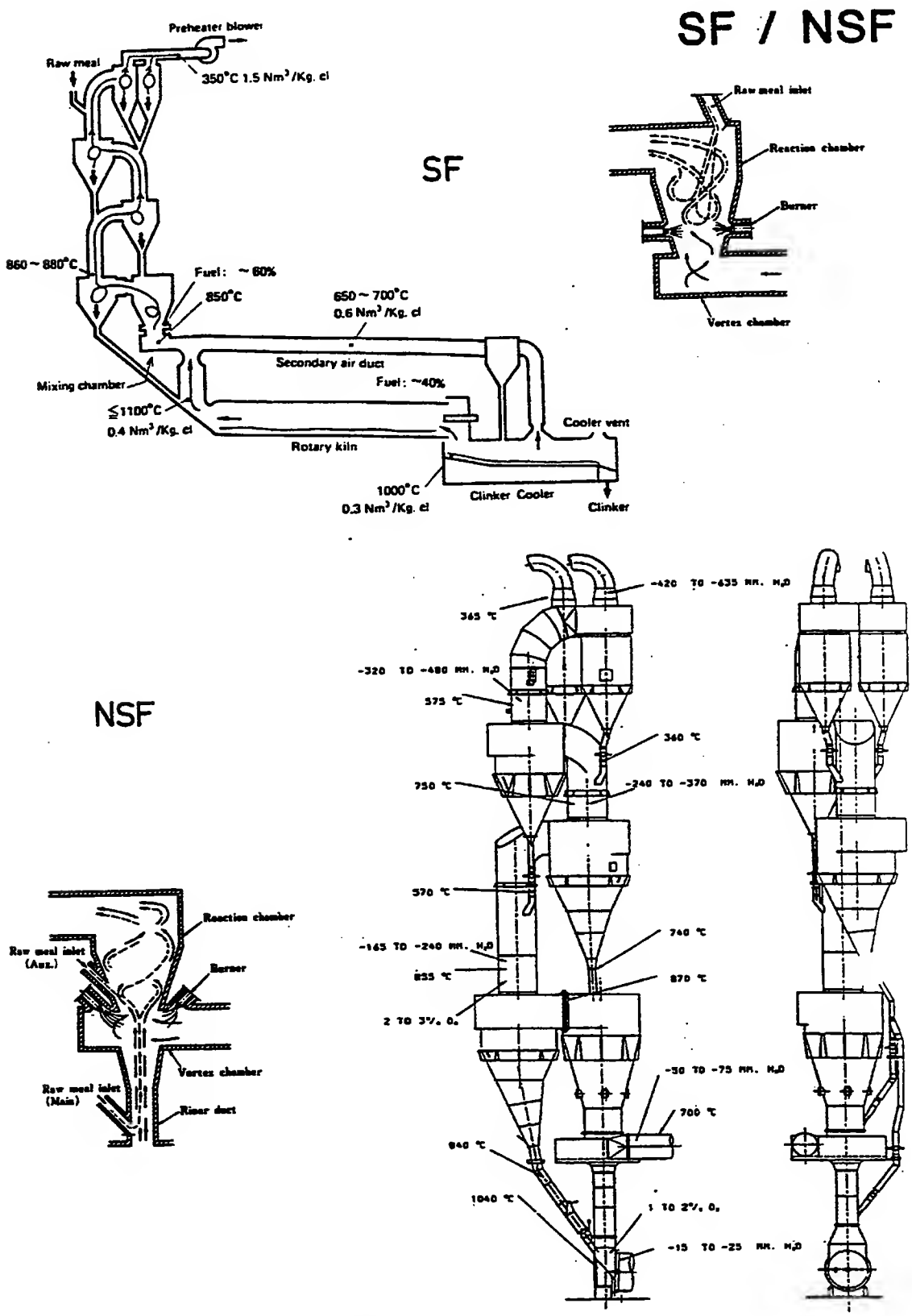
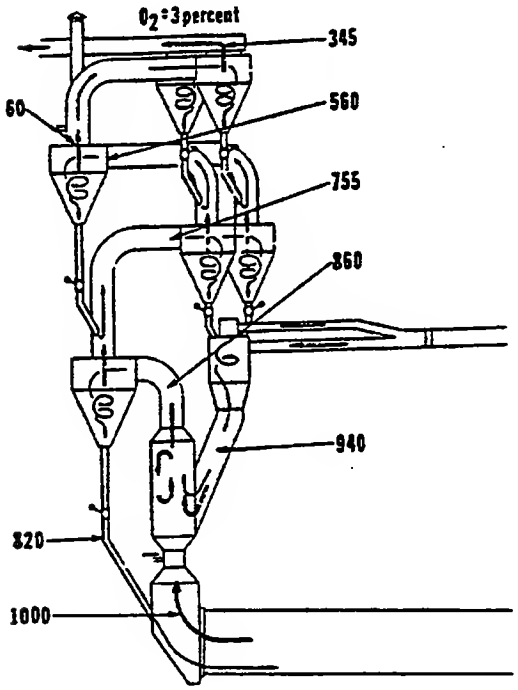
Fig. 20 SF / NSF

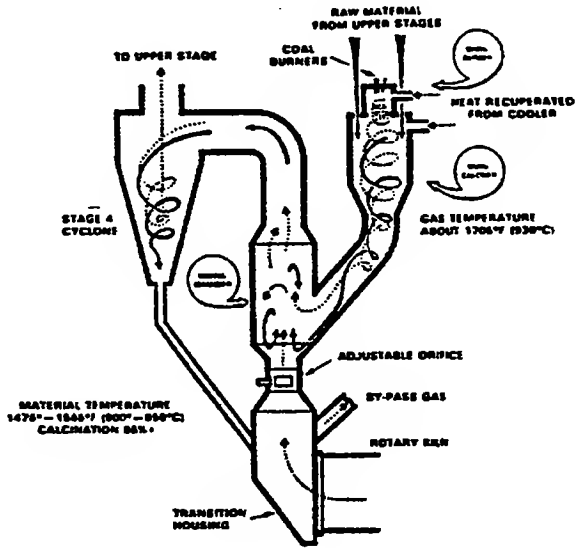


Features of SF/NSF PC system
Suppliers: Ishikawajima-Harima Heavy Industries
Fuller Company
Fives-Cail Babcock

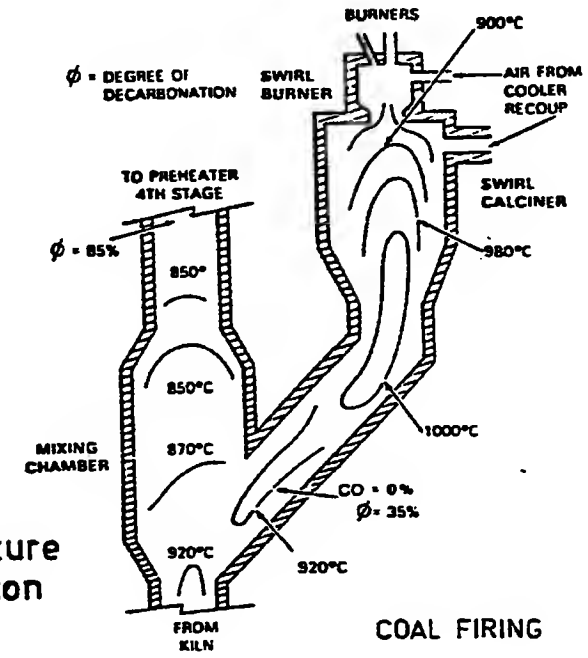
Fig. 21 RSP



Gas Temperatures °C



Precalciner



Temperature distribution

COAL FIRING

Features of RSP PC system

Suppliers: Onoda Engineering & Consulting
Kawasaki Heavy Industries
Allis Chalmers
Creusot - Loire Entreprises

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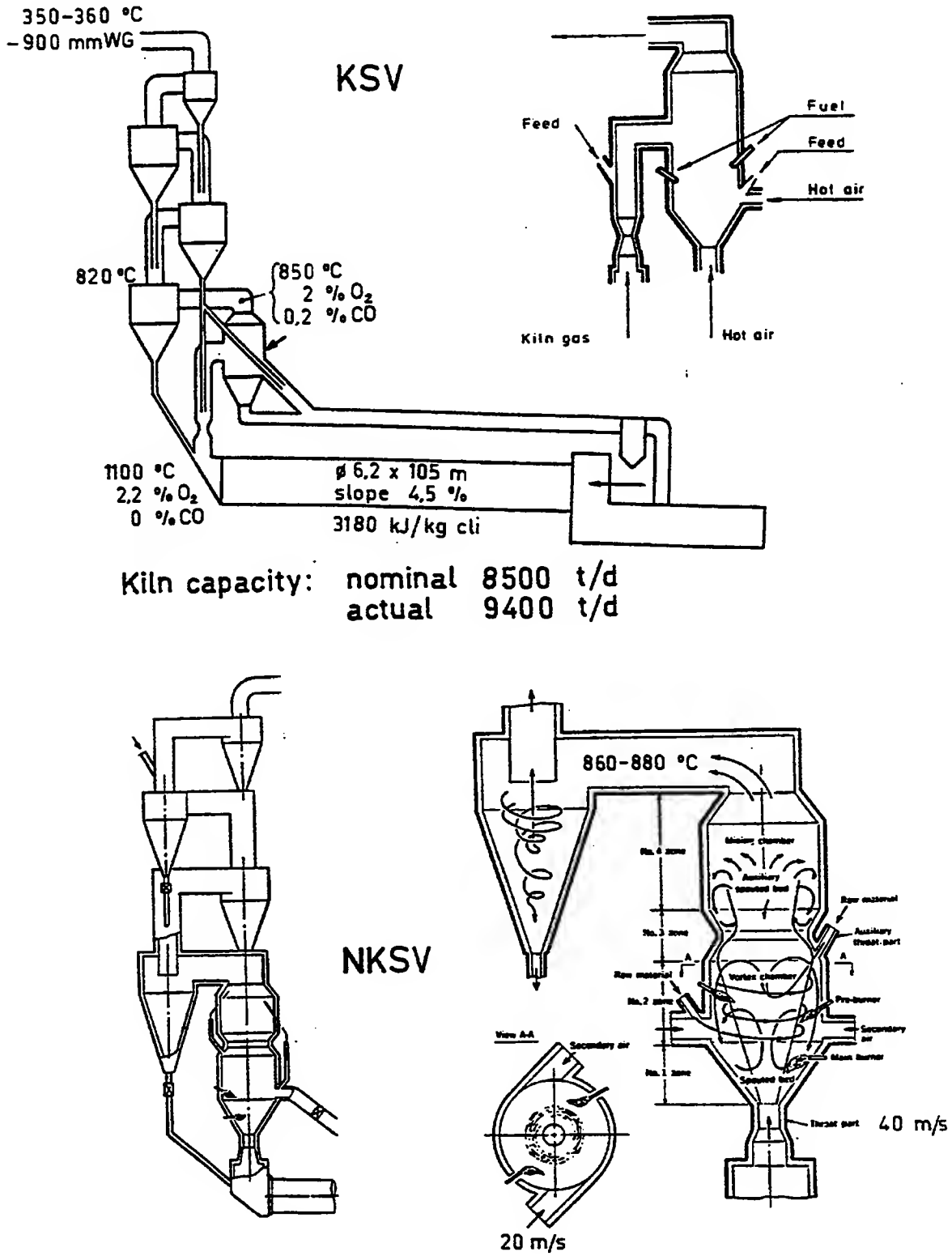
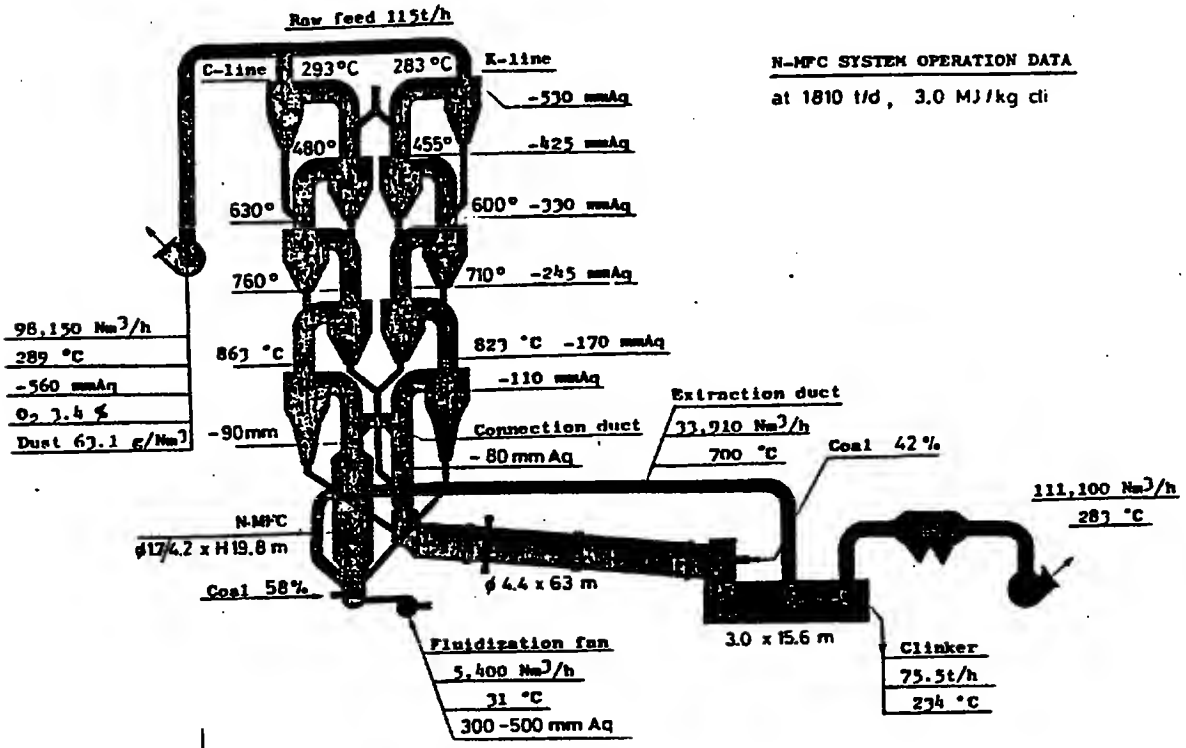
Fig. 22 KSV / NKSV

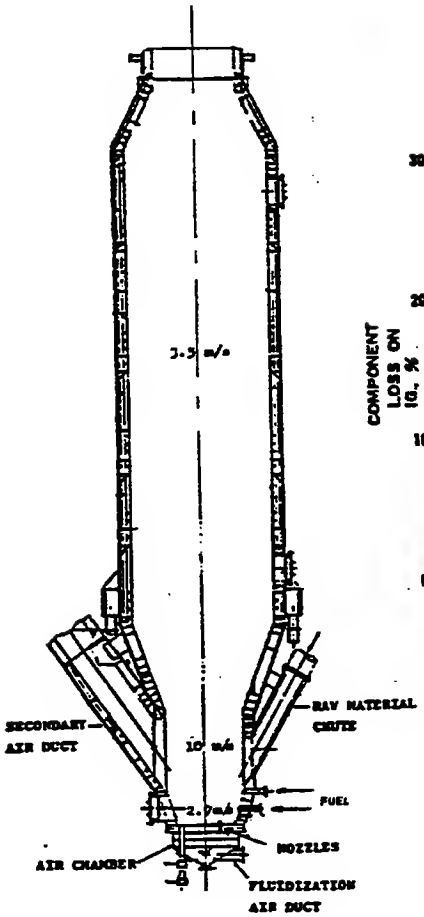


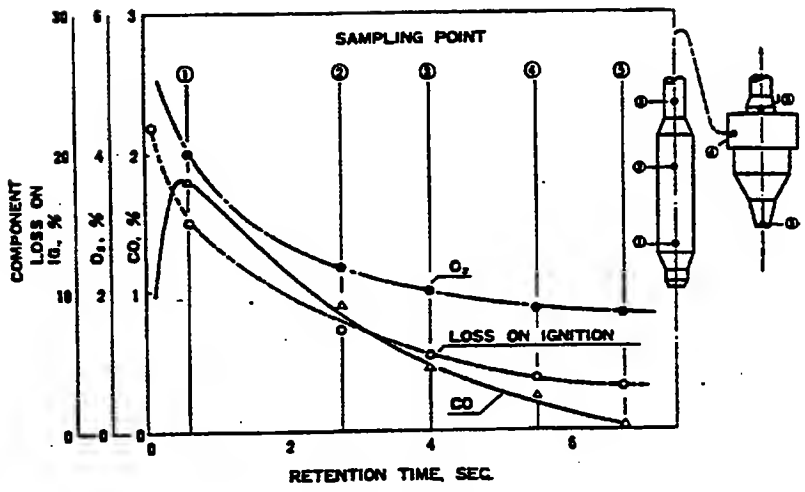
Features of KSV/NKSV PC system

Supplier: Kawasaki Heavy Industries

Fig. 23 MFC







COMBUSTION AND DECARBONATION IN MFC

Features of MFC PC System
Supplier: Mitsubishi Heavy Industries

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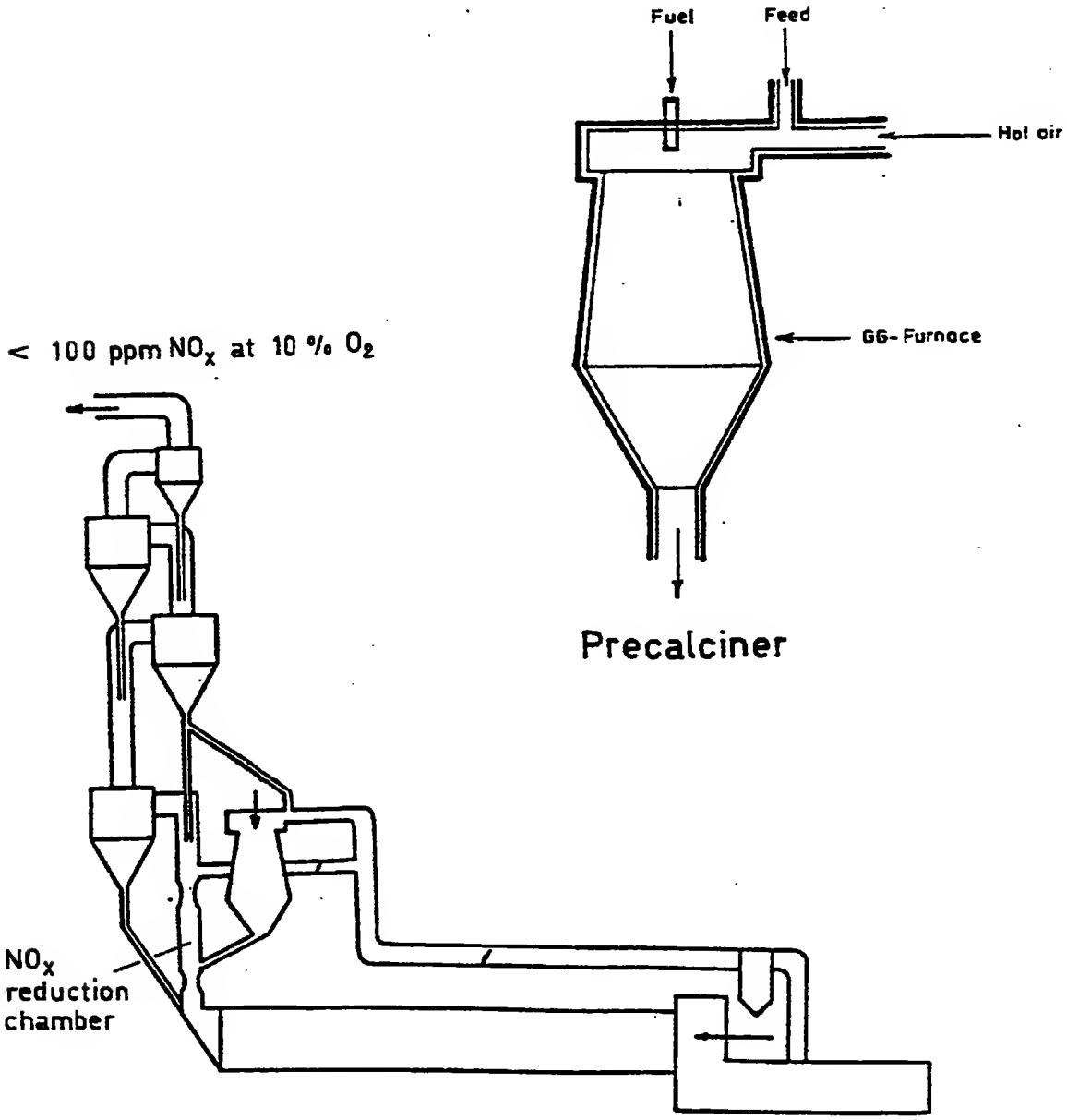
Fig. 24 GG



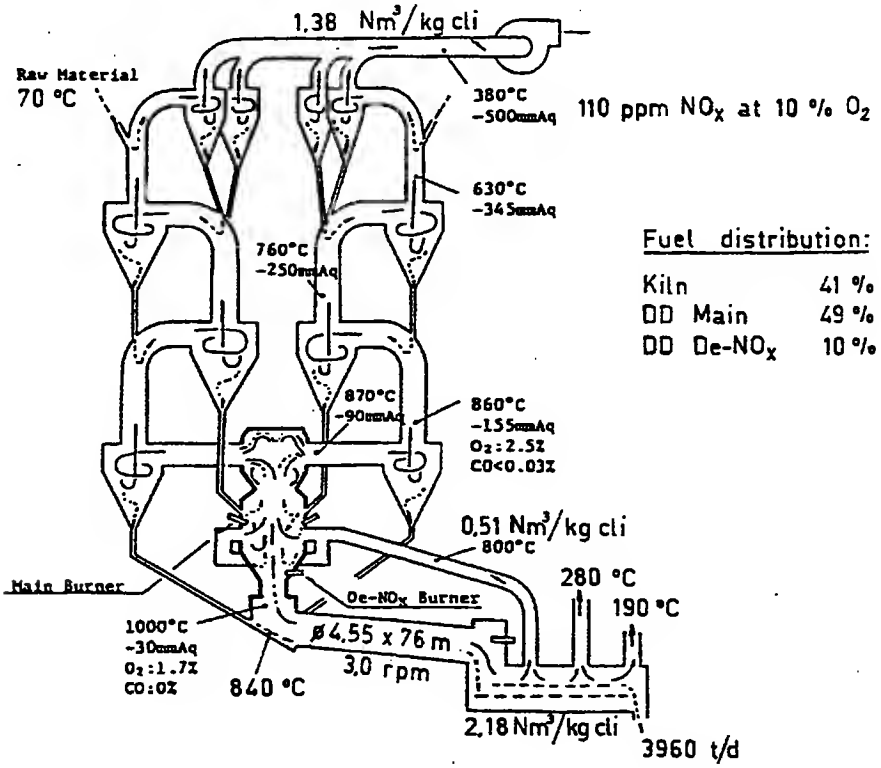
General arrangement

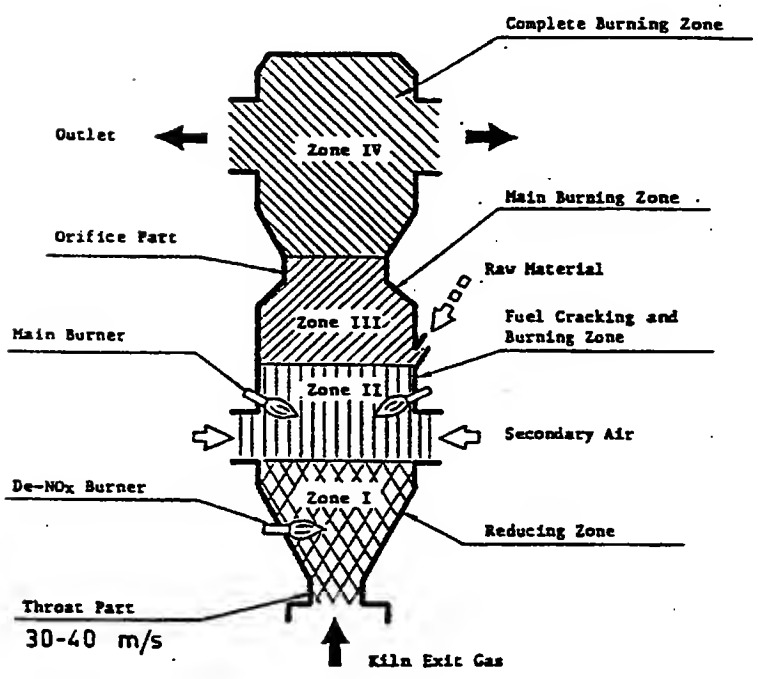
Features of GG PC system

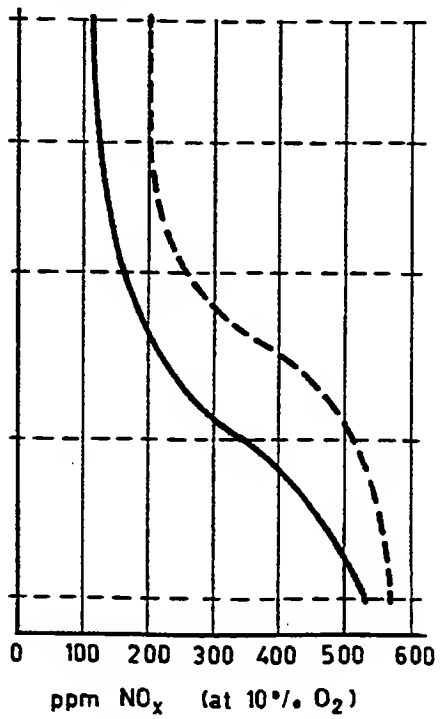
Supplier: Mitsubishi Heavy Industries

System abandoned

Fig. 25 DD







Features of DD PC System
Supplier: Kobe Steel

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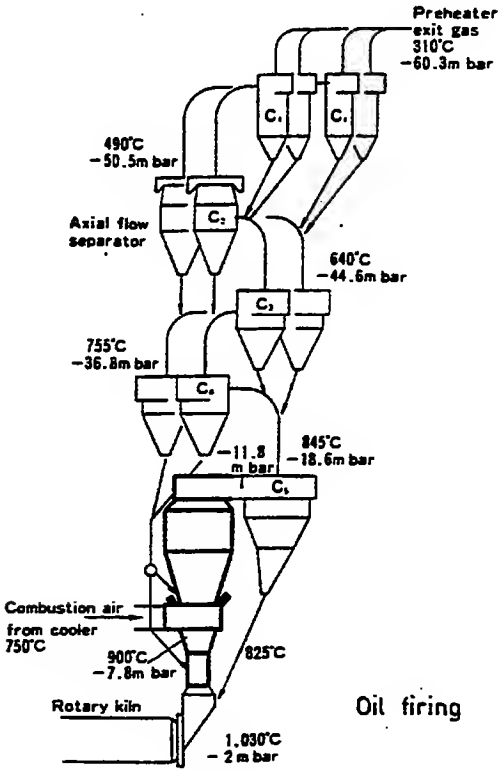
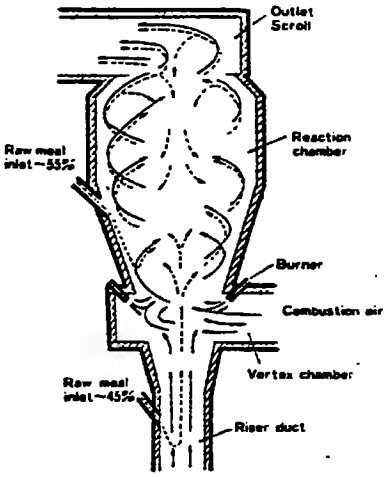
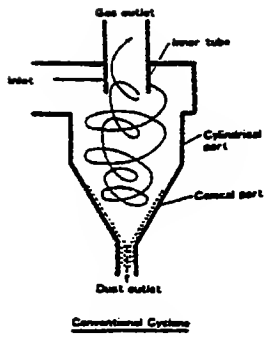
Fig. 26 CSF



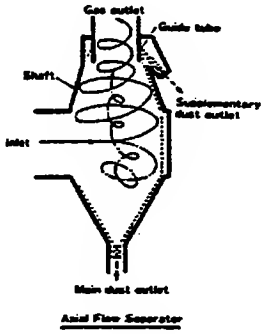
Oil firing



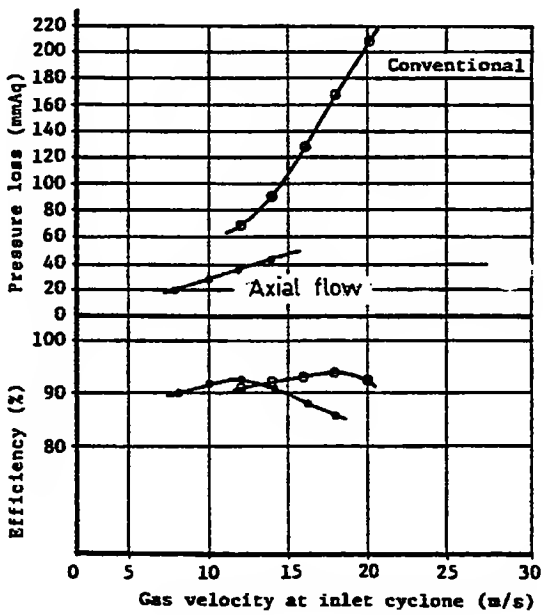
Modified NSF precalciner
(C-FF calciner)



Conventional Cyclone



Axial Flow Separator



Features of CSF PC system

Supplier: Chichibu Cement in own plants

Fig. 27 Voest Alpine PASEC System

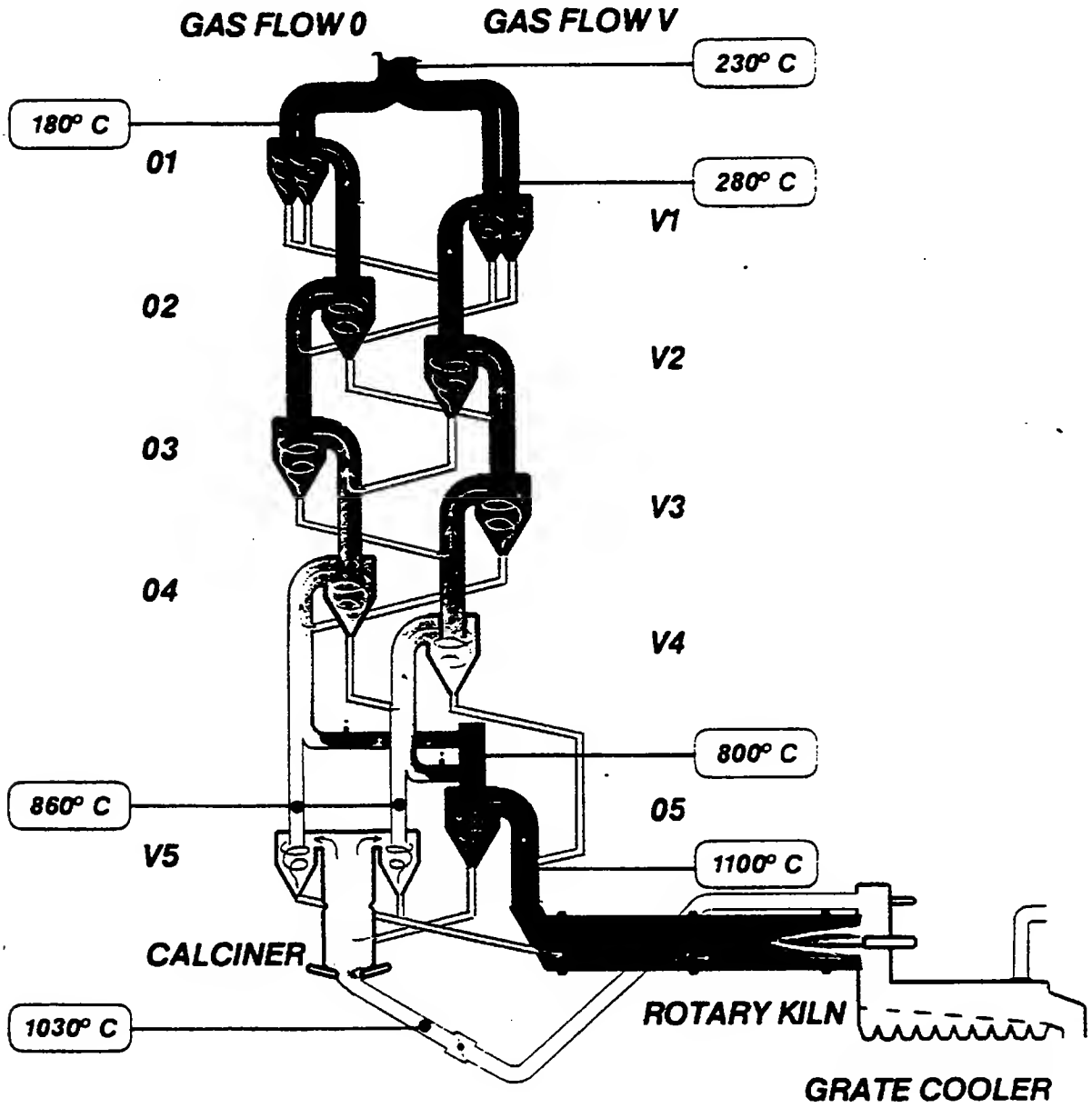


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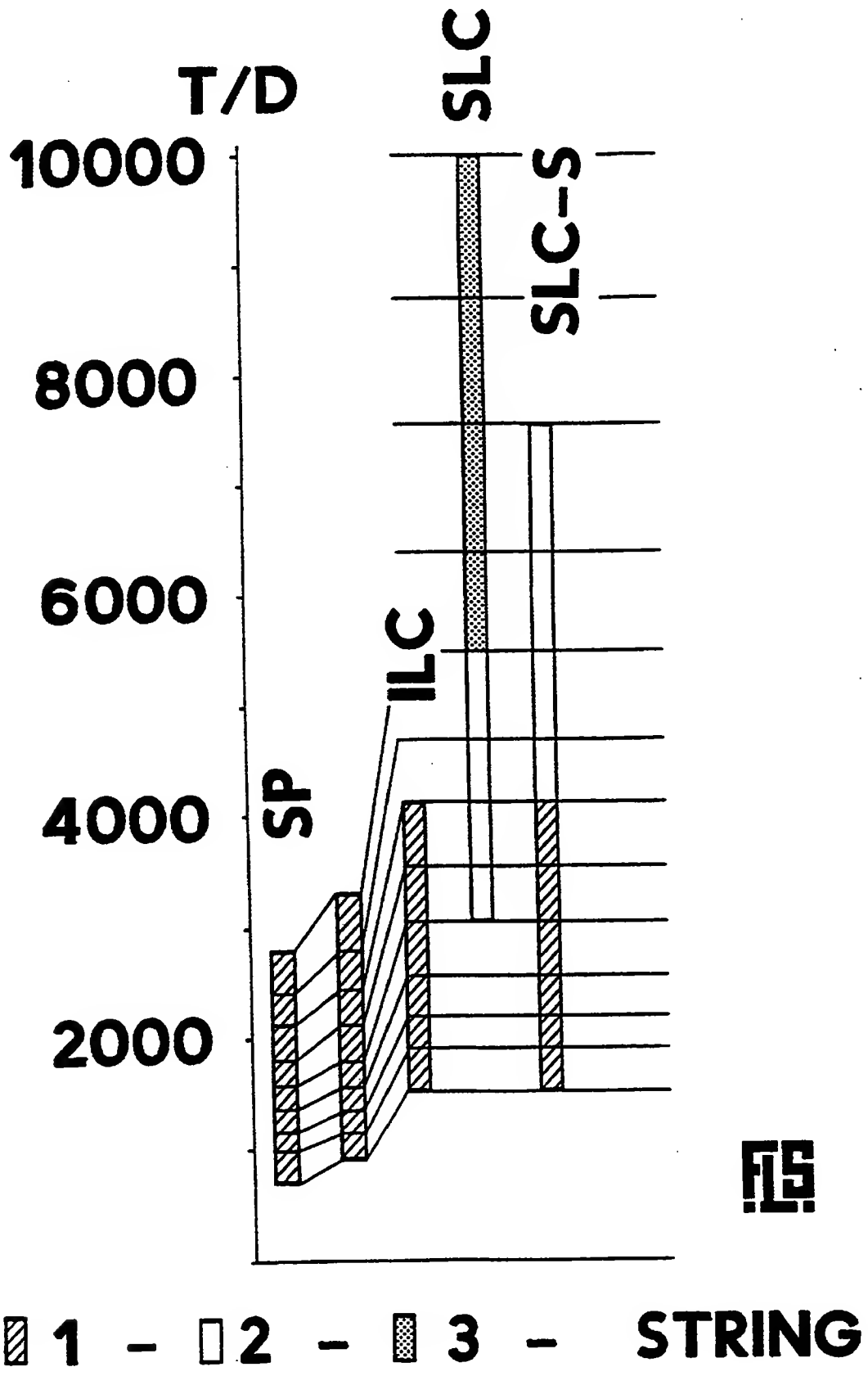
Fig. 28 FLS



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Clinker Coolers

U. Gasser / D. Brassel
PT 97/14232/E (Revision 1, February 1999)

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SUMMARY

Clinker coolers have two tasks to fulfil:

- Recuperate as much heat as possible from the hot clinker by heating up the air used for combustion
- Cool the clinker from 1400°C to temperatures adequate for the subsequent process equipment, normally to 100 - 200°C.

There are mainly two different types of clinker coolers in operation with the following features:

Grate coolers

- Crossflow heat exchange through horizontal clinker bed with cold air from below.
- Cooling airflow exceeding combustion air requirement allows low clinker temperatures, but necessitates excess (waste) air dedusting.
- Modern cooler technology with sophisticated plates and forced aeration systems allow combustion air temperatures exceeding 1000°C.
- Trend to wider and fewer grates, less cooling air and fixed inlets
- Largest units: 10'000 t/d
- Travelling grate (Recupol): last unit built around 1980

Rotating coolers

- Rotary tube coolers with separate drive or planetary cooler attached to kiln shell
- Quasi counter-current flow heat exchange
- Cooling air determined by combustion air, no waste air
- Heat exchange (recuperation) determined by condition of internal heat transfer equipment
- Limited unit size, up to 3000 t/d
- Planetary cooler not suitable for precalciner technology
- Practically no new installation built anymore

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1. INTRODUCTION

The clinker cooler is a vital part of the kiln system and has a decisive influence on the performance of the plant. Three key indicators characterize a good cooler:

- Maximum heat recuperation
- Minimum cooling air flow
- Unrestricted availability

There have been periodic changes in trends during the past decades. Grate coolers were first introduced by Fuller Company (USA) around 1930. While its design was continuously being optimized, the grate cooler became the predominant type in the 1950's. In the late 1960's, the planetary cooler gained popularity which reached its peak in the 1970's, mainly due to its simplicity. Larger unit capacities with precalciner technology made the grate cooler the preferred solution again. A wave of grate cooler reengineering starting in the mid 1980's has generated a much improved grate cooler technology as well as a new situation on the suppliers' side. New problems were experienced and have been or are being solved.

Since cement plants have life cycles of 40 years and more, numerous units of each cooler type, planetary, rotary or grate cooler of old or new designs, will remain in operation for many more years.

2. GENERAL CONSIDERATIONS

The clinker cooler has the following tasks to fulfil:

- Process internal heat recuperation by heat transfer from clinker to combustion air
- Reduce clinker temperature to facilitate clinker handling and storage
- Provide maximum cooling velocity to avoid unfavorable clinker phases and crystal size

2.1 Heat Flow in a Kiln System

The importance of the cooler as a heat recuperator can be well demonstrated with a heat flow (Sanki) diagram.

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Figure 1 **Clinker cooler and kiln system**

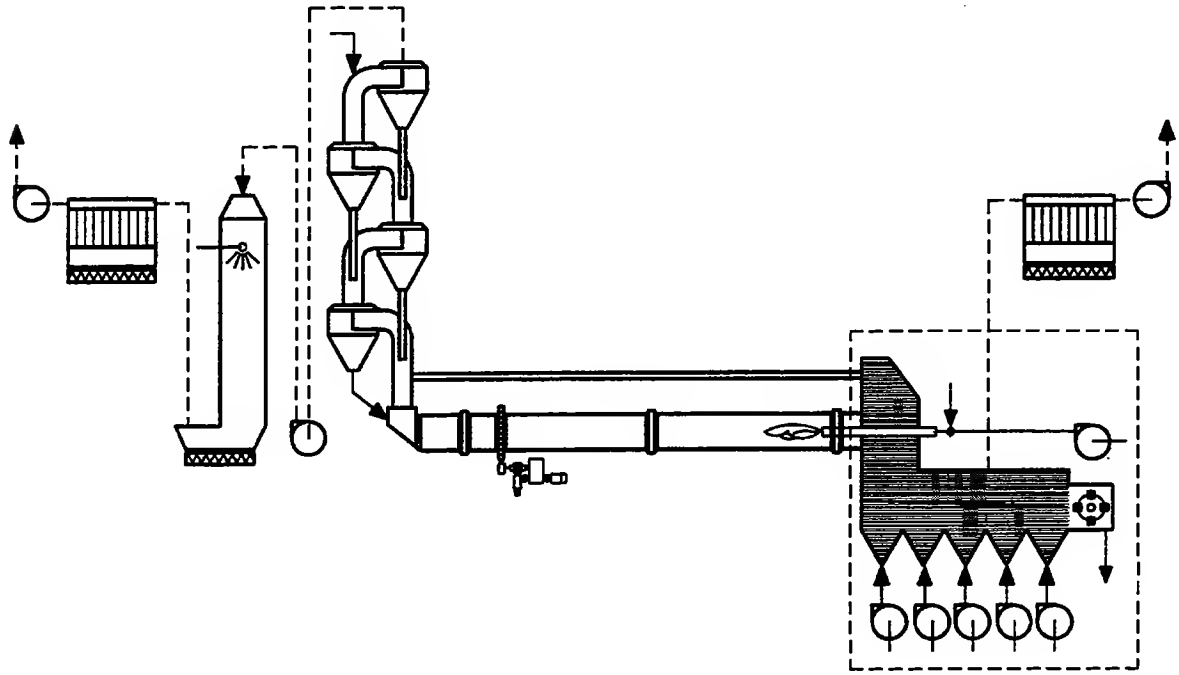
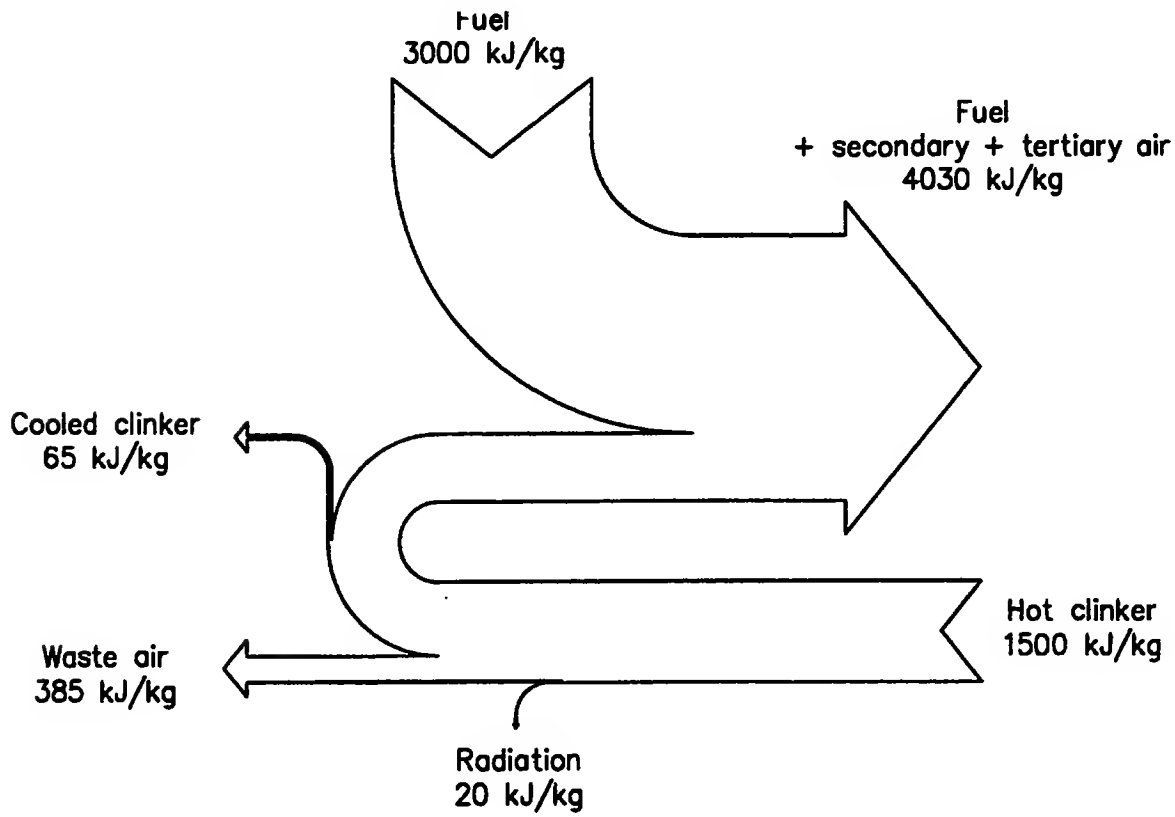


Figure 2 **Energy turnover (Grate cooler)**



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2.2 Definitions

- As for other components of the kiln system, specific figures for clinker coolers refer to **1 kg of clinker**. This eliminates the influence of plant size and allows direct comparison of clinker coolers of different types and sizes.
- **Cooling air** is the air which passes the clinker thus being heated up while cooling the clinker. It corresponds approximately to the combustion air requirement, only grate coolers allow additional air for better cooling.
- **Primary air** is the air which is required for proper functioning of the burner. Ambient air insufflated by a separate small fan plus the air from a pneumatic transport system, amounting from < 10% up to > 30% of the air required to combust that fuel. Some precalciner burners are equipped with primary air fans (for cooling) as well.
- **Secondary air** is the hot air entering the rotary kiln via clinker cooler. Its flow is determined by the combustion of the burning zone fuel. While cooling the clinker, it reaches temperatures of 600 to over 1000°C, depending on type and condition of the cooler.
- **Tertiary air** is that part of the combustion air which is required for combusting the precalciner fuel. It is extracted from kiln hood or cooler roof, and then taken along a duct (=tertiary air duct) parallel to the kiln to the precalciner. It reaches temperatures near or equal to the level of the secondary air.
- **Middle air** (grate cooler only) is extracted from the cooler roof if drying of process materials requires a temperature level which is higher than the waste air. If the quantity is small, up to 450°C can be expected at normal cooler operation.
- **Waste air** (grate cooler only) is also called cooler exit air or cooler excess air. The total cooling airflow from the fans is normally higher than the flow required for combustion (=tertiary + secondary air). The extra air, which has normally a temperature of 200 to 300°C, must be vented to ambient via a dedusting system.
- **False air** is cold air entering the system via kiln outlet seal, burner opening, casing or clinker discharge. It either dilutes secondary air thus reducing recuperated heat or adds load to the waste air system of grate coolers.
- **Specific air volumes** are airflows per kg of clinker (m^3/kg cli, Nm^3/kg cli). Independent of the kiln size, airflows of cooler systems can be directly compared.
- **Specific loads** express the relation of clinker production to a characteristic dimension of the cooler (t/d m, t/d m^2, t/d m^3). Exact definitions vary with cooler type.
- **Radiation losses** from the cooler casing/shell are particularly important for planetary coolers, where they actively support the cooling of the clinker.
- **Efficiency** expresses the quality of heat transfer from clinker to the air which is used for combustion in the burning zone and precalciner firing.

Remark: Since the heat recuperated is proportional to hot air used for combustion and temperature, an **efficiency** figure is **only meaningful** if it is related to a **heat consumption** figure (resp. a combustion airflow).

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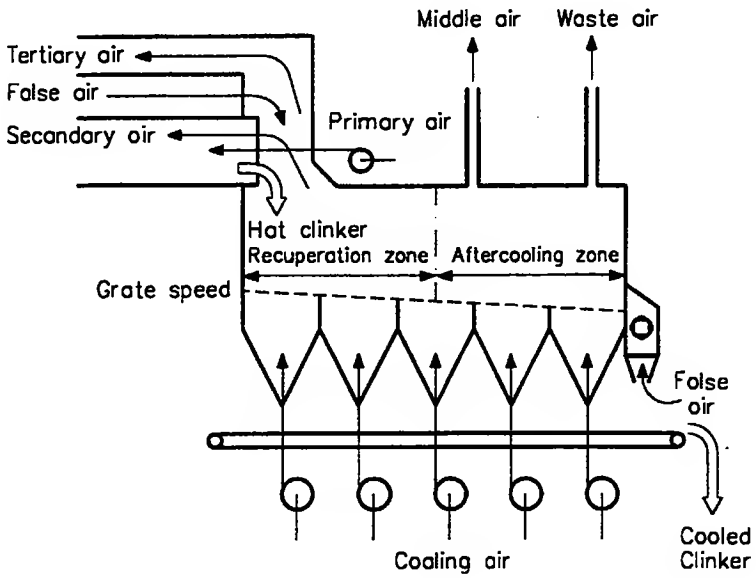
Figure 3 Clinker coolers - Definitions

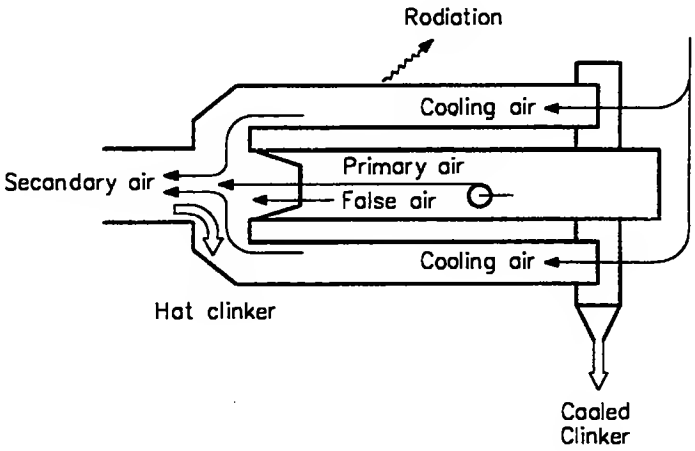
Grate Cooler

Active grate area	:	A	(m^2)
Grate length	:	L	(m)
Grate width	:	W	(m)
Grate inclination	:	Θ	(o)
Number of grates	:	n_G	(–)
Installed fan power	:	P_F	(kW)
Installed drive power	:	P_D	(kW)



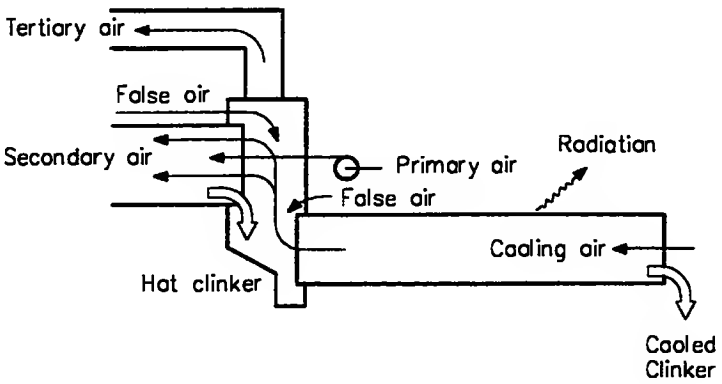
Planetary Cooler

Tube length	:	L	(m)
Tube diameter	:	D	(m)
Elbow cross section	:	A_{elbow}	(m^2)
Number of tubes	:	n_t	(–)
Cylindrical surface	:	A_t	(m^2)



Tube Cooler

Tube length	:	L	(m)
Tube diameter	:	D	(m)
Slope	:	Θ	(o)
Installed drive power	:	P_D	(kW)



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2.3 Calculations

The calculations below are examples of heat balance investigations:

• **Heat in hot clinker Q_{cli} :**	
$Q_{cli} = m_{cli} * cp_{cli} * (t_{cli} - t_{ref})$	*Example with m_{cli} =1 kg/h:* t_{cli} = 1400°C: Q_{cli} = 1 kg/h * 1.090 kJ/kg°C * (1400°C-20°C) = 1504 kJ/h
• **Heat in hot air Q_{air} :**	
$Q_{air} = V_{air} * cp_{air} * (t_{air} - t_{ref})$	*Example with V_{air} = 1Nm³/h:* t_{air} = 1066°C: Q_{air} = 1 Nm³/h * 1.421 kJ/Nm³°C * (1066°C-20°C) = 1486 kJ/h
• **Radiation loss Q_{rad} :**	
$Q_{rad} = C_R * \varepsilon * A \{(t/100)^4 - (t_o/100)^4\}$	Grate cooler Q_{rad} = 20 kJ/kg cli (from experience)
Cooler efficiency η_{cooler}	$\eta_{cooler} = \frac{Q_{\text{combustion air}}}{Q_{\text{clinker from kiln}}} = 1 - \frac{\sum Q_{loss}}{Q_{\text{clinker from kiln}}}$

The secondary (+ tertiary) air requirements are dictated by the amount of fuel fed to the burners. Per this definition, the efficiency of a cooler is getting better with increasing kiln heat consumption. It is thus obvious that a cooler efficiency figure is only meaningful if the corresponding heat consumption (or airflow) is indicated.

Example:	production	5000 t/d
	heat consumption	3000 kJ/kg cli
	secondary and tertiary air temperatures	1066°C
	Primary air main burner	10%
	PC fuel ratio	60%
	False air and excess air neglected (not realistic!)	

$Q_{comb\ air}$:

$V_{Comb\ air}$ = 3000 MJ/kg cli * 0.26 Nm³/MJ * 5000/24*10³ kg/h * (1-0.4*0.1)
= 156'000 Nm³/h

$t_{comb\ air}$ = 1066°C → $q_{combustion\ air}$ = 1.421 kJ/Nm³° * (1066-20)° = 1486 kJ/Nm³

$Q_{comb\ air}$ = $V_{comb\ air}$ * $q_{comb\ air}$ = 1486*156'000 kJ/h = 231'816 GJ/h

$Q_{clinker}$:

$m_{clinker}$ = 5000 t/d /24 h/d *10³ kg/t = 208'333 kg/h

$t_{clinker\ from\ kiln}$ = 1400°C → $q_{clinker\ from\ kiln}$ = 1.09 kJ/kg° * (1400-20)° = 1504 kJ/kg

$Q_{clinker}$ = 208'333 * 1504 kJ/kg = 313'333 GJ/h

Efficiency η = 231'816 / 313'333 * 100% = 74.0%

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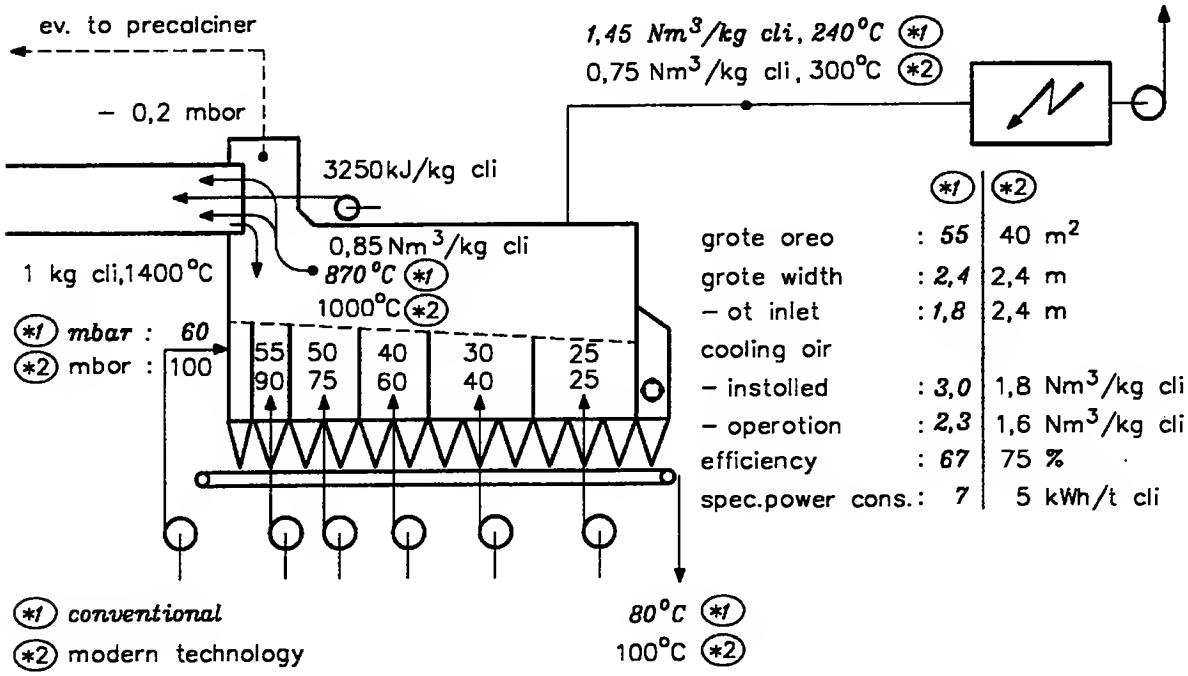
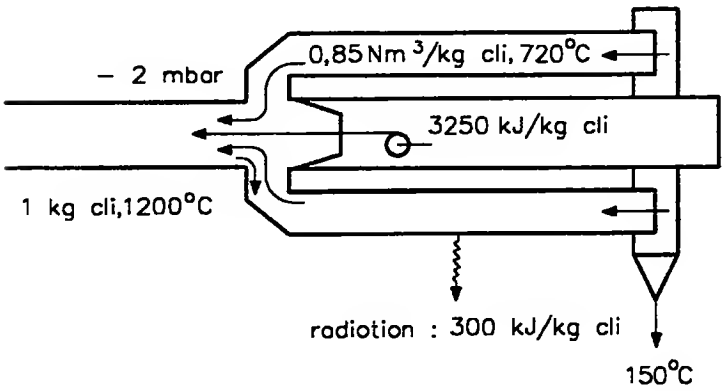
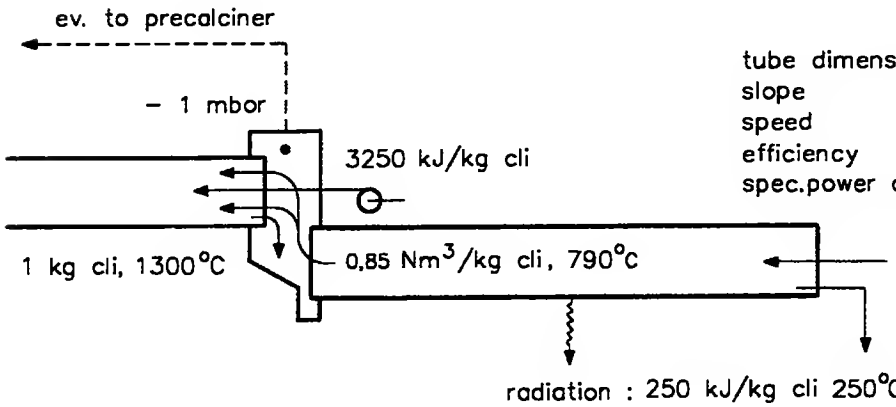
Figure 4 Clinker cooler typical data (4-stage SP Kiln, 2'000 t/d)



	(*1)	(*2)
grote oreo	: 55	40 m²
grote width	: 2,4	2,4 m
– ot inlet	: 1,8	2,4 m
cooling oir		
– instolled	: 3,0	1,8 Nm³/kg cli
– operotion	: 2,3	1,6 Nm³/kg cli
efficiency	: 67	75 %
spec.power cons.	: 7	5 kWh/t cli



number of tubes : 9
tubes dimensions : ⌀ 2,1 x 20 m
efficiency : 67 %
spec.power cons. : 1,5 kWh/t cli



tube dimensions : ⌀ 5 x 60 m
slope : 4,5 %
speed : 2,5 – 3,5 rpm
efficiency : 67 %
spec.power cons. : 4 kWh/t cli

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3. GRATE COOLERS

3.1 The Reciprocating Grate Cooler

The reciprocating grate cooler is the most widely applied type and is exclusively used for new plants.

3.1.1 Principle

- The following major **system components** can be distinguished:
  - Casing with kiln hood and connections for air at different temperature levels
  - Reciprocating grate with drive system
  - Aeration system with fans, undergrate compartments and direct air ducts
  - Riddling (= fall through) extraction system with hoppers, gates and transport
  - Clinker crusher
- **Material transport**
  The clinker is pushed by the vertical part of the front edge of the preceding plate. The entire grate consists of a combination of fixed and moving rows which results in a quasi-continuous motion of the clinker bed.
- **Heat exchange**
  Heat exchange from clinker to air is according to the **cross current** principle. The cooling air penetrates the clinker bed which is laying on the grate from underneath and leaves it at the surface. While passing through the hot clinker, the air is accumulating heat which is transferred from the clinker.
- **Cooling air**
  Normally, ambient air is blown to underneath of the grate plates loaded with clinker by a number of cooling air fans. Delivery pressure must be sufficient to penetrate the clinker bed and to compensate for the expansion (increase of actual volume) of the air from heating it up
  Under ideal conditions, the required cooling air depends directly from the desired clinker temperature. One part of the cooling air is used for combustion in the kiln, the rest is cleaned and vented to ambient, unless it is further used, e.g. for drying.
- **Cooling curve**
  A simplified mathematical model for clinker cooling in a conventional, optimized grate cooler gives the relation between cooling air quantity and clinker temperature as follows:

$$\frac{T_{cli} - T_{amb}}{T_{cli\,in} - T_{amb}} = \exp[-(V_{air} / 0.77)]$$

with $T_{cli\,in}$ = clinker temperature at cooler inlet °C
T_{amb} = ambient temperature °C
V_{air} = cooling air quantity Nm^3/kg cli

The above approximation (curve Fig. 17: $T_{cli} = 1400°C$) has been found to give satisfactory results for conventional grate coolers from various suppliers.

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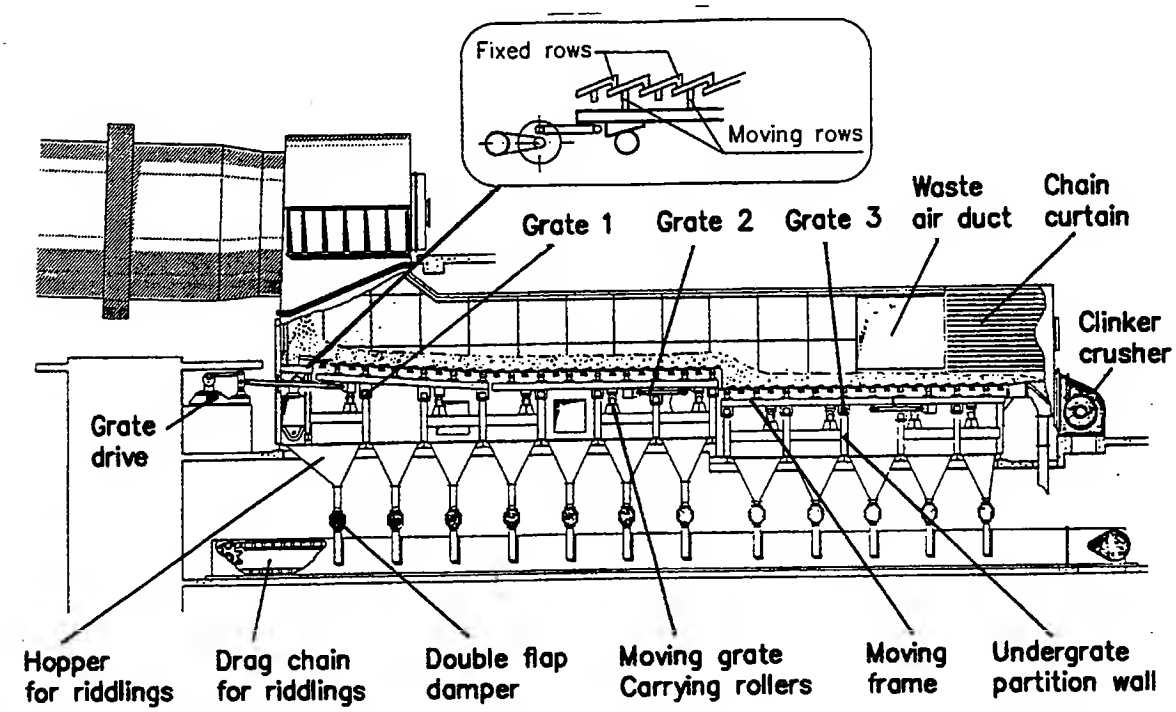
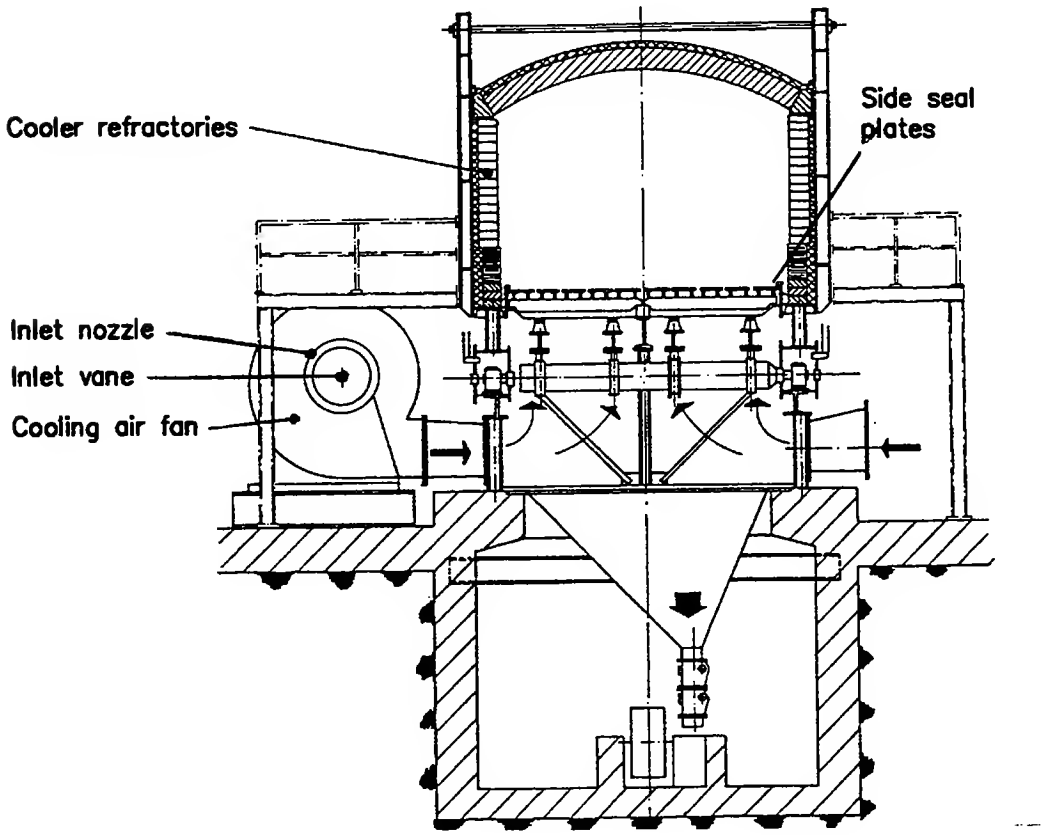
Figure 5 Reciprocating Grate Cooler: Design Features





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3.1.2 History

It was the Fuller Company (USA) who introduced the first reciprocating grate cooler in the late 1930's with a grate slope of 15°.

Fluidized material running down the grate leads to 10° grate inclination. The 10° cooler was predominantly used until the mid 1950's. Problems were encountered with those 10° coolers when the clinker was fine and started to fluidize. As an attempt to solve this problem, wedge grate plates were used. Another drawback of those 10° coolers was the building height required for larger units.

In the mid 1950's, the first horizontal grate coolers were introduced. They were initially just 10° grates installed horizontally with accordingly reduced conveying capacity. Some of these coolers were severely damaged by overheating, due to fluidization and accumulation of hot fine clinker at the feed end.

This drawback of the horizontal cooler lead to the development of the so-called **combi cooler.** Is has one (or formerly two) inclined grates with normally 3° slope, followed by one or two horizontal grates. Not all suppliers followed the same philosophies, so all three concepts (all horizontal, combi and all inclined) can be found all over the world.

The planetary cooler boom period in the 1970's came to an end, when large production capacities were in demand. Precalciner technology required grate coolers which eventually needed to be reengineered again. Problems related to the clinker distribution, growing awareness of heat and power consumption as well as the demand for higher availability forced the suppliers to introduce new solutions. Initiated by the new company IKN, the grate cooler technology underwent significant changes since the mid 1980's. **Modern grate plates**, **forced (direct) aeration** and better **gap design** were introduced by all cooler makers helping to reduce cooling airflow and cooler size.

The new approach lead to better recuperation in most cases. However, serious **wear problems** with the new systems forced most of the companies to modify their solutions once again. Today, in the mid 1990's, we are still gaining experience with latest designs.

The ultimate solution would be the **waste air free** grate cooler with unlimited flexibility and availability. However, right now the cement industry would be happy with smooth operation, high recuperation, low cooling air and no cooler related kiln stops.

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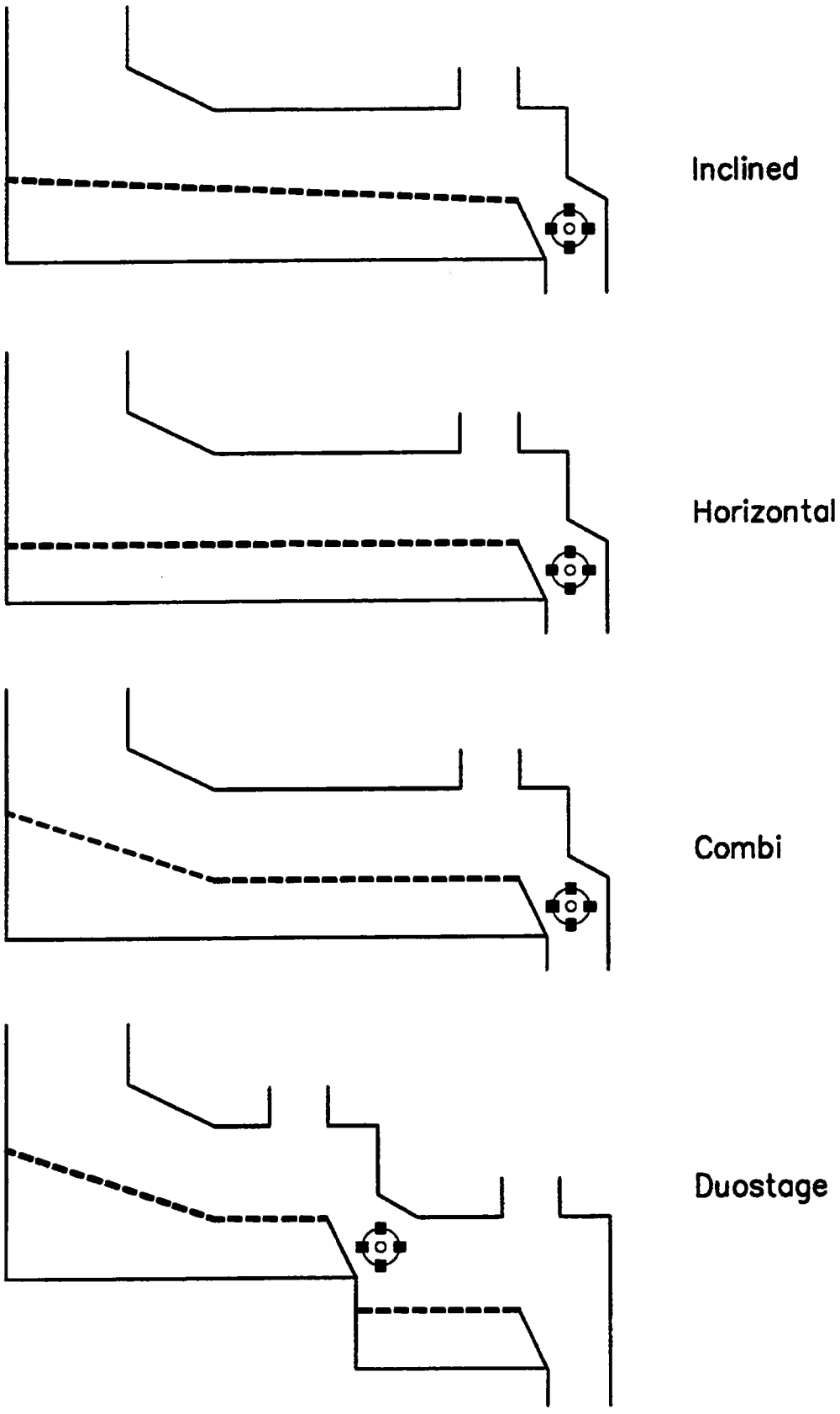
Figure 6 Various configurations of reciprocating grate coolers



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3.1.3 Conventional Grate Coolers (1980's)

3.1.3.1 Typical Design Features

- Grate plates with round holes
- Two to three grates, depending on size
- Grate slope 0° or 3° or both, depending on supplier
- Mechanical excenter drives for reciprocating grate
- Chamber aeration
- Fan pressure 45 mbar (first) to 25 mbar (last)
- Smaller compartments at inlet, larger towards outlet
- Clinker riddling extraction with hoppers, gates and dragchain (some earlier designs: internal drag chain without hoppers)
- Hammer crusher at cooler discharge

World's largest kilns (10'000 t/d in Thailand) are equipped with conventional grate coolers from CPAG with 4 grates.

3.1.3.2 Strengths and Weaknesses of Conventional Grate Coolers

Strenghts	Weaknesses
• Lower clinker end temperature due to higher amount of cooling air • Possibility of adjusting cooling air and grate speed provides higher flexibility • Optimization possibilities during operation	• Waste air handling system (dedusting, fan) required • More complex cooler requires higher capital investment • Higher power consumption than planetary or tube cooler • Uneven clinker discharge / segregation leads to several problems • Red river • Snowmen • Air breakthrough (bubbling, geyser) • Reduced plate life • Excessive clinker fall through between gaps

Causes and mechanism of those problems are further explained in the next paragraph.

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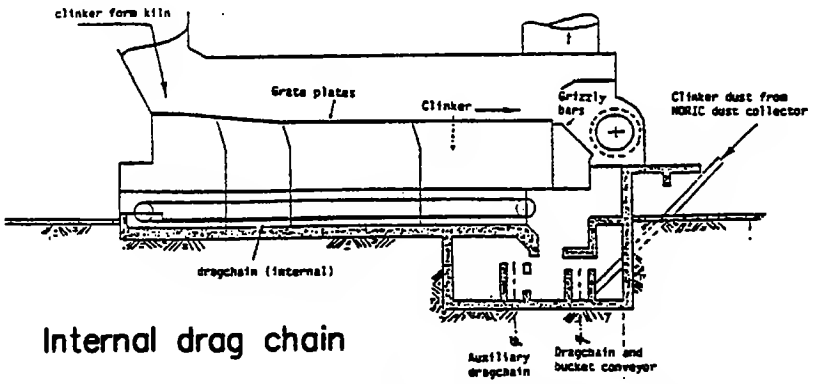
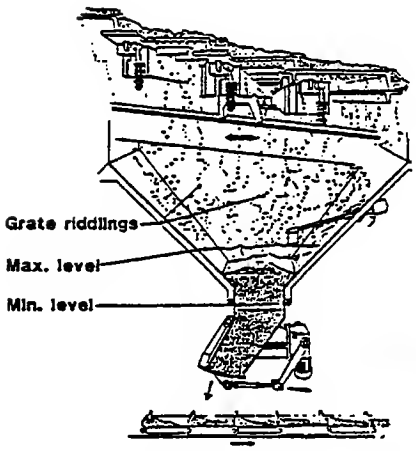
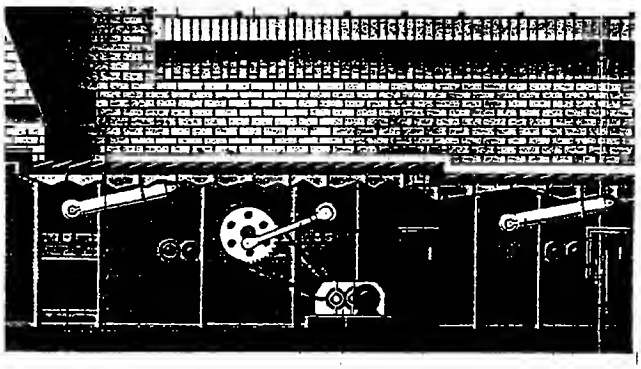
Figure 7 Conventional grate coolers: Design features



Internal drag chain



Knife gate (CPAG)



Grate drive with shock absorbers to center the grate (KHD)

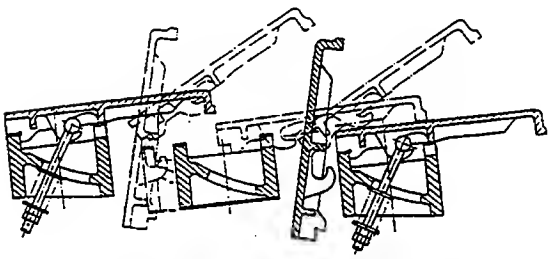


Plate installation (Polysius)

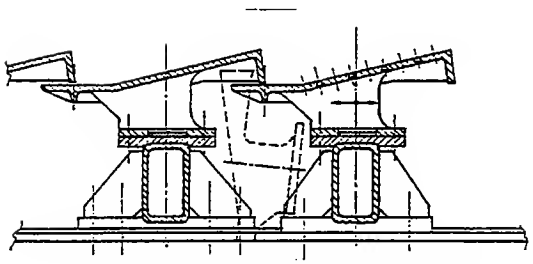


Plate installation (FLS)

3.1.4 Typical Grate Cooler Problems

Most grate coolers show a tendency to one or more of the system inherent problems, and in many cases there is no real cure. Investigations of the causes lead to the development of the modern cooler technology.

- **Segregation**:
  Due to its physical properties, the clinker is lifted by the kiln rotation before it is discharged into the cooler. Installation of the grate axis offset from the cooler axis should compensate for this effect. However, since discharge behavior of finer and coarser clinker particles differ from each other, the clinker fractions are not evenly distributed across the grate. Fines are discharged later and are thus found predominantly on the rising side of the kiln shell (Fig. 8a).

- **Thin clinker bed in recuperation zone**:
  With a conventional grate cooler with chamber aeration, the clinker bed thickness is limited directly by the installed cooling fan pressure and indirectly by the quality of compartment seals and distribution of the clinker across the width. In order to avoid overheated plates, the operator will set the bed not higher than allowed to guarantee airflow through the plate carrying the clinker with the highest bed resistance.
  Thin bed operation leads to unfavorably high air to clinker ratio and poor heat exchange on the sides with consequently low recuperation efficiency.

- **Red river**:
  The infamous red river is one of the most feared problems with grate coolers.
  Due to segregation, fine clinker has always its preferred side (see above).
  Different bed resistance on either side and only one air chamber across the entire width often cause fluidization of the fine clinker laying on top. This fluidized clinker does no longer follow the speed of the grate, but shoots much faster towards the cooler discharge end. Because the residence time of that fine clinker is much reduced, it does not follow the general cooling curve and forms a red hot layer on top of the regularly cooled, already black clinker. Hence the term "red river".
  It is not the missed heat recuperation, but the red hot material being in touch with cooler walls, plates and side seals in the colder area where such temperatures should normally not occur. Premature destruction of those pieces results in poor availability, high maintenance and ultimately in loss of production and sales revenues.

- **Snowman**:
  The sticky consistence of the hot clinker leaving the kiln combined with the compaction at the drop point often leads to formation of solid clinker mountains on the grate. Not permeable for cooling air, they grow larger and disturb the flow pattern of the clinker in this anyway critical inlet area.

- **Air breaking through**:
  Due to the different resistance of the clinker bed and the fear of overheated plates, too much air is put on the first grate compared to the clinker bed. The result is air shooting through the bed, hardly taking any heat and thus not contributing to the heat exchange. In addition to that, the clinker is mixed which can be seen by the bubbling action, and the layered clinker bed (colder clinker below, hotter on top) is destroyed thus disturbing the cross flow heat exchange pattern.
  The results are low recuperation and too much heat going to the aftercooling zone.

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Figure 8a: Segregation at cooler inlet

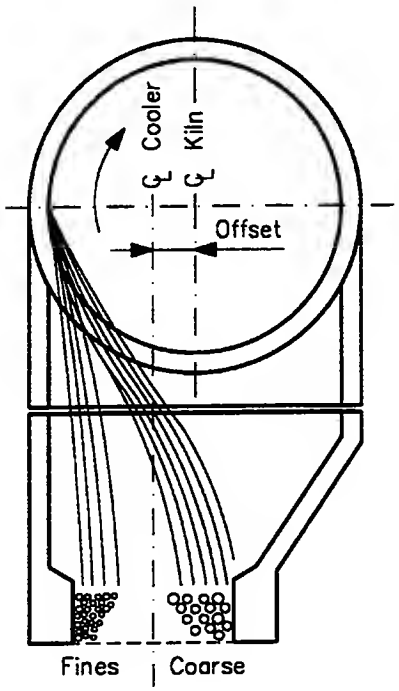


Figure 8b: Clinker bed depth effect on cooling

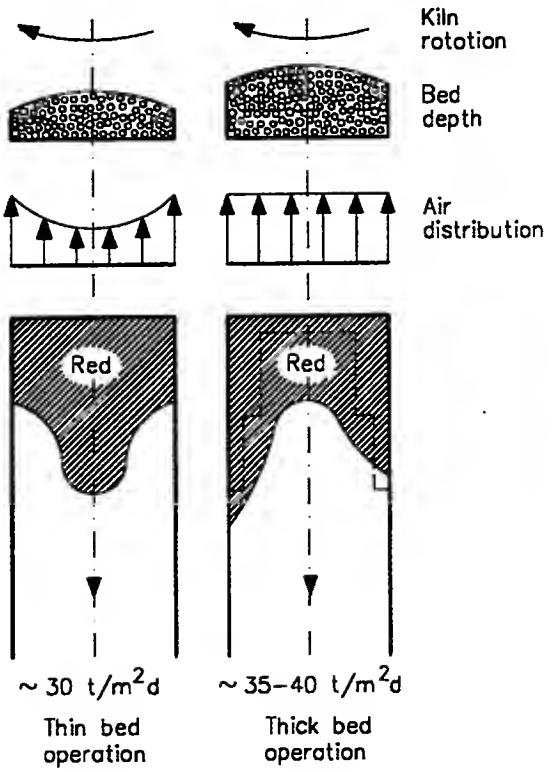


Figure 8c Red River

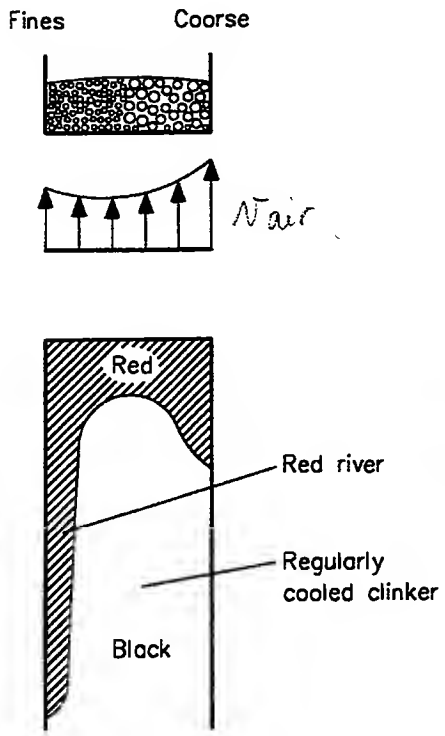
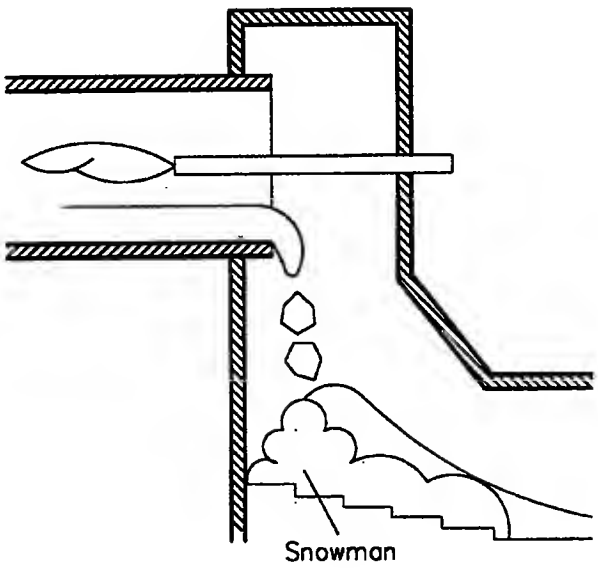


Figure 8d Snowman



3.1.5 Modern Grate Coolers (1990's)

3.1.5.1 Design Features

The successful clinker cooler has:

① Correct allocation of cooling air to clinker
② Sustainable gap widths in the entire cooler

All new or redesigned clinker coolers are aiming at the above two goals:

- Modern grate plates, designed to cope with high temperature differences
- Inclined inlet section without moving rows
- Pattern of zones for individually adjustable aeration in recuperation zone
- Modern plates for a tight grate in the after cooling zone
- New, improved side seal plate design for tight gaps and low wear
- Careful undergrate compartment sealing
- Adequate seal air system with correct control
- Wider and shorter coolers; lower number of grates
- Improved and wear protected moving grate support and guidance
- Hydraulic grate drive with optimized control system
- Cooling air fans with inlet vane control and inlet nozzle for measuring flow
- Roller crusher

3.1.5.2 Strengths and Weaknesses of Modern Grate Coolers

Strenghts	Weaknesses
• More constant heat recuperation → improved, smoother kiln operation • Cooler inlet: improved clinker distribution across grate width • Elimination / control of red river • Significantly reduced grate riddlings (clinker fall through) • Higher waste air temperature (valuable for drying) • Lower heat consumption due to higher heat recuperation (cooler efficiency) • Reduced power consumption due to less waste air • Lower civil cost due to more compact cooler • Lower investment due to smaller waste air system • Reduced cost for maintenance	• More complicated mechanical installation (varies with supplier) • Higher secondary air temp. increases wear of nose ring and burner refractories • Higher actual (m^3/h) tertiary air flow can increase dust entertainment at take off point • Teething problems with new designs -> design changes still in progress

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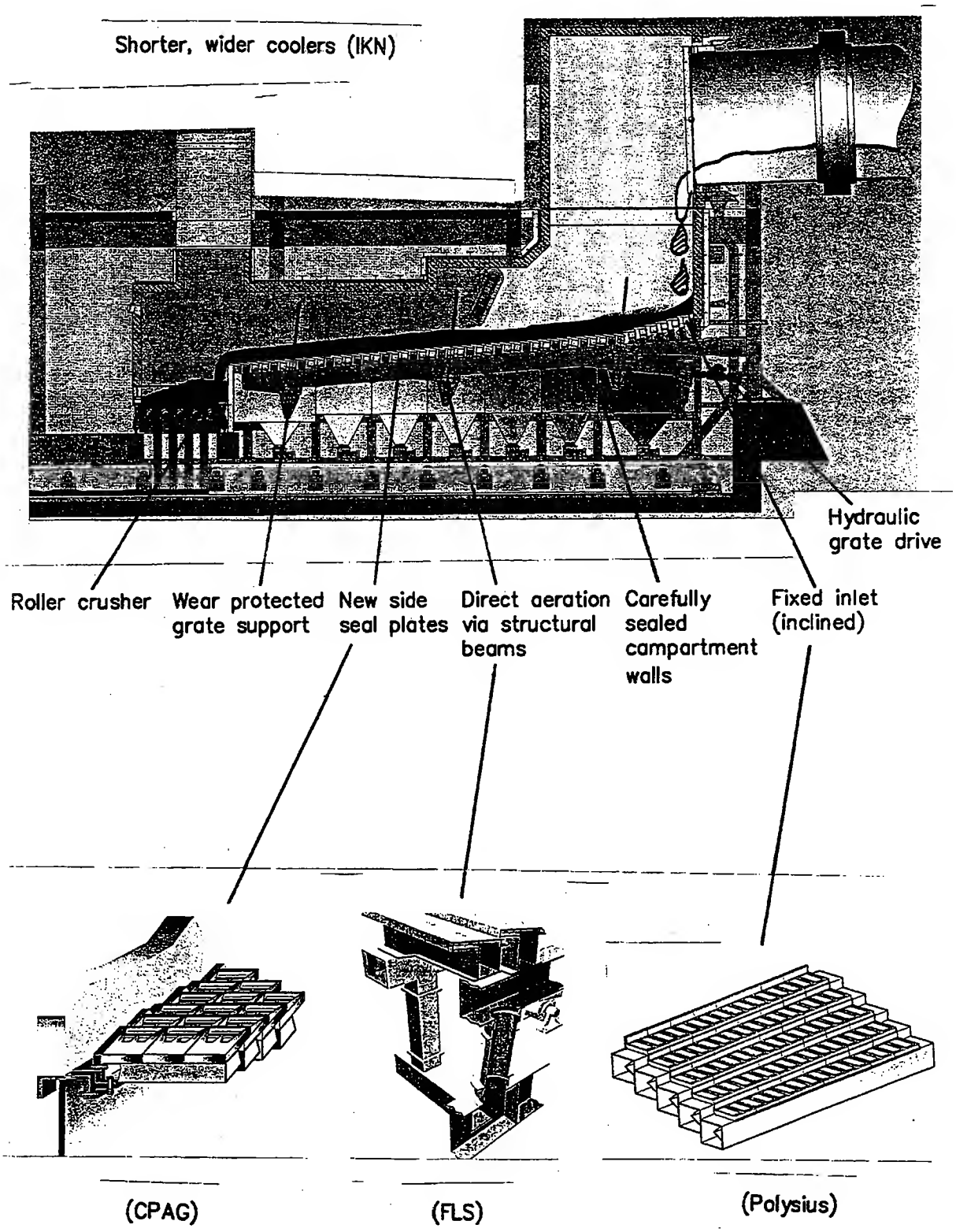
Figure 9 Modern Grate Coolers: Design features



3.1.6 Design Highlights of Modern Grate Coolers

3.1.6.1 Modern Grate Plates

In the mid 1980's, the first modern grate plates were installed in grate coolers by IKN and CPAG. They were designed for the following targets:

- Allow for lower air/clinker ratio in the recuperation zone for higher recuperation
- Improve clinker distribution across the grate width
- Assure that all grate plates are always sufficiently cooled by air

The above targets were reached using the following ideas:

- **Higher built-in pressure drop**
  Similar to the effect of thick bed operation, a higher pressure drop across the plate reduces the relative influence of variations in permeability of the clinker bed.
- **No more fine clinker falling through**
  Fine clinker falling through means loss of heat and thermal stress on the drag chain. For forced aeration (below) it is mandatory that no material can fall in the air ducts where it would cut off the air supply.
- **Forced (direct) aeration via air ducts**
  In order to ensure that all plates get enough air, to allow individual allocation of air to different areas and to avoid that air escapes through gaps, groups of plates are supplied with air directly via a special duct system
- **Tight gaps between plates and plates/casing**
  Not only through the grate surface, but also through gaps between plates within the same row as well as from one row to the next, fine clinker can fall through. Those gaps have to be sealed as well, e.g. by interlinked steps in the plate sides (Fuller, Polysius) or by bolting them together as packages (IKN).

The modern grate plates are the basis of modern cooler technology. Problems experienced with the first generation of modern grate plates lead to several detail modifications:

- Cracks in corners of air outlet openings
  → Solution: modified shape
- Plastic deformation caused premature failure with many designs
  → Solution: thermally flexible plates built from two or more pieces
- Preferred plate internal airflow left plates locally uncooled
  → Solution: plate internal guide vanes, optimized air channelling

Modern grate cooler, as the IKN Pendulum Cooler, use also Pneumatic Hopper Drains (PHD) to withdraw the fine clinker fall through.

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Figure 10 Modern grate plates

Fig. 10a Conventional hole plate

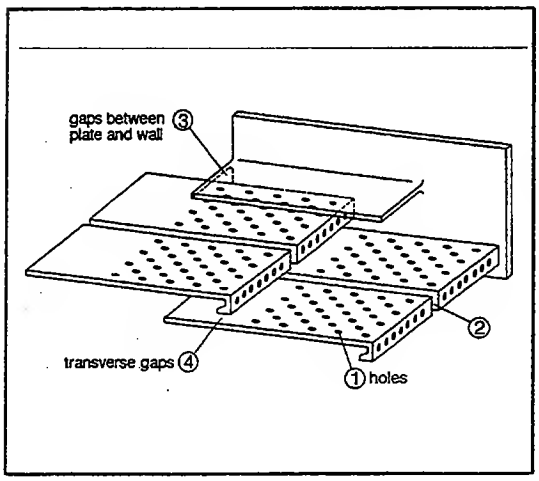


Fig. 10b Coanda plate (IKN)

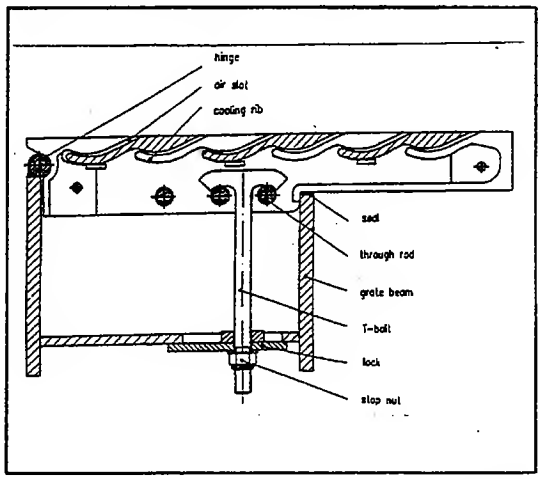


Fig. 10c Mulden plate (CPAG)

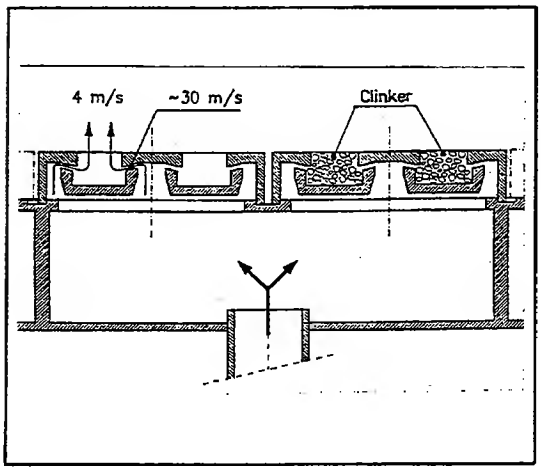


Fig. 10d Controlled flow grate CFG plate (Fuller)

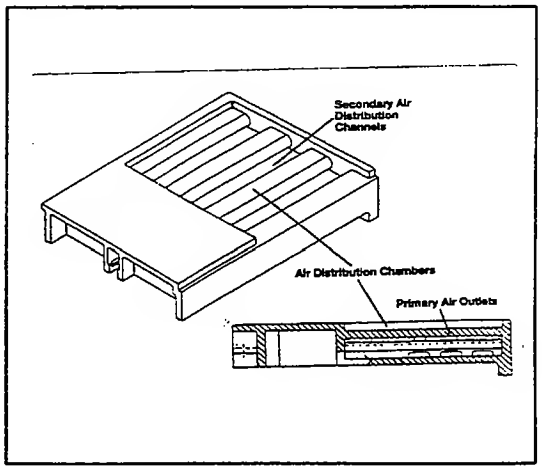


Fig. 10e Jet ring plate (Polysius)

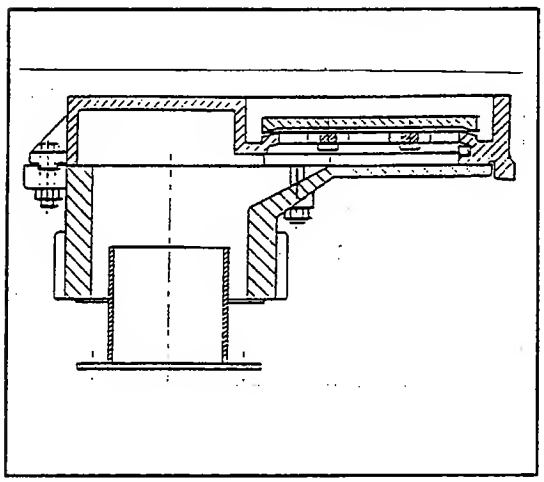
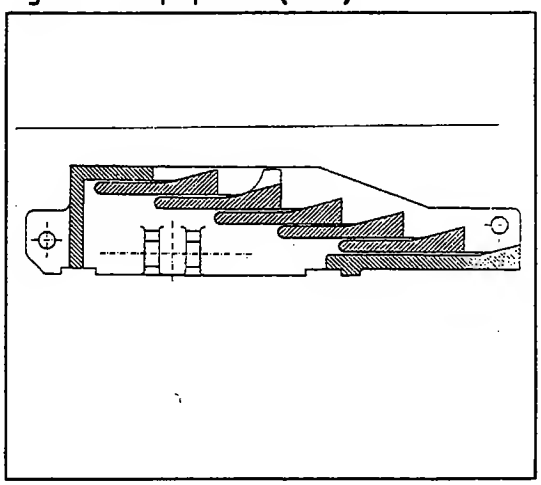


Fig. 10f Step plate (KHD)



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3.1.6.2 Air Ducts

The concept of forced aeration, i.e. the idea to bring the air directly to the grate plates requires a flexible air connection between the (stationary) fan and the moving rows.

Initially, the most obvious and simple approach was chosen: flexible hoses or bellows. IKN, CPAG, Polysius and Fuller used this solution at the beginning.

However, experience showed that those hoses were sensitive to design (geometry), installation and material qualities. While many coolers operated without any problem, others showed frequent rupture of those hoses, very often causing severe plate damage and consequently kiln downtime.

Meanwhile, all suppliers developed new solutions. Only KHD avoided these problems by using telescopic ducts from the beginning.

The individual suppliers are now using the following standard solutions:

- Telescopic air connector (BMH-CPAG, KHD)
- Ball and socket type air connector (FLS, Fuller)
- Gate type air connector (Polysius)
- Open air beam (IKN)

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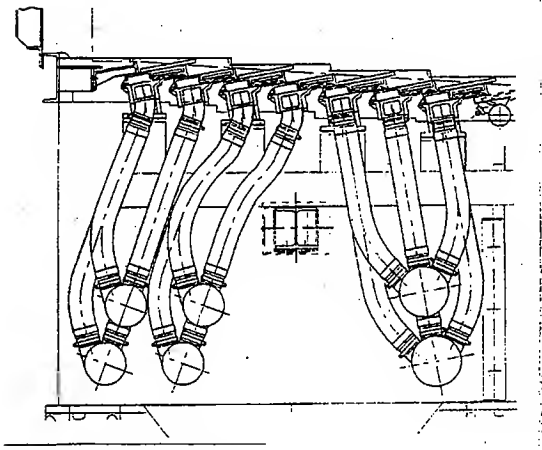
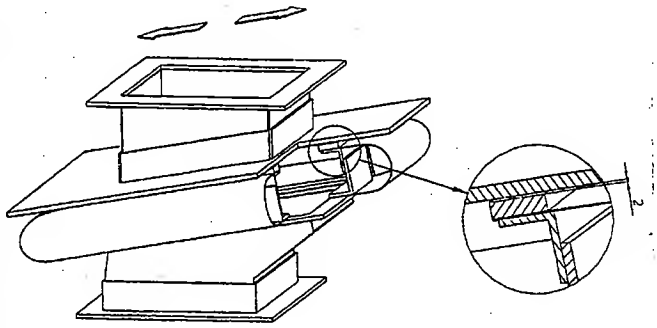
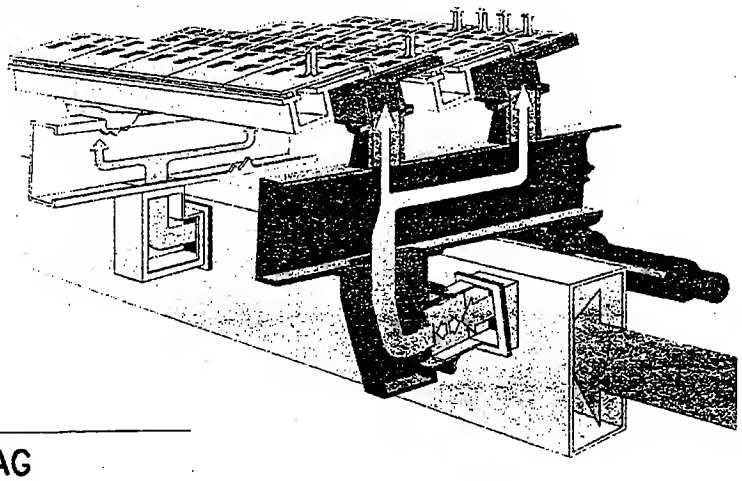
Figure 11 Forced (direct) aeration to moving rows: Flexible ducts



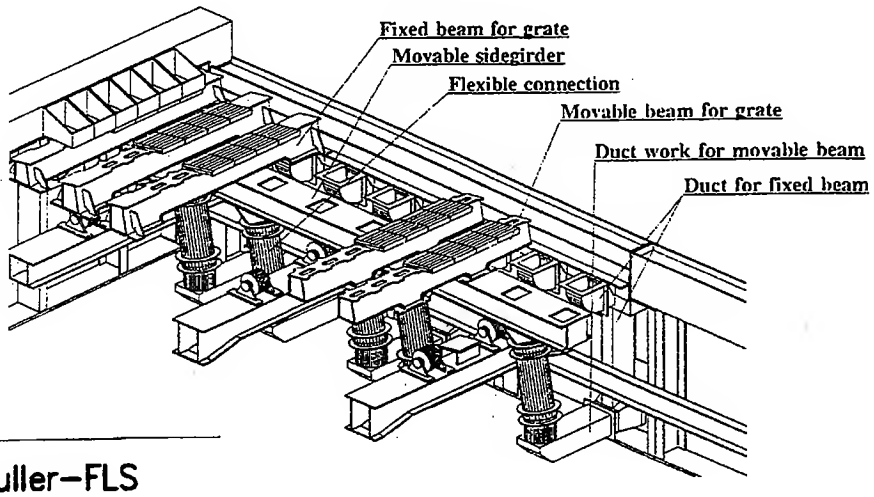
Polysius (old)



Polysius (new)



CPAG



Fuller–FLS

3.1.6.3 Aeration Concept

It was soon recognized that only a few (6 to 8) rows of direct and individual aeration are not sufficient to improve clinker distribution or to eliminate/control red river formation. The number of rows with direct aeration was gradually increased and soon the suppliers started to equip the entire recuperation zone or even the entire cooler with direct aeration. Indeed, this improved the control possibilities, but created the following new drawbacks:

- Complicated and expensive equipment
- More parameters to control
- Difficult access underneath grate
- High number of potential problem areas (flexible hoses!)

Ways had to be found to reduce the number of air ducts to the individually aerated cooler zones. There are two ways to achieve this:

- Reduce number of individually aerated zones
- Modify the air duct system

Today, the following different solutions with varying degrees of experience are presently available from the suppliers:

- No moving rows requiring flexible air connectors in inlet section
- Longitudinal structural beams designed as air ducts
- Short air ducts from one moving row to the next ("Air bridge")
- Direct aeration for fixed rows only ("hybrid aeration")
- Full chamber aeration with modern grate plates

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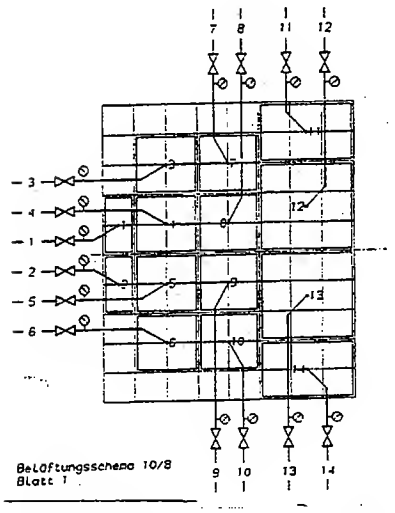
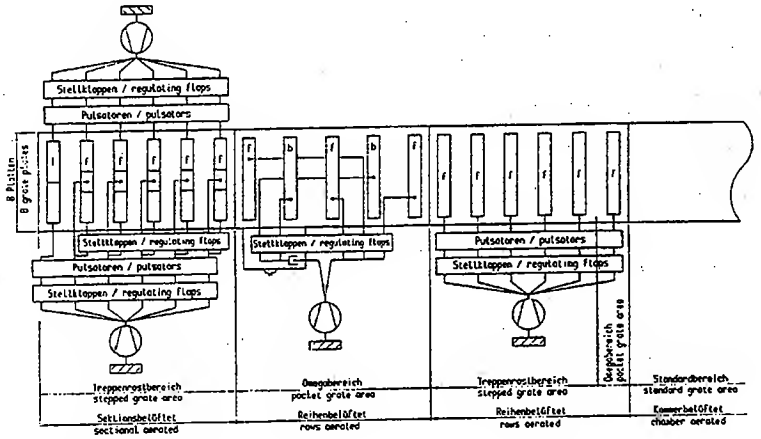
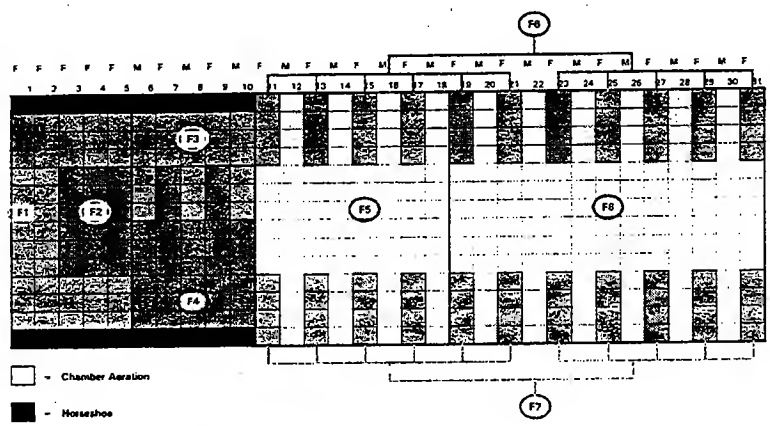
Figure 12 Aeration patterns



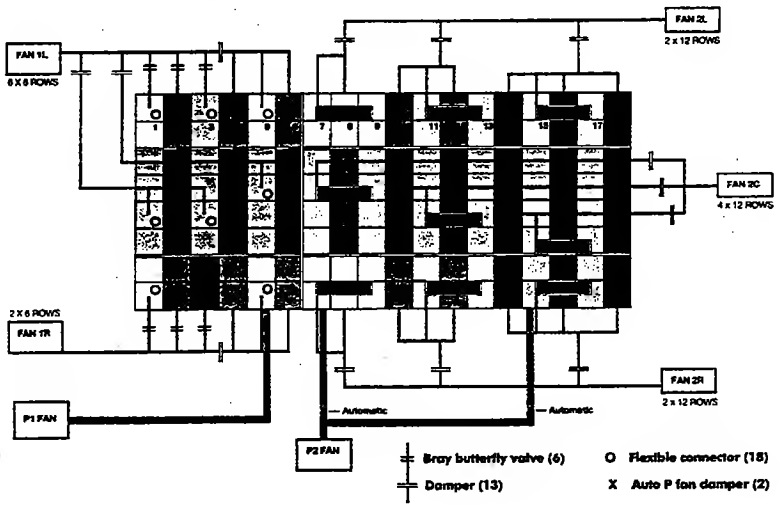
Polysius



KHD



CPAG



Fuller-FLS

3.1.6.4 Seal Air (Confining Air)

When direct plate aeration was introduced, the significance of the seal air or confining air was not properly investigated. It was expected that direct individual aeration of the plates alone would be enough to get the desired improvement due to better air to clinker allocation.

If the cooler grates were tight and had no or very narrow gaps between moving and fixed rows or between grate and cooler casing, this would indeed be true. However, real grates have large gaps, which is one of the reasons why direct aeration was introduced.

The effect of insufficient seal air pressure for direct aerated grates can be explained as follows:

- High resistance in clinker bed (bed thickness, kiln upset, granulometry)
- Cooling air sneaks around plate edge to undergrate compartment instead
- Clinker dust carried in this air → abrasion / wear
- Gap becomes larger → seal air can escape → more "sneak" air
- Stops for repair reduce availability and increase operating cost

Today it is generally accepted that partition, sealing and pressurizing of the undergrate compartments is even more important than with chamber aerated coolers.

Ideally, the partition of the undergrate compartments should repeat the pattern of the individually aerated grate zones of the grate itself. Since this would lead to very complicated and expensive designs with difficult access, simpler solutions had to be found.

One of the most common countermeasures is, to install larger seal air fans. It was interesting to observe the installed cooling air to be gradually increased with each new project. This did not only lead to larger waste air systems but also to higher cooling fan motor power which partially offset the savings expected from modern coolers.

The suppliers have proposed the following improvements:

- Larger seal air fans
- Seal air branched off from cooling air fans
- Seal air from booster fan using air from cooling air fans
- Undergrate pressure controlled by cooling air fan pressure
- Careful sealing of undergrate compartments
- No more moving rows in hot inlet zone

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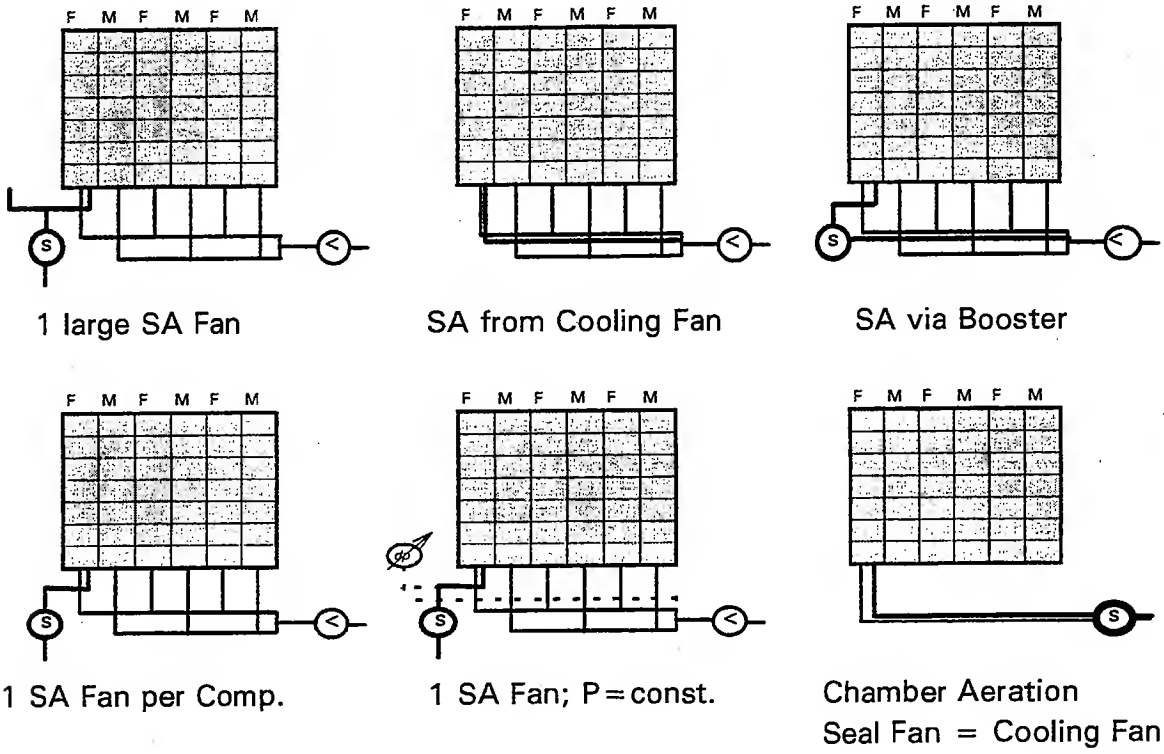
Figure 13 Seal air systems



3.1.6.5 Side Seal Systems

Extremely serious wear problems occurred along the side seal plates on each side of the grate. Excessive fall through along the sides and shockingly short lifetime of the side seal plates, mainly in the recuperation zone, were the result. The main reasons for this problem can be listed as follows:

- The same seal element used for lateral and longitudinal movement
- Side seal plates fixed to cooler casing
- Entire thermal expansion to be compensated by (cold) gap on each side
- Side plates used for lateral guidance of the grate (older designs)
- More lateral thermal expansion of wider grates for large units

The following new solutions have been developed and are now part of the contemporary standards:

- Entirely new side seal plate concepts
- Side seal plates bolted to cross beams of fixed rows (no longer to cooler casing)
- Joints for thermal lateral expansion and mechanical longitudinal movement between moving rows and casing separated
- Center grate guide for large coolers

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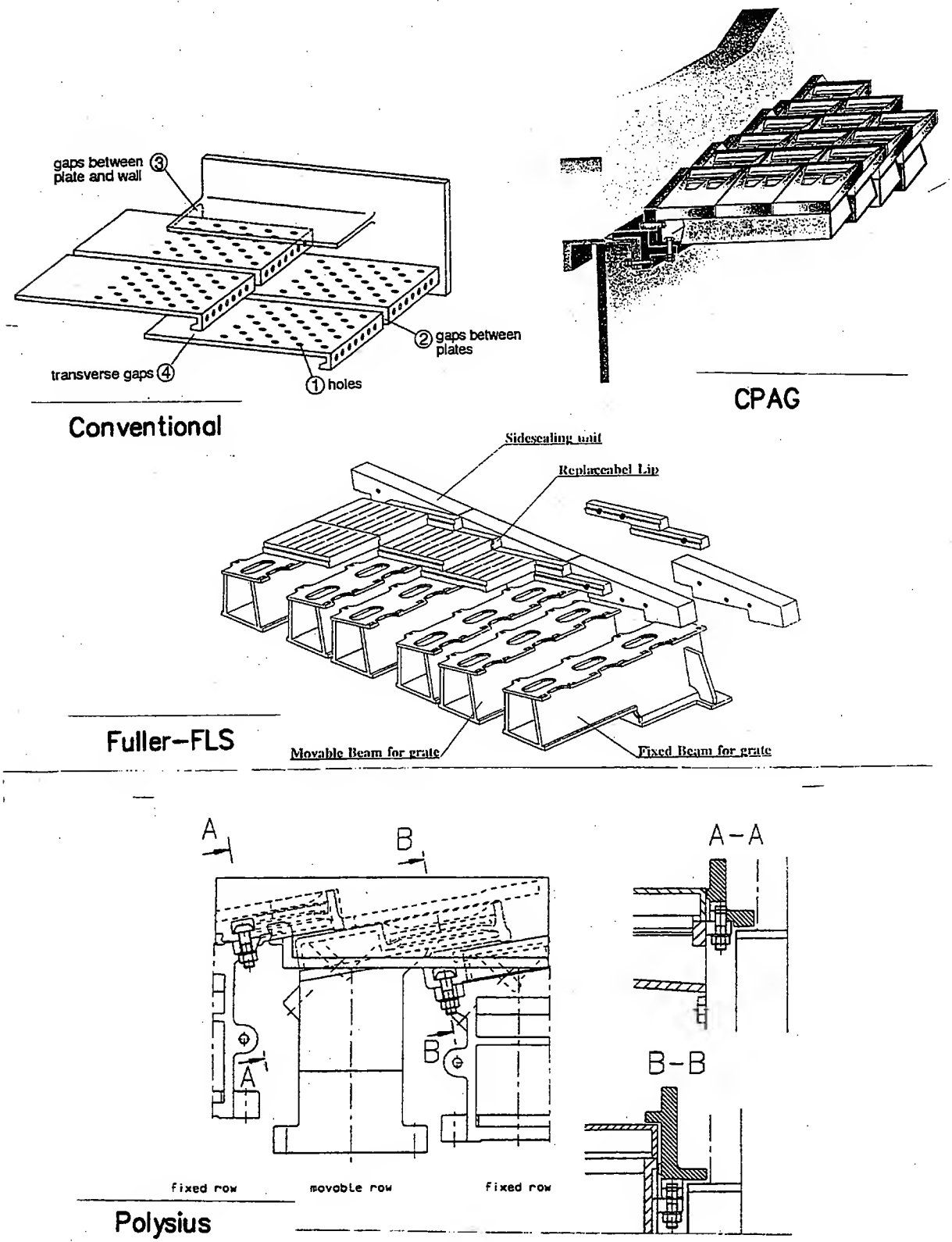
Figure 14 Side seal designs



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3.1.7 Clinker Crushers

All kiln systems produce larger than normal clinker lumps more or less frequently. Large balls of material enter the cooler when coating drops during kiln upsets.

Such large clinker masses can only be cooled superficially and contain a lot of heat. Before being discharged to the clinker conveyor, they must at least be crushed to smaller particles.

All clinker coolers, regardless of the type, are equipped with a clinker crusher. Traditionally, this is a hammer crusher which has proven to be reliable.

In order to cool large clinker lumps, they must be crushed within the cooler. In reality, this means installing the crusher before the last grate. Early trials with hammer crushers were not successful, however.

Based on the idea and experience with roller grate bottoms in shaft kilns (and shaft coolers), CPAG developed the roller crusher to be used as intermediate crusher in a step cooler.

The advantages of the roller crusher make it also superior at the cooler outlet. Hydraulic or electric drives as well as different combinations of reversing rollers are available from various suppliers.

Compared to the hammer crusher, the roller crusher is rated as follows:

Strengths	Weaknesses
• low speed	• higher initial investment
• low wear	• chokes easier
• low dust generation	• more difficult to design
• equalization of material rushes	
• suitable for high temperatures	
• lower power consumption	

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Figure 15a Hammer crusher

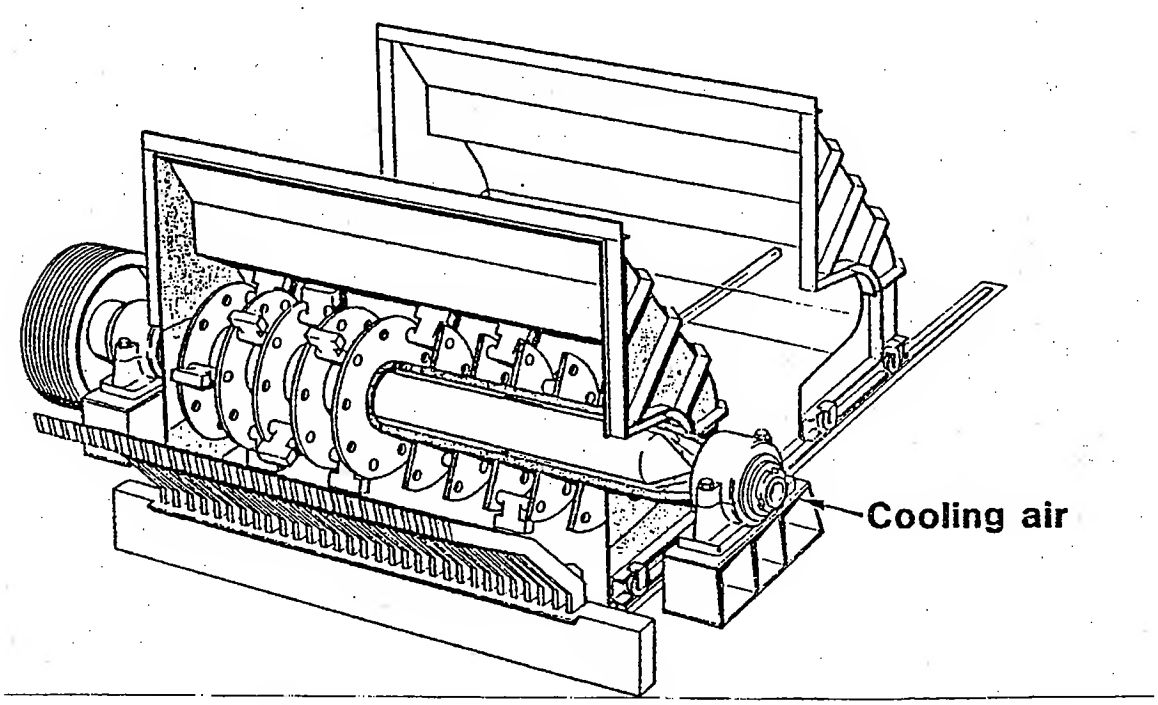

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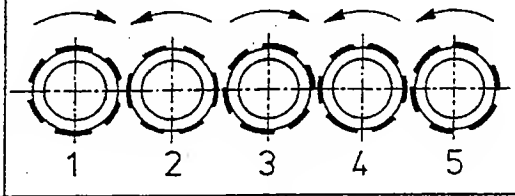
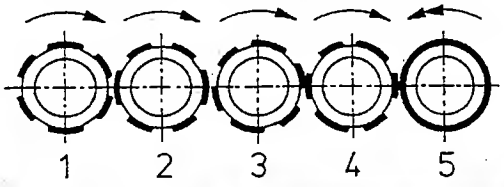
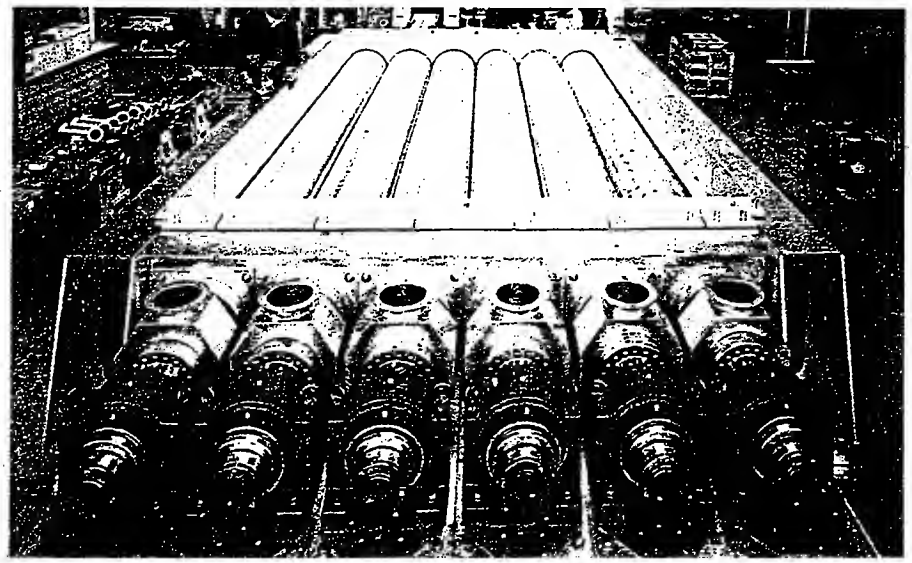
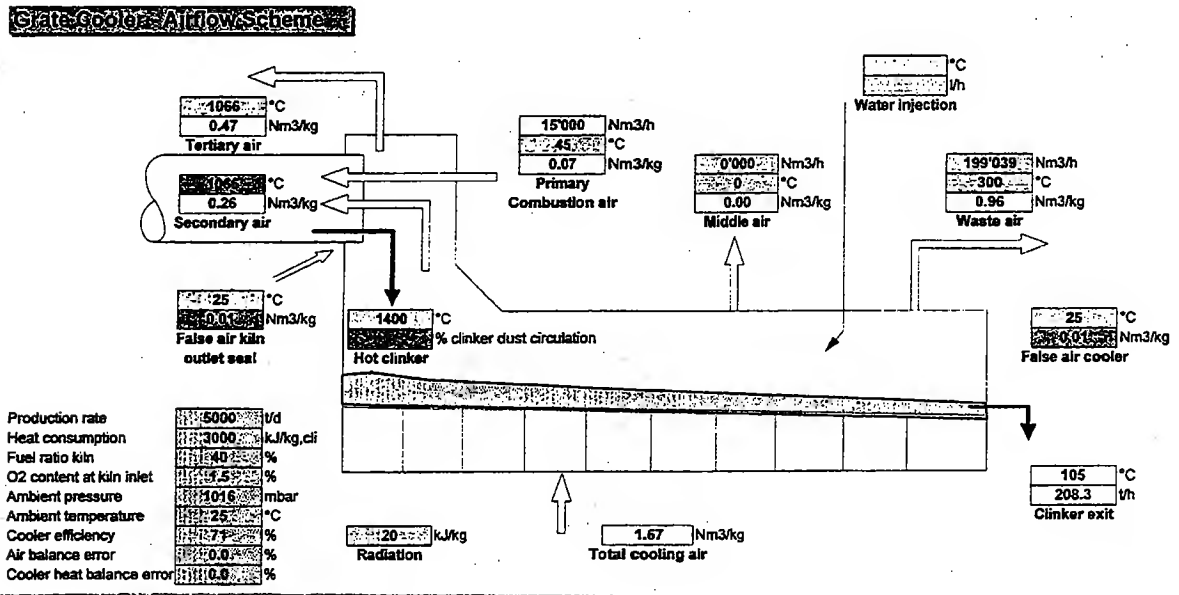
Figure 15b Roller crusher

NORMAL OPERATION ROLLER 1 - 4 : 4 RPM ROLLER - 5 : 16 RPM	
	CLINKER FLOW DIRECTION >>>
REVERSING OPERATION ROLLER 1 - 4 : 4 RPM ROLLER - 5 : 180 RPM	



CPAG

Figure 16 Heat and air balance of a modern Grate cooler



HEATBALANCE SUMMARY

INPUT		[°C]	[kJ/kg,cli]	
Clinker from kiln		1400	1504.2	99.3%
Cooling air	sensible heat	25	10.8	0.7%
False air	sensible heat	25	0.1	0.0%
Water injection	sensible heat			
Total of inputs			1515.0	100.0%
OUTPUT				
Clinker	sensible heat	105	66.9	4.4%
Secondary air (dust incl.)	sensible heat	1066	379.3	25.0%
Tertiary air	sensible heat	1066	695.4	45.9%
Middle air	sensible heat	0	0.0	0.0%
Waste air	sensible heat	300	353.7	23.3%
Radiation loss			20.0	1.3%
Water evaporation				
Rest 0.0 %			-0.2	0.0%
Total of outputs			1515.0	100.0%

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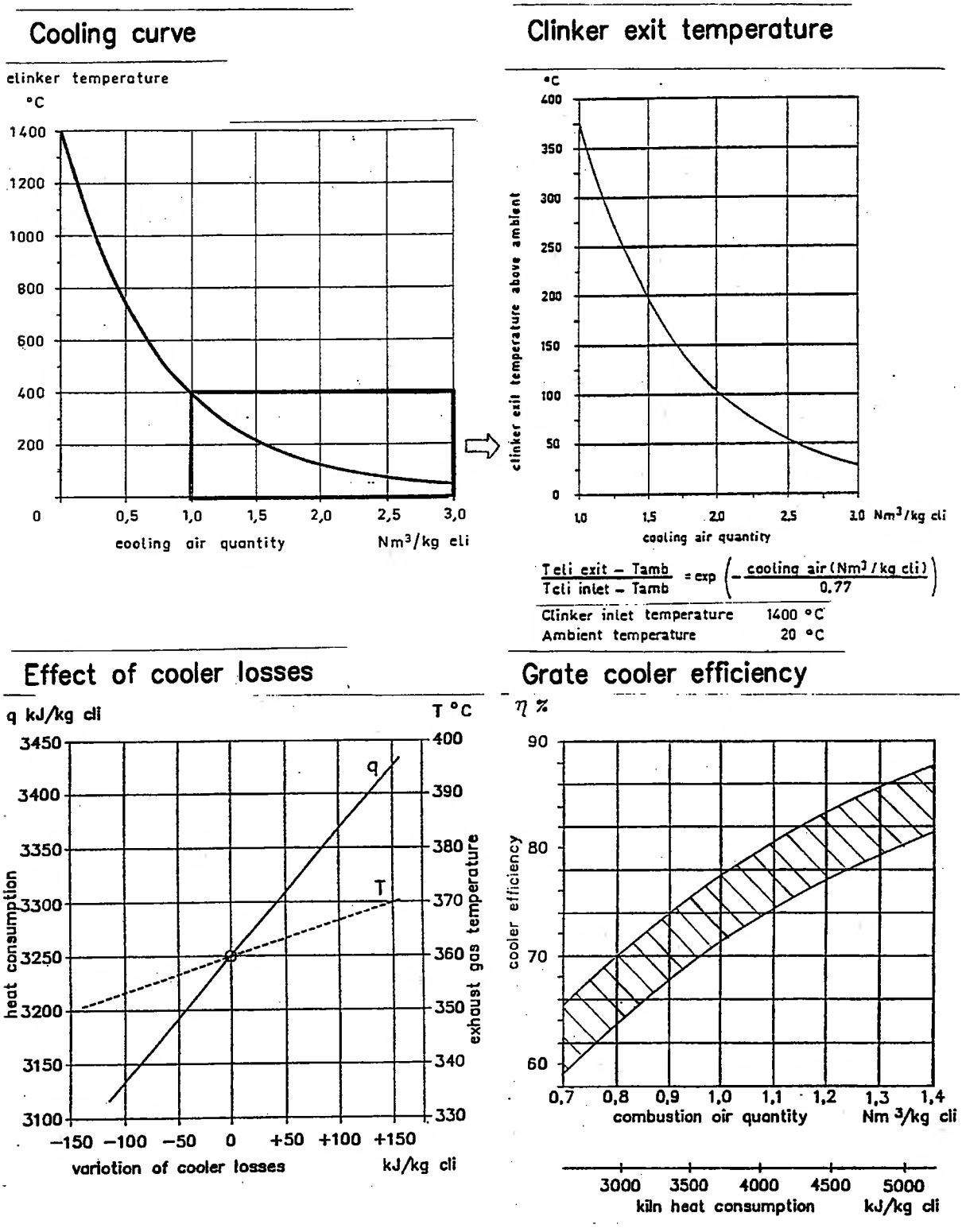
Figure 17 Optimization



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3.1.8 Cooler control

One of the advantages of the reciprocating grate cooler is its high flexibility, due to operating variables adjustable independently from kiln operation. Usually three main variables are controlled automatically.

a) Grate speed

In order to prevent the clinker bed resistance from exceeding the pressure capabilities of the cooling fans (which would mean too little cooling air and danger of heat damage), the bed resistance on the grate should be kept constant.

To do this, each grate section drive is controlled by the undergrate pressure of the first or second compartment in each grate section. An increase in pressure indicates an increase in bed resistance (either more material in the cooler or finer material). The reaction is an increase of the grate speed, causing the bed to become thinner. If the undergrate pressure decreases, the drive slows down and the bed becomes thicker.

Another possibility is to control only the first grate by the undergrate pressure, and to keep the speed of the following grates proportional to the speed of the first grate.

More sophisticated control systems use the weighted average of several undergrate pressures to control first grate speed. In many cases, however, control systems amplify fluctuations from the kiln instead of smoothening them. Increasing the bandwidth of the control system has shown good results in several cases.

b) Airflow

This control is complementary to the grate speed control. It maintains a constant volume of cooling air entering the cooler independently from the grate underpressure.

Each cooling fan is equipped with a piezometer sensor which will recognize an increase or decrease of the airflow and cause the cooling fan damper to close or open (in case of inlet vane damper control) or the fan motor speed to decrease or increase (in case of variable speed fan drives).

During normal conditions the cooling fans operate at about 2/3 to 3/4 of their maximum performance so that enough spare capacity is left to cope with eventual kiln rushes.

Together, grate speed and air flow control will on one hand ensure a sufficient cooling air supply to the cooler and, on the other hand, tend to provide more uniform combustion air temperature to the kiln.

c) Hood draft

The third component of the cooler control system is the hood draft control.

An automatically controlled grate cooler can improve the whole kiln operation and allows the operator to concentrate on other problems.

The kiln hood pressure is used to regulate the cooler vent air fan speed to maintain a constant pre-set draft. As the draft tends to become positive, the cooler vent fan speed is increased. This takes more air from the cooler and maintains the draft setpoint. As with the other controls, reaction in the opposite direction is just as important.

Coolers with radiation walls (IKN) allow hood draft control by one of the first cooling air fans.

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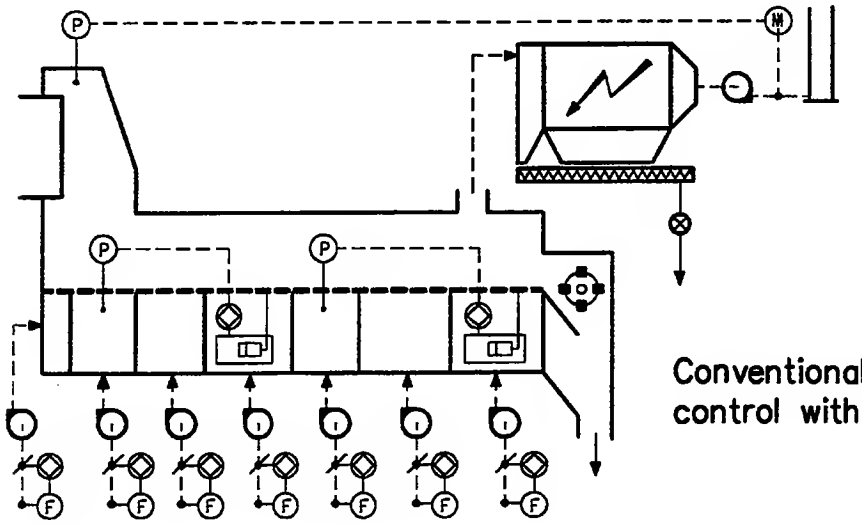
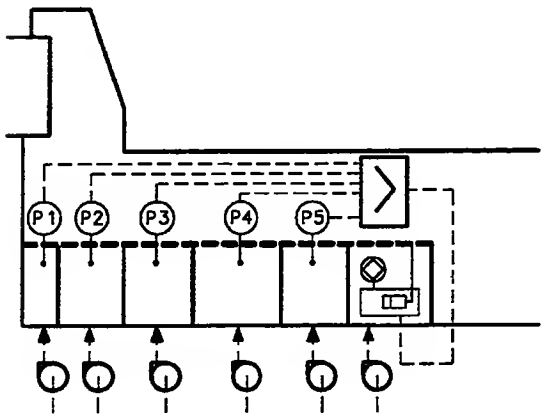
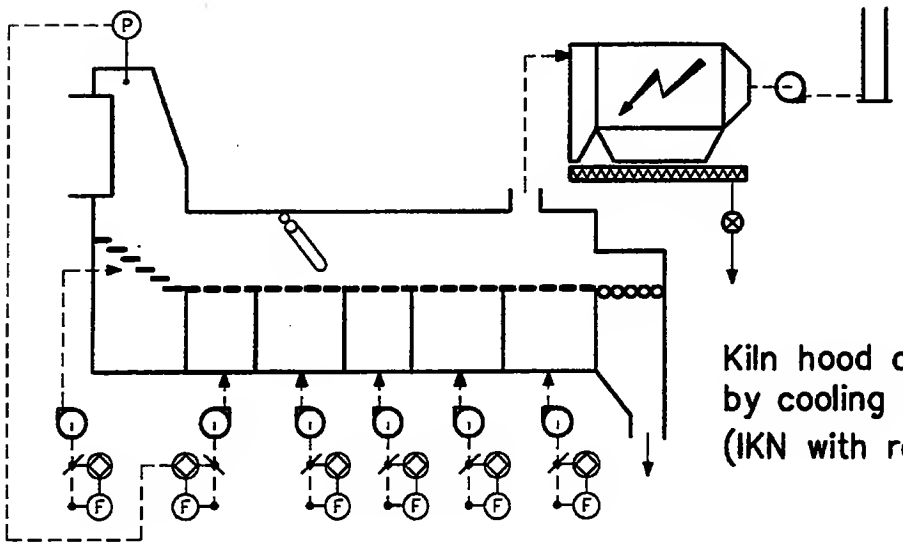
Figure 18 Cooler control



Conventional grate cooler control with 3 loops



First grate speed control with weighted pressures from several compartments



Kiln hood draft control by cooling air fan (IKN with radiation wall)

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3.1.9 Cooler Dedusting

While dedusting of kiln exhaust gas can be commonly solved by using one type of dust collector only (electrostatic precipitator), the choice of the most adequate system for dedusting clinker cooler vent air raises quite often many discussions. This choice problem is basically a result of the special and fluctuating conditions of the vent air to be dedusted:

		normal operation	kiln upset
airflow (actual volume)	%	100	up to 150
air temperature	°C	200 - 250	up to 450
air dew point	°C	5 - 20	5 - 20
dust load	g/Nm^3	5 - 15	25 - 35

The dust particle size distribution can vary in a wide range depending on the burning conditions in the kiln.

Dimensioning of the dedusting equipment must take into account the worst conditions, in order to maintain the required clean gas dust content even at kiln upset condition.

The types of dust collectors for this application are compared below. Today's trend is:

- multiclones will no longer be tolerated in new and many existing plants
- gravel bed filters have proved to be inefficient and expensive
- use of electrostatic precipitators is possible without restriction
- bag filters with cooling of the vent air in a heat exchanger are often used nowadays

Type of collector	Strengths	Weaknesses
multiclone	simple low investment cost low space requirement not sensitive to temperature peaks	poor efficiency for particles < 20 µm efficiency sensitive to gas flow fluctuation comparatively high pressure loss high operating cost
electrostatic precipitator	low pressure loss low operating cost low maintenance cost	big unit required or use of pulse generator -> high investment cost possibly water injection required
gravel bed filter	not sensitive to temperature peaks	highest investment cost highest pressure loss high operating cost
bag filter	high efficiency relatively low investment cost	no bags for temperatures up to 450°C → precooling required high pressure loss high operating cost high maintenance cost

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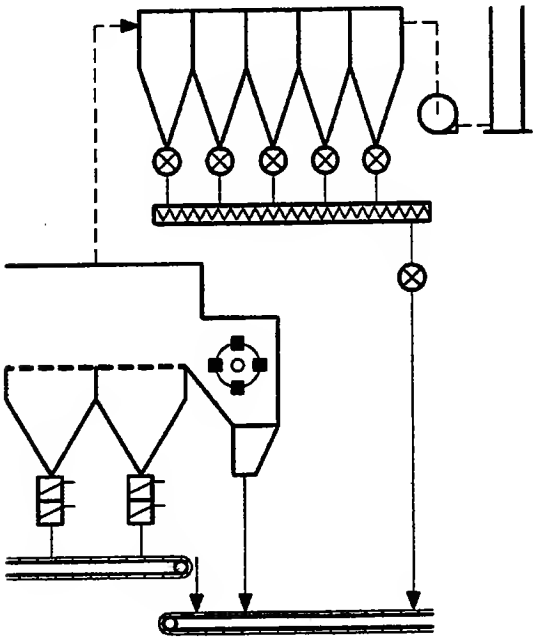
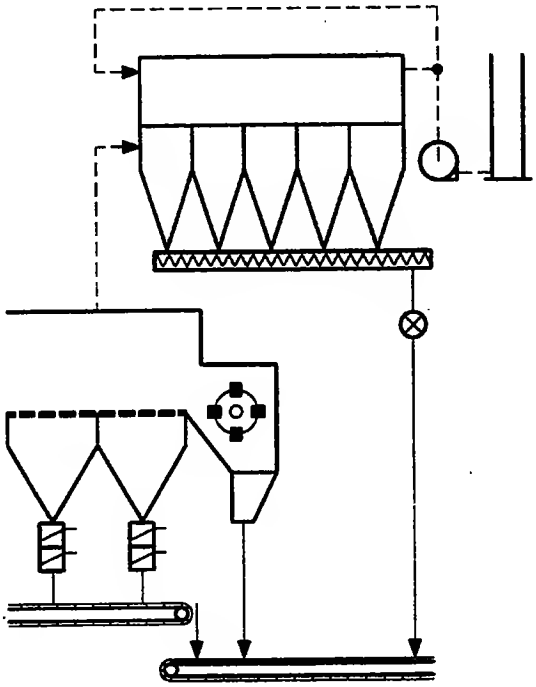
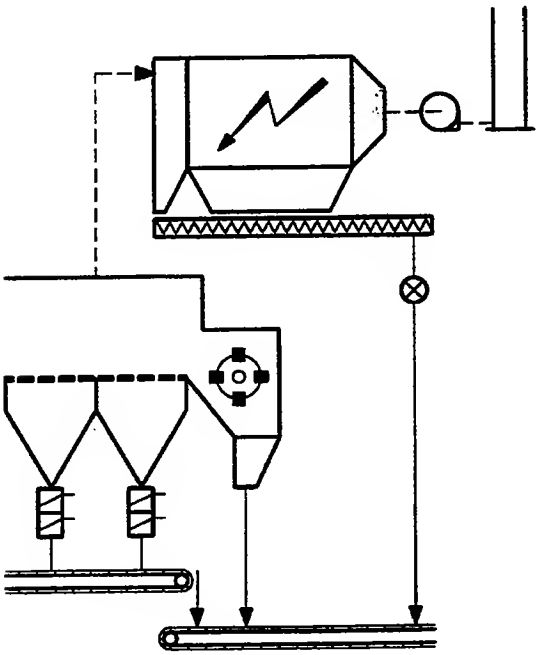
Figure 19 Grate cooler dedusting



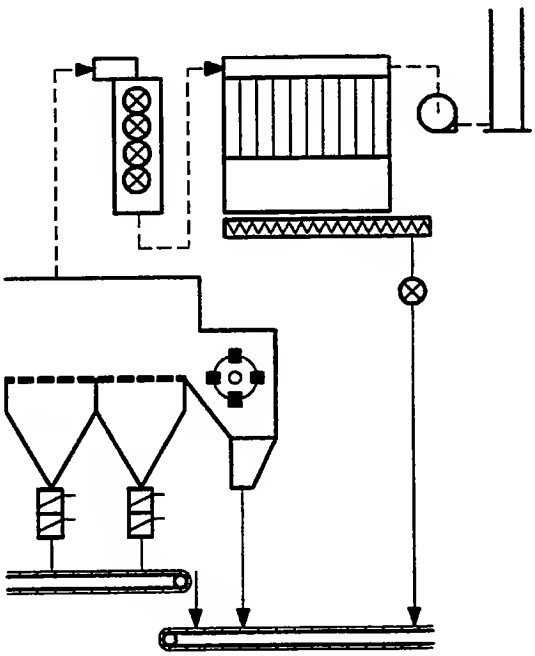
Multiclone



Gravel bed filter



Electrostatic precipitator



Air to air heat exchanger and bag filter

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3.1.10 Developments

Air recirculating (Duotherm) cooler

A patent has been taken out in 1970 by the "Société des Ciments Français" concerning the recirculation of the vent air after sending it through a heat exchanger.

The first application of the unconventional system has been realized in 1970 at the Beaucaire plant of the above mentioned company, on a 1500 t/d Fuller cooler.

Initial experience gained with this installation was very satisfactory.

Only few installations using this principle have been realized, e.g. in the Ulco plant. The main advantages and disadvantages of this system are:

Strengths	Weaknesses
• no dust emission at all • simple • low investment cost • heat recovery possible (at various temperature levels) • extension possible by adding further heat exchange units	• possible wear of fan blades (preventative measures necessary) • maintenance and operating costs higher than conventional cooler dedusting system with EP

Modern cooler technology and problems in some cases have pushed this idea in the background. However, it might be reactivated if it can be combined with modern cooler systems.

Dual pass cooler

A completely new principle of cooling in a grate cooler has been introduced by Polysius in 1994: the dual pass cooler or REPOL-ZS.

This cooler can be considered a two-grate cooler with intermediate crusher where grate 1 and 2 are identical.

The hot, 1400°C clinker from the kiln is fed on top of a layer of colder clinker already laying on the cooler grate. At the end of the grate, the now cold lower clinker layer is extracted via a special system consisting of reciprocating bars and a hopper. The upper layer which has reached about 500°C passes a roller crusher and is then returned to a intermediate hopper below the kiln from where it is fed onto the empty grate to pass the cooling air a second time, this time below the fresh hot clinker.

One 1400 t/d unit is in operation in Germany using Jet-Ring technology. With less than 1.6 Nm^3/kg cooling air, extremely low clinker temperatures have been reported. The crucial problems of this solution are intermediate transport and storage.

In spite of the compact size, high cooling degree with low air flow and low plate temperatures, this cooler will only be successful if the intermediate temperature level can be increased and the heat losses reduced.

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Figure 20a Non venting cooler

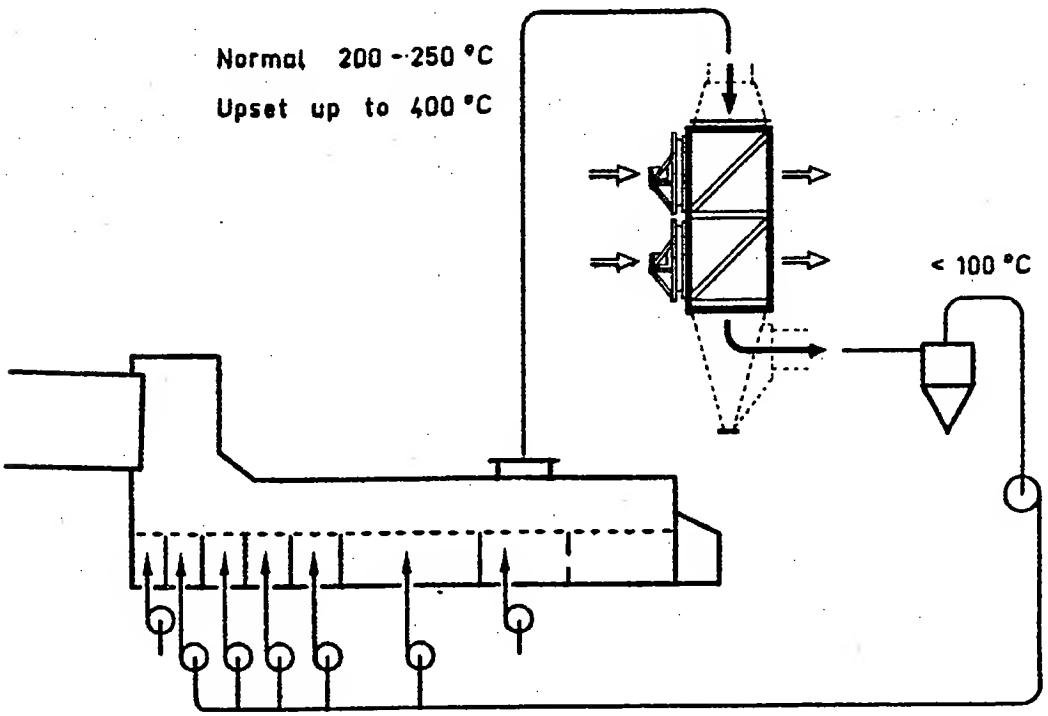
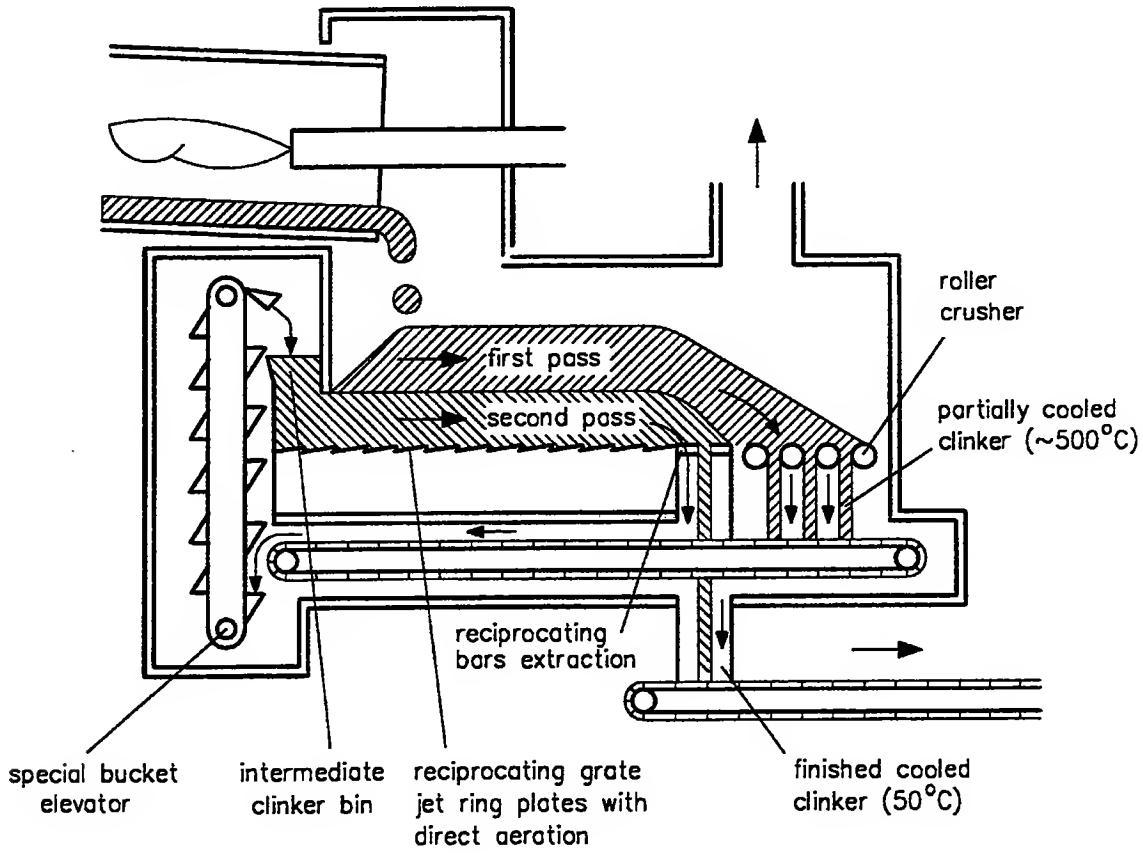


Figure 20b Dual pass cooler (Polysius)



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3.2 The Cross Bar Cooler

3.2.1 Principle

F.L.Smidth and Fuller developed together the new SF (Smidth - Fuller) Cross Bar Cooler representing a completely new concept.

The basic idea was to develop a cooler in which conveying of clinker and air distribution systems are separated. The SF cooler has a clinker conveying device installed above an entirely fixed grate.

In addition the cooler should be less complicated, more efficient and easier to operate than other grate coolers on the market. Sealing air is eliminated and the distribution of air is optimized for all modes of operation

The thermal behavior of the SF cooler (e.g. heat balance, recuperation) is similar to the other grate coolers.

3.2.2 Main features

- One inclined fixed grate.
- Clinker conveying by cross bars, separate from air distribution.
- No thermal stress of grate.
- Minimum wear on grateplates due to a dead layer of clinker (50 mm) protecting the grate surface. The thickness is given by the space between the cross bars and the grate. (Anticipated service life time at least 5 years)
- Dynamic flow control unit (mechanical flow regulator) for each grate plate. The mechanical flow regulator maintains a constant airflow through the grate and clinker bed, irrespective of the clinker bed height, particle size distribution, temperature, etc.
- No fall through of clinker to the undergrate compartment.
  → Eliminating undergrate clinker transport resulting in low installation height for new plants.
- Easy cooler operation by elimination of sealing air and automatic control of air distribution.
- Modularized cooler concept → short delivery and installation time.
- Different drive speeds across the cooler possible.
  → Additional control of clinker distribution.
- Fewer and less expensive wear parts (easy to replace).
- Easy visual inspection of undergrate compartment (clean undergrate, windows).
- Sustainably high thermal cooler efficiency throughout the lifetime of the cooler.
  → Reduced system heat consumption.

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Figure 21a: SF Cross Bar Cooler

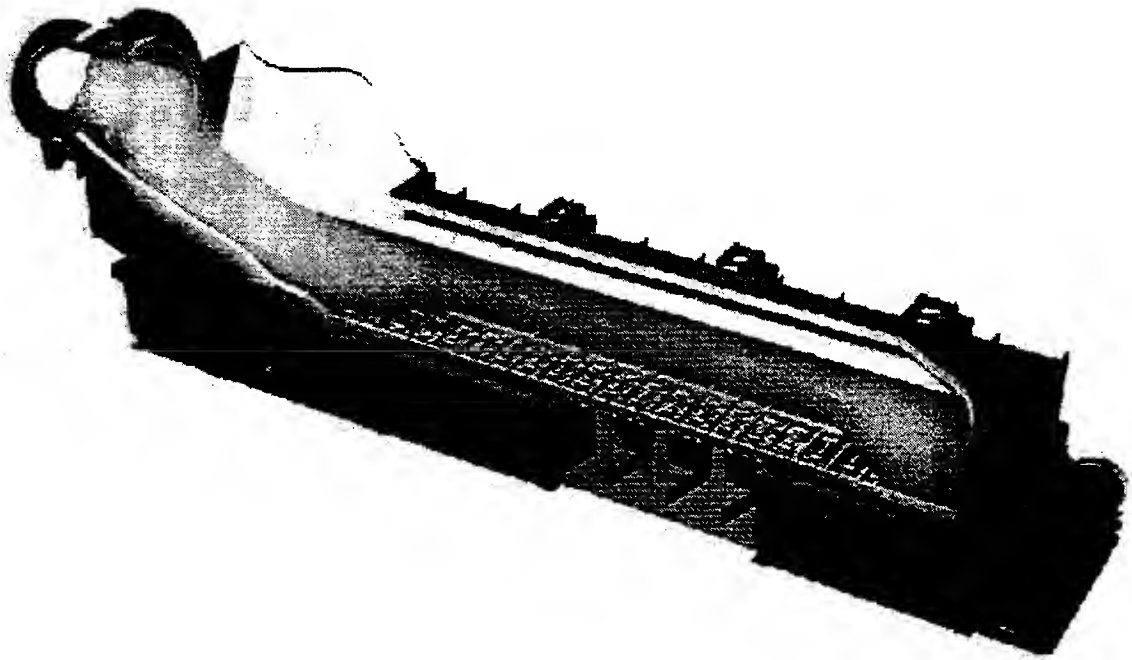
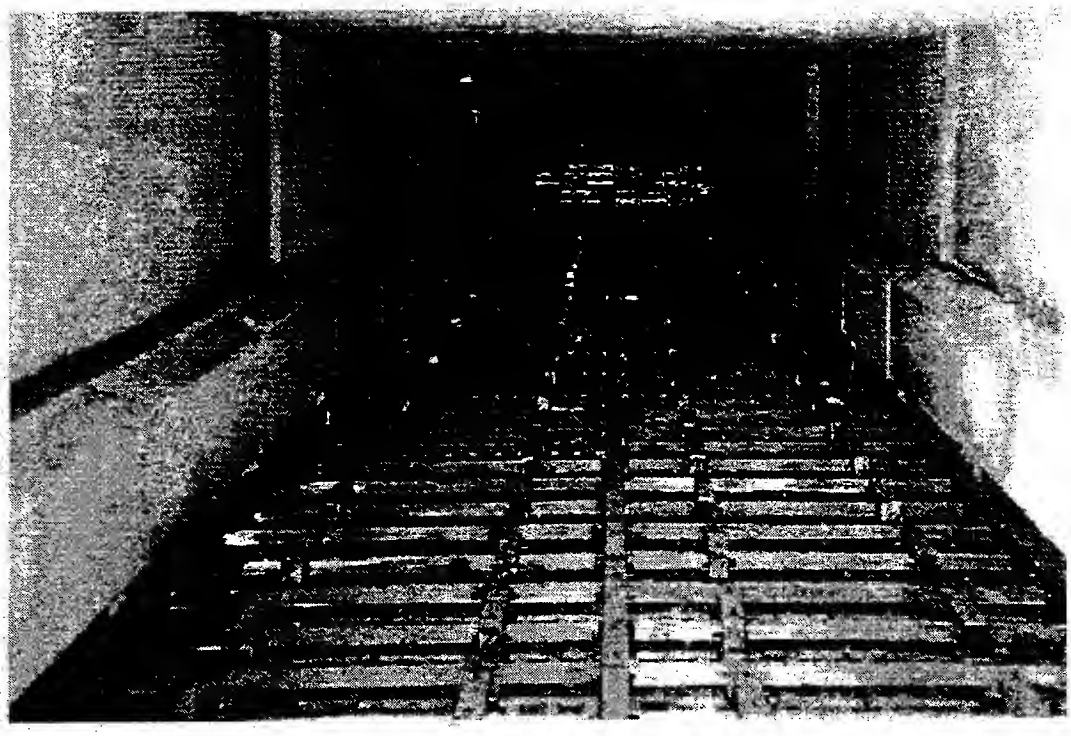


Figure 21b: SF cooler grate with cross bars



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3.2.3 Strengths and Weaknesses

Strengths	Weaknesses
• No clinker fall through (no hoppers, no dragchain). • The grate is protected from overheating. • Very high availability is expected. • Wear and tear affects only the conveying system and not the air distribution system. • For each plate, the cooling air is individually controlled. • The amount of cooling air is about 1.6 to 1.8 Nm^3/kg. • Reduced height and maintenance required since the undergrate clinker transport can be dropped. • Time for installation is short due to modular concept.	• The clinker bed seems to be influenced by the conveying reciprocating cross bar, resulting in disturbed clinker layers. • In case of fine clinker and coating drops, air breakthroughs can occur. • The performance of the mechanical flow regulator (amount of cooling air) and its distribution is yet to be assessed. • Airflow through the fixed grate at the cooler inlet (CIS) can generate dust and dust cycle.

Remark: So far, no SF Cross Bar Cooler is in use within the "Holderbank" group and therefore no first hand experience is available. Worldwide, there are only three SF cross bar coolers installed. Two of a capacity of 450 t/d and one of 2000 t/d. (as of January 1999)

Figure 22a: Cross Bars: Easy to replace wear parts

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Figure 22b: Mechanical flow regulator

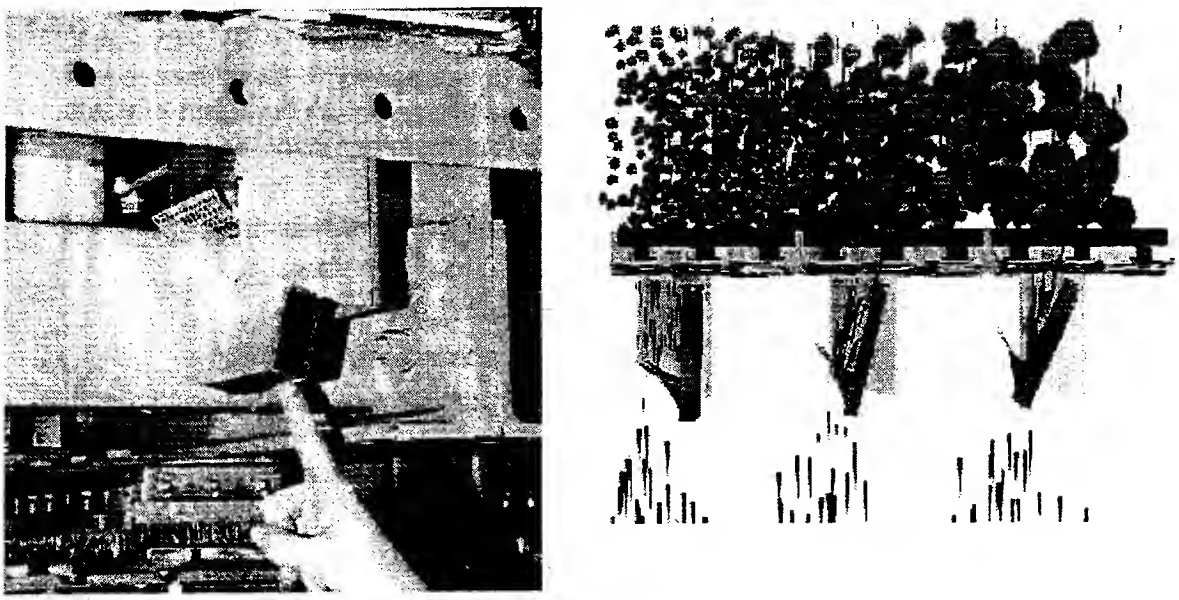
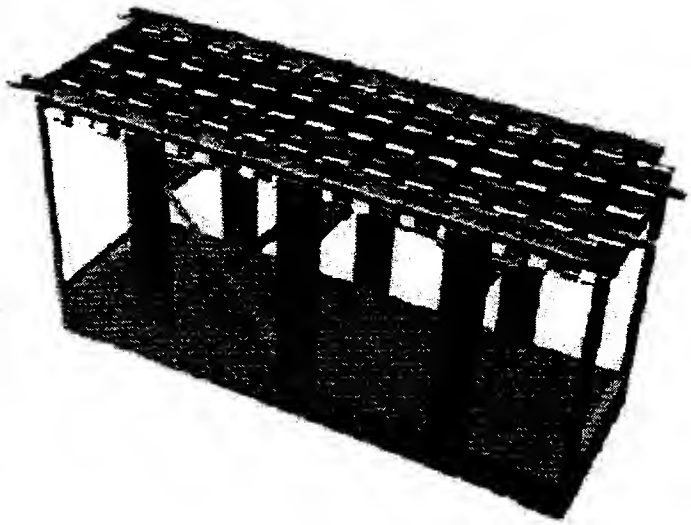


Figure 22c: Modular concept: One module



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3.3 The Travelling Grate Cooler

3.3.1 Principle

The traveling grate cooler (Recupol) was originally developed by Polysius for use in combination with grate preheater (Lepol) kilns. Using the same principle and similar technology, it uses the same wear parts. The following main components can be distinguished:

- Casing with kiln hood and connections for air at different temperature levels
- Inlet with water cooled chute (2nd generation) and pulsator
- Traveling grate with return carrying idlers and drive system
- Aeration system with fans, undergrate compartments
- Riddling extraction system with chutes, flap gates, hoppers and transport
- Clinker crusher

- **Material transport**
  The clinker is carried by a horizontal traveling grate which works like a stationary caterpillar chain with perforated chain plates. In contrast to the reciprocating grate cooler, the clinker does not tumble over plate edges, but remains as undisturbed layered bed from inlet to discharge.

- **Heat exchange**
  Heat exchange takes place, like for the reciprocating grate according to the **cross current** principle. Because the layers remain, it should be even better, at least theoretically.

- **Cooling air**
  Ambient air is blown by a number of cooling air fans to underneath of the travelling grate plates carrying the clinker. Pressure and flow criteria of cooling air are basically as for the reciprocating grate cooler.

- **Water cooled inlet chute**
  In order to achieve rapid cooling in the inlet section, but also to protect the travelling grate from the highest clinker temperatures, Recupol coolers were equipped with a water cooled inlet chute.

- **Key figures / KPI**

Specific grate loading:	25 - 30 t/d m^2	(design)
Largest units:	3000 t/d	(Lägerdorf kiln 10)

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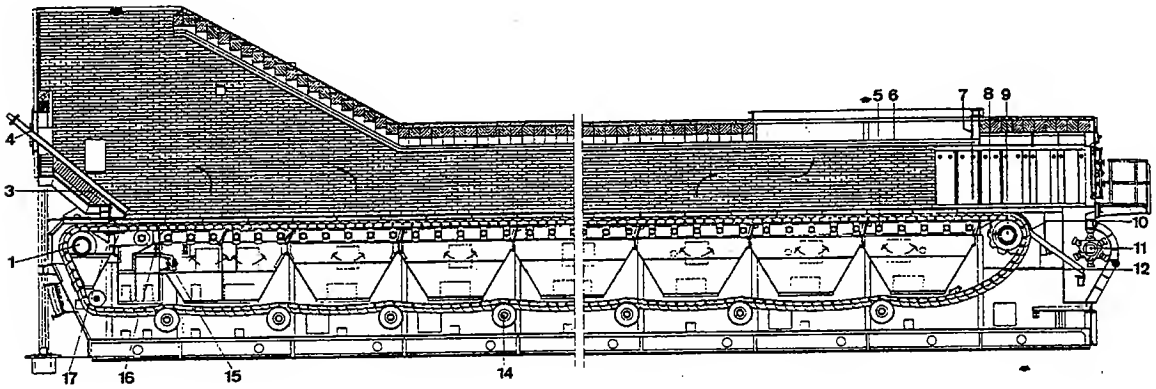
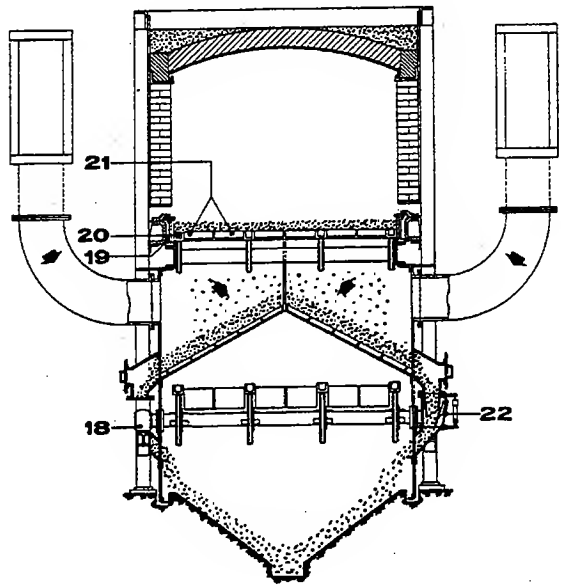
Figure 23 Travelling grate cooler





1 Turning shaft
3 Supporting girder
4 Water cooled steel plate
5 Shaft for upper traveling route
6 Siding plate
7 Chain curtain
8 Chain wheel
9 Drive shaft
10 Strip off grate
11 Clinker breaker
12 Grate bolt
14 Shaft for lower traveling route
15 Pulsator
16 Blower nozzle
17 Drag plate
18 Slide bearing
19 Sealing elements
20 Chain link
21 Grate plate
22 Flap gate

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3.3.2 Strengths and Weaknesses

Travelling grate cooler compared to reciprocating coolers:

Strengths	Weaknesses
• Possibility of replacing grate plates during operation (on the returning part) • Undisturbed, layered clinker bed is better for optimum heat exchange	• Larger machine for the same grate area equipment requiring more space and higher civil cost • Lower specific grate loadings adding further to overall size • More expensive to build than a reciprocating grate cooler • The absence of clinker movement (see above) was often considered a disadvantage because of cases where a solid (fritted) layer on top of the clinker bed made it impermeable for air. For this reason, pulsators were installed for first cooling fans. • Much higher maintenance requirement with ageing equipment • Heat loss via cooling water for inlet chute

Due to the mentioned weaknesses, Polysius eventually decided to develop their own reciprocating grate cooler (Repol) around 1980:

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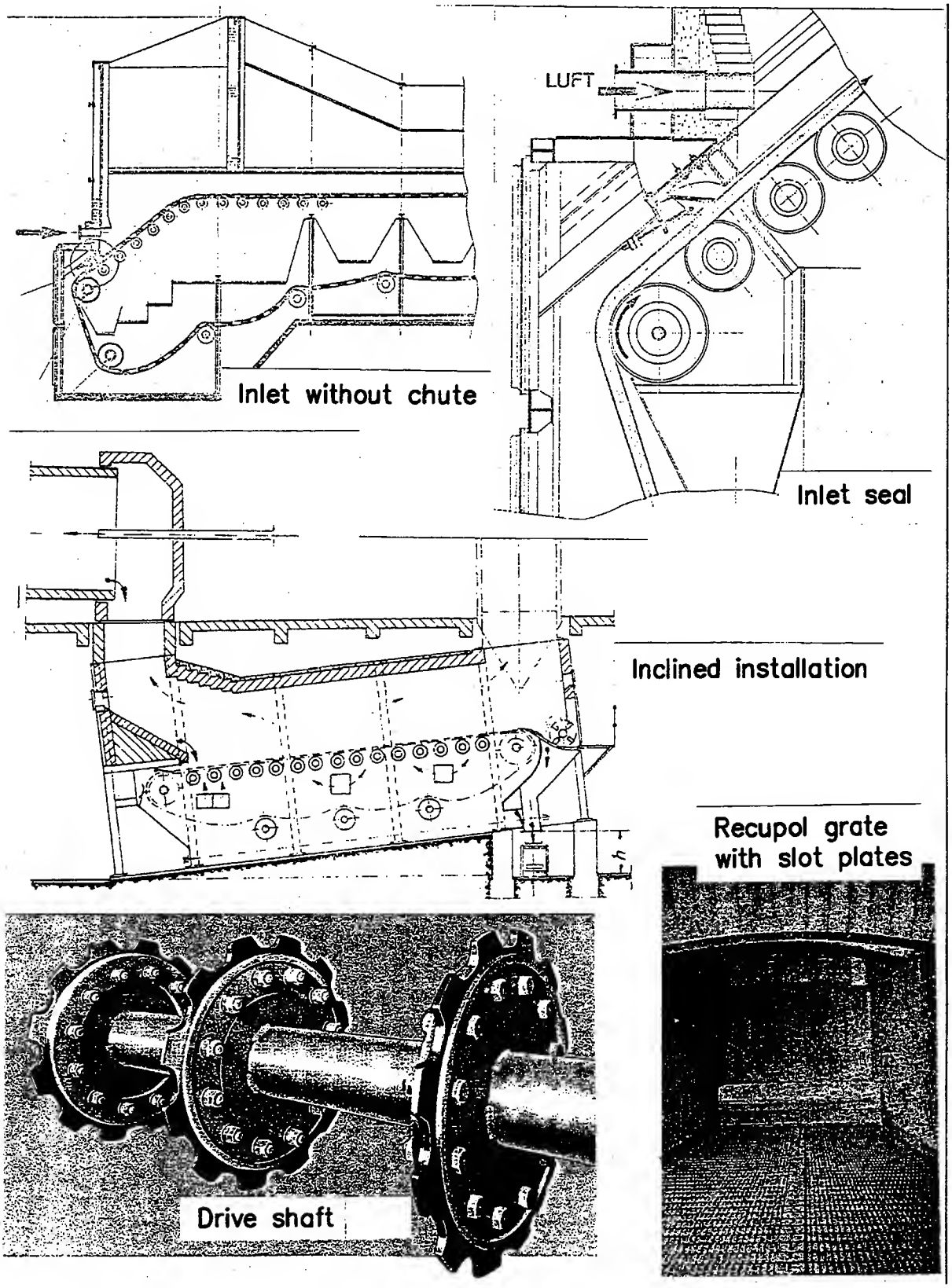
Figure 24 Travelling grate cooler: Design details



4. ROTATING COOLERS

4.1 The Rotary Cooler or Tube Cooler

4.1.1 Principle

The rotary cooler consists mainly of a rotating cylinder, similar to a rotary kiln.

The clinker is fed through the inlet chute and is then cooled by air while being transported towards the outlet end. Cooling is performed in countercurrent flow. The tube is equipped with internal lifters which improve the heat transfer. About 2/3 (66%) of the cooler length is lined with refractory bricks.

The rotary cooler is of simple design and is the oldest type of clinker coolers. It was seldom used for modern, large kiln systems. Therefore comparatively little design and operating experience is nowadays available for rotary coolers above 2000 t/d. However, the application of rotary coolers still offers certain advantages. Presently units up to 4500 t/d (dimensions dia 6.3/6.0 x 80 m) are in operation. It will be interesting to follow the future development of large rotary coolers.

4.1.2 Design Features

- **Arrangement** of the rotary cooler is normally in the extension of the kiln axis; in many cases the reverse manner (underneath the kiln) has been applied.
- The **diameter** of the cooler is similar to that of a corresponding suspension preheater kiln. Likewise the rotating speed is in the same range as for the kiln (max. 3 rpm). Length/diameter ratio: L/D ~ 10.

Many cooler tubes are designed with an extension in diameter in order to reduce air velocity.

- The **inclination** is comparatively high (in the order of 5%).
- Like for all rotating coolers, the **internal heat transfer equipment** is an important part of the rotary cooler. Its task is to generate additional area by scattering the clinker without generating too much dust. Basically a similar design may be applied as in a planetary cooler tube (see next chapter) however the following differences must be considered:
  - The clinker falling heights are larger. Wear protection of shell and lining is essential.
  - At a comparative length position the clinker in a rotary cooler is hotter than in a planetary cooler.

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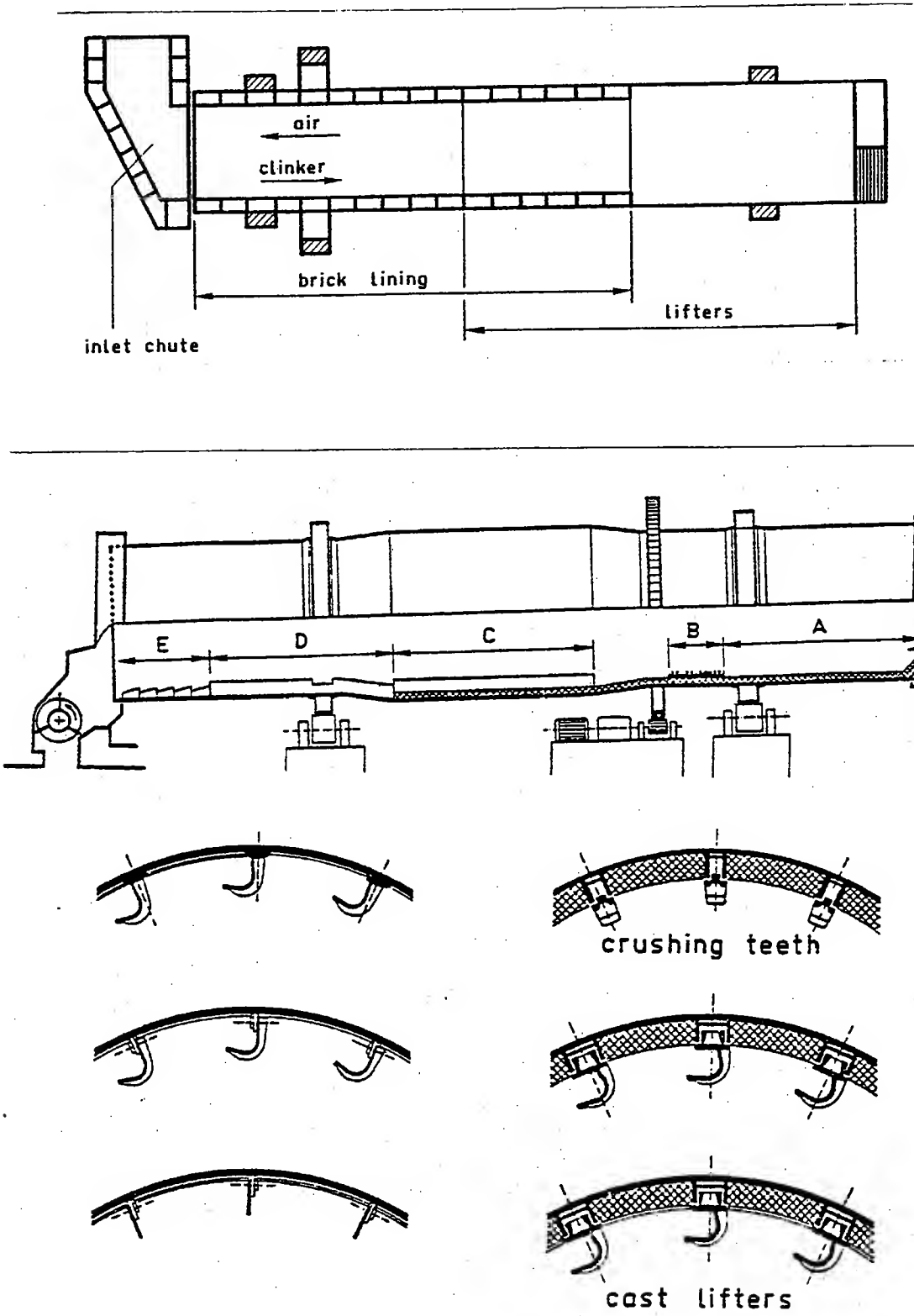
Figure 25 Rotary cooler



The following zones can typically be distinguished in a rotary cooler (simplified):

A Lined inlet zone

B Lined crushing teeth zone
(metallic teeth)

C Lined cast lifter zone, lining protected by wearing plates
(at least in the second half)

D Cast lifter zone, shell protected by wearing plates
(having air gap, giving also insulating effect)

E Sheet metal zone with wearing plates

Construction materials have to be selected according to the high temperature and wear requirements.

4.1.3 Cooling performance

Depending on the design and the shape of the lifters clinker outlet temperature usually tends to be high. In many cases it is necessary to enhance the cooling by injecting water into the tube (up to 60 g/kg clinker) in order to reach reasonably low clinker temperatures of 100° to 150°C.

The cooling efficiency (heat recuperation) is equal or even slightly better than on a planetary cooler.

4.1.4 Strengths / Weaknesses

Strengths	Weaknesses
• Simplicity of cooler design, robust piece of equipment. • No special mechanical problems comparable to a rotary kiln. • No control loops. • Easy commissioning. • No waste air and therefore no dedusting equipment required • Electrical energy consumption up to 5 kWh/t lower compared to grate cooler. • Rotational speed can be adjusted and therefore upset kiln conditions can be handled easier than with a planetary cooler. • Suitable for AS type precalcining system tertiary (extraction of hot air is possible).	• Not recommended for large units (above 2000 t/d) • Formation of build-ups ("snowmen") in the inlet chute. A water-cooled chute or a dislodging device is required in such case. • Clinker outlet temperatures tend to be high and therefore water injection is usually required. • Due to large falling height wear protection in the tube must be reinforced (compared to a planetary cooler). • High kiln foundations are required. • Cooler inlet seal can contribute to additional false air inlet.

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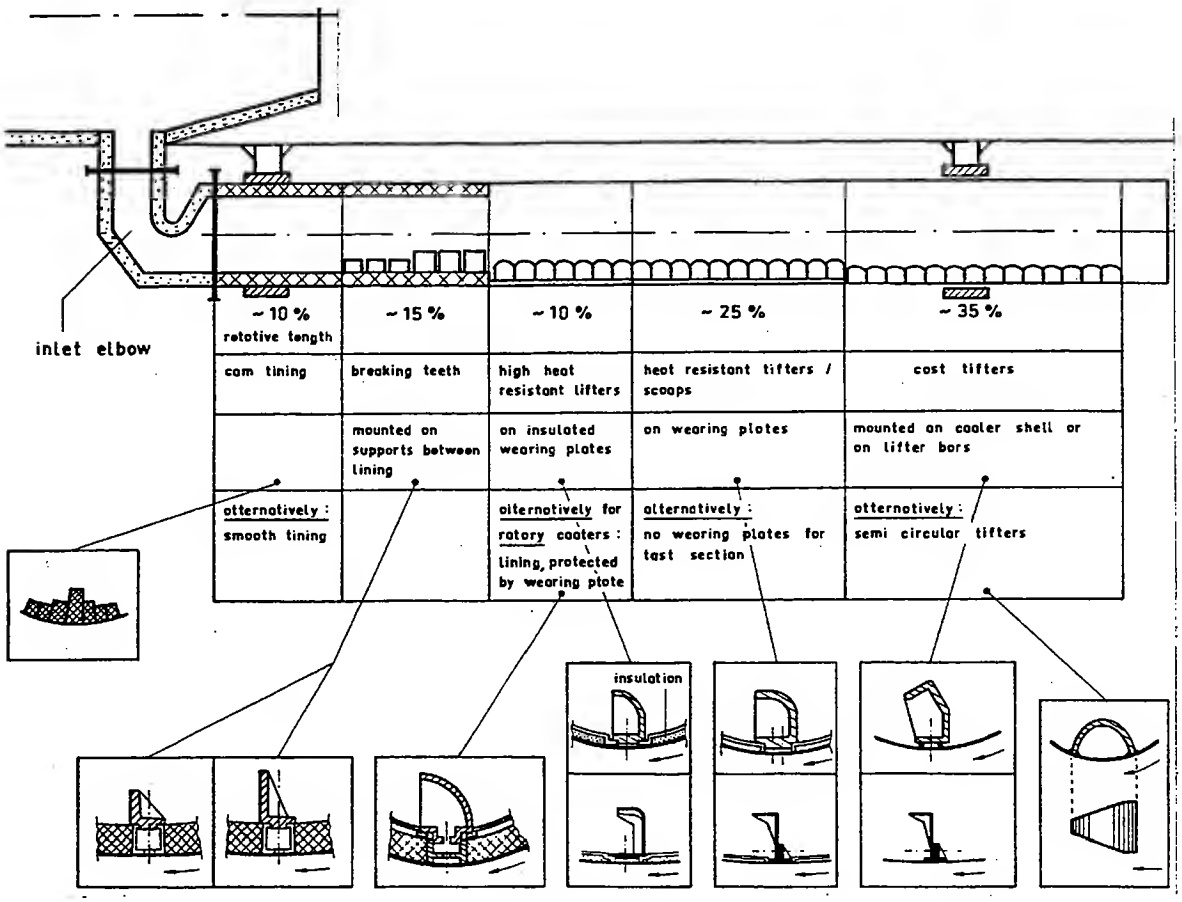
Figure 26 Internal transfer equipment for rotary and planetary coolers



4.2 The Planetary Cooler

4.2.1 Principle

The planetary cooler is based on the same cooling principle as the rotary cooler in the preceding chapter. However, the essential difference of a planetary cooler is the number of individual cooling tubes. The flow of clinker is subdivided into 9 to 11 (usually 10) cooling tubes which are installed around the kiln circumference at the kiln outlet (see Fig. 15). Therefore the planetary tubes follow the kiln rotation. Because of their connection to the kiln rotation, planetary coolers do not need a separate drive. This fact already illustrates one main advantage of the planetary cooler: its simplicity in operation.

Strictly speaking the cooling of clinker does not only start in the cooling tubes but already in the kiln. In the case of a planetary cooler the kiln burner pipe is always inserted into the rotary kiln so that a cooling zone behind the flame of 1.5 to 2.5 kiln diameters is created. This zone is called the "kiln internal cooling" zone and must be considered as an integral part of any planetary cooler. In this zone the temperature of the clinker drops from 1450° to 1200 - 1300°C. This temperature reduction is important for the protection of the inlet opening, the elbow and the first section of the cooling tubes.

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After this first cooling in the kiln internal cooling zone the clinker falls into the elbows when they reach their lowest point of kiln rotation. The hot clinker is then cooled by air in counterflow (the amount of air equals the amount of secondary air). The air is heated up to approx. 700°C. The clinker reaches final temperatures which are typically in the range of 140° to 240°C.

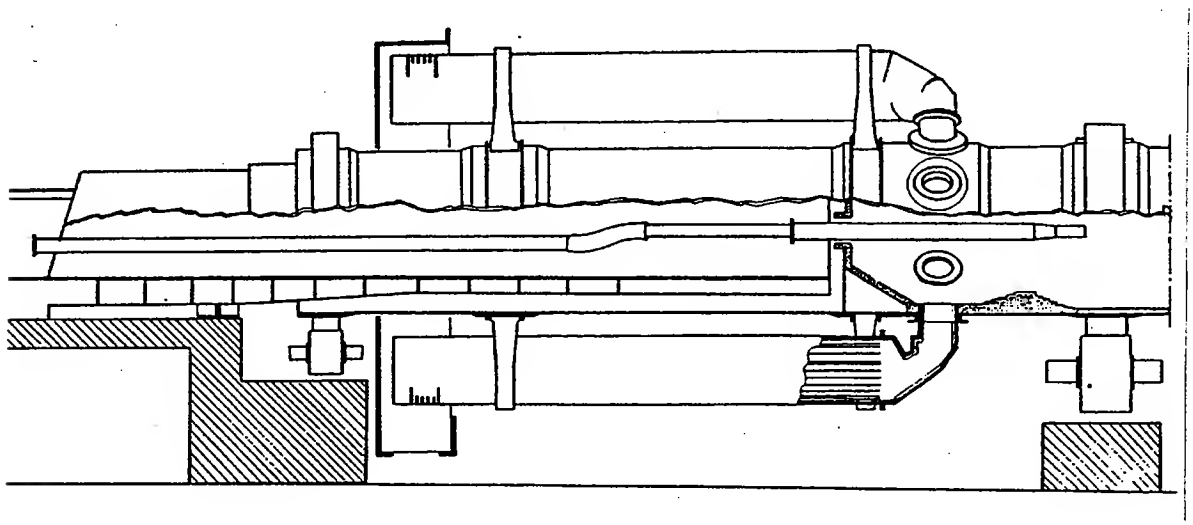
A considerable amount of heat is also transferred to ambient by radiation and convection since approx. 75% of the cooler shell is not insulated.

4.2.2 Historical

Planetary coolers have been used since 1920. When large kiln units and grate coolers were developed planetary coolers were abandoned for many years. But about 1966 planetary coolers of large capacities were introduced. At that stage serious mechanical problems occurred on these first large planetary coolers. As a consequence a lot of work had to be done in order to improve the mechanical design of planetary coolers. As a result of extensive computer calculations and operating experience the planetary cooler became a mechanically reliable piece of equipment.

In the late 1970's, the design had reached a high standard and a considerable level of perfection. Units of up to 5000 t/d were envisaged. With the demand for permanently larger units using precalciner technology with separate tertiary air dusts, the boom period of the planetary coolers came to an end.

Figure 27 Planetary cooler



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4.2.3 Design features

Planetary coolers in the late 1970's had the following design features:

- **Shell extension:**
  The kiln shell is extended beyond the cooling tube outlets and is supported by an additional roller station.
- **Fixation of cooling tubes:**
  Fixed support of cooling tubes near inlet and loose support near outlet end.

With larger coolers, the cooling tubes can consist of two separate sections requiring three supports. In that case two fixed supports are located near inlet and near outlet and a loose support is located at the interconnection point in the middle.

- **Design of cooler supports:**
  The kiln shell is reinforced (high thickness) where the cooler support structure for the cooler is welded on. The support structure (base and brackets) itself is of heavy design consisting of reinforcement ribs and box beams.
- **Cooler length:**
  Length/diameter ratio of tubes is approx. 10:1
- **Inlet openings:**
  The inlet openings to the cooler elbows weaken the kiln shell and high mechanical and thermal stresses occur in that zone.
  The openings are made of oval shape and the kiln shell is considerably reinforced in its thickness (up to 140 mm in large kilns) in order to compensate for the weakening.
  In some cases a diagonal retaining bar (made of high heat resistant steel) is incorporated in the opening in order to avoid that large lumps can enter the cooler.
- **Kiln-to-elbow joint:**
  This joint is designed in a manner that no forces due to thermal expansion and deformation are transmitted from elbow to kiln.
- **Elbow:**
  In order to prevent that clinker is falling back into the kiln while the opening is on top position, the position of the cooling tube is displaced back against the direction of rotation. The elbow design must avoid excessive dust backspillage and wear.

4.2.4 Internal heat transfer equipment (see Fig. 26)

Cooling performance depends strongly on efficient lifters of solid and durable design. Since high heat resistant metallic lifers are available on the market also the high temperature zones can be adequately equipped. Special high temperature alloys can be used for this purpose. They can withstand maximum temperatures of up to 1150°C. These alloys are usually characterized by a high chromium content of approx. 30% Cr. Other elements as Ni or Mo can occur in various proportions. Fig. 26 shows a typical arrangement of heat transfer internals. Breaking teeth are applied in the hottest zone. They are able to crush large lumps of clinker and create also a tumbling effect, which improves the heat transfer. They are of heavy design and mounted on separate supports.

The first rows of lifters must be carefully selected regarding design and material. Their functioning is very important since they also protect the following lifters from overheating.

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Figure 28a Temperature profile in planetary cooler

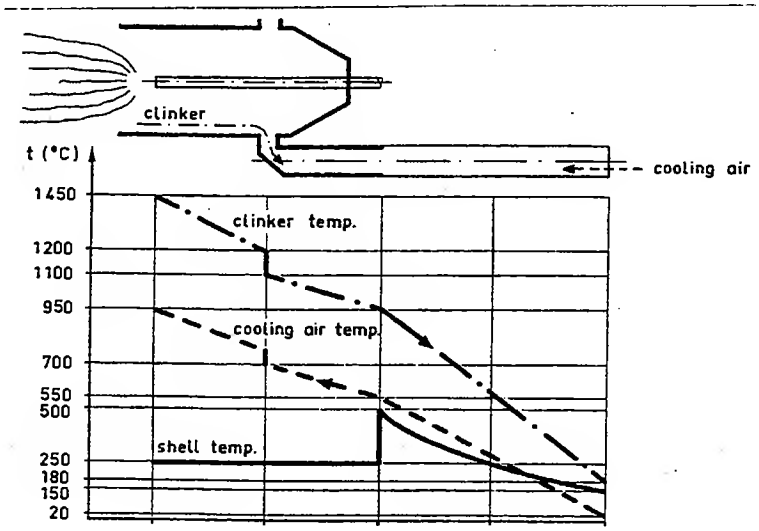
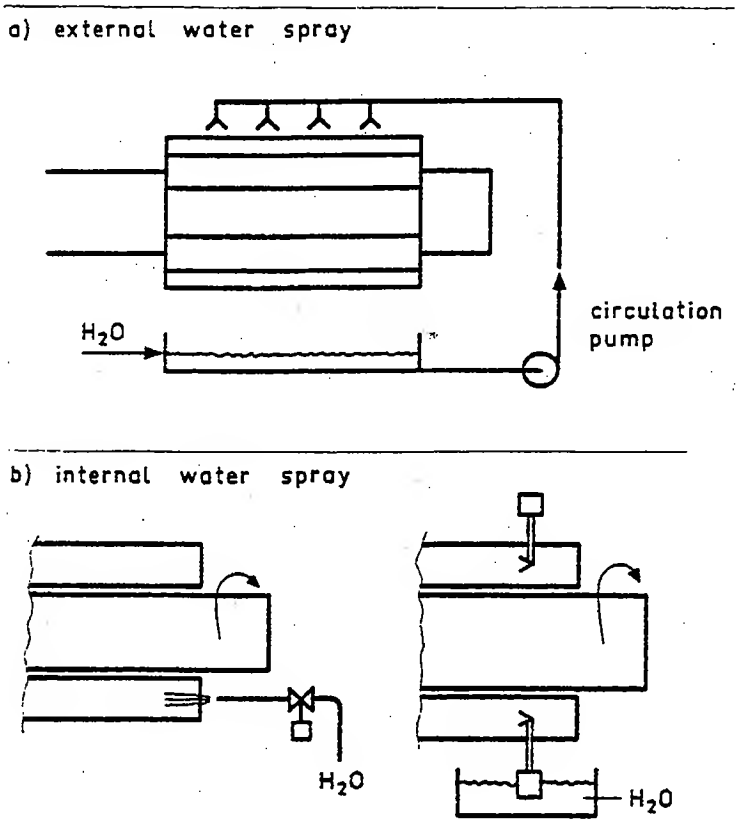


Figure 28b Water cooling for planetary coolers



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5. VERTICAL COOLERS

5.1 The Gravity Cooler (G - Cooler)

The Claudius Peters Company have developed the "g-cooler". The letter "g" stands for gravity since clinker movement is performed by gravity.

This cooler is designed as an after cooler and can therefore only be used in connection with a primary cooler such as a short grate cooler or a planetary cooler. The installation together with a grate cooler is shown in Fig. 29.

An intermediary crusher reduces the clinker size to 20 - 30 mm. The material of approx. 400°C is then filled by a drag chain into a vertical shaft. Cooling is performed by horizontal rows of tubes which are cooled by internal air flow. The heat is therefore exchanged indirectly and the air remains dust-free. The clinker slowly drops down (at a speed of 20 – 30 mm/s) and reaches final temperatures of approx. 100°C at the discharge.

There is no dedusting equipment required for the cooling air. However, the system according to Fig. 29 as a whole is usually not free from dusty waste air. In case of a suspension preheater kiln system there is still some waste air required on the grate cooler since the kiln cannot take all the hot air produced during the first cooling step. In addition, a marginal amount of dusty air is produced by the g-cooler itself (top and discharge).

The application of this cooler type is often considered for kiln extension projects. If an existing grate cooler (or a planetary cooler) has to be operated at higher capacity the new clinker outlet temperature can become too high. In this case the clinker temperature can be reduced by a g-cooler used as an aftercooler.

5.2 The Shaft Cooler

A shaft cooler can be operated waste-air-free and theoretically offers an ideal countercurrent heat exchange and thus high recuperating efficiency. Based on the idea the first large shaft cooler was designed and constructed on a 3000 t/d kiln in 1973.

The experience gained in the plant shows that it is possible to operate such equipment but some serious disadvantages have to be taken into account:

- All depends of the clinker granulometry! Theoretically, an extremely uniform clinker granulometry having no fines and no coarse material would be required. This is hardly achievable in a cement kiln. Therefore, fluctuations occur.
- High cooling air quantity (= secondary air) of 1.05 Nm^3/kg cli is required but even so the clinker exit temperature of 350°C is very high.
- High power consumption (10 kWh/t)

For the above reasons, the technical realization is not yet solved. The shaft cooler so far is not a reasonable alternative to the conventional clinker coolers.

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Figure 29 Gravity cooler (g-cooler, CPAG)

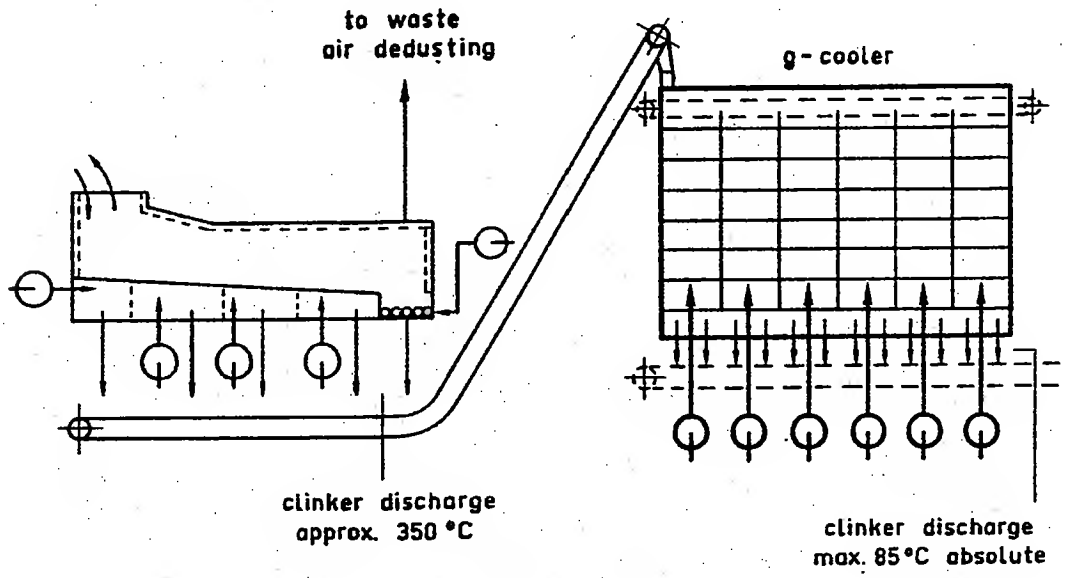

Figure 30 Shaft cooler

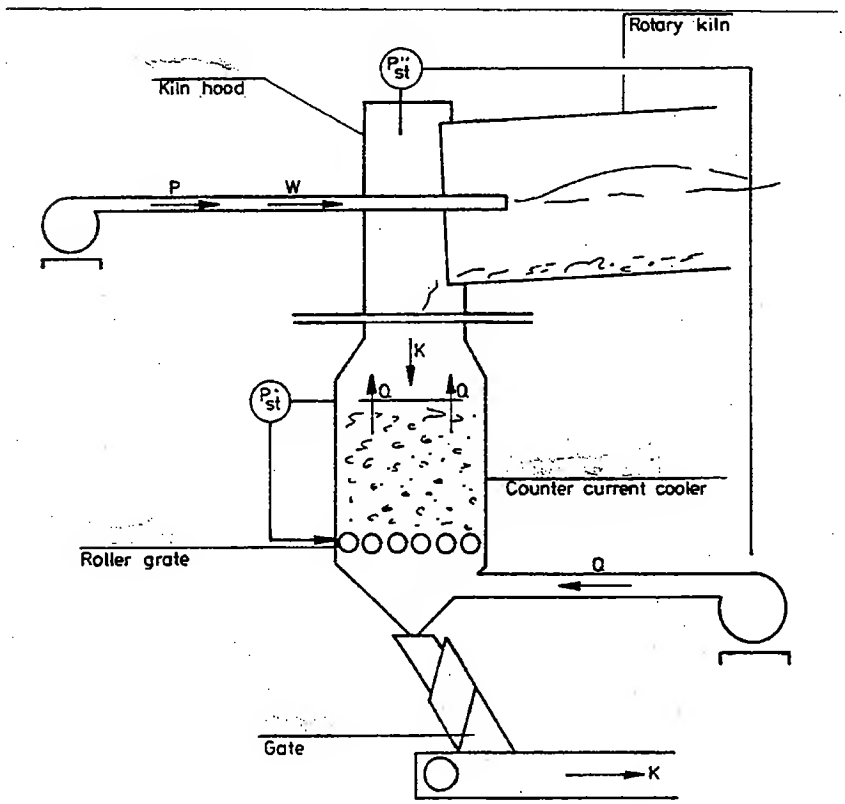

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Figure 31 Claudius Peters CPAG: Combi Cooler

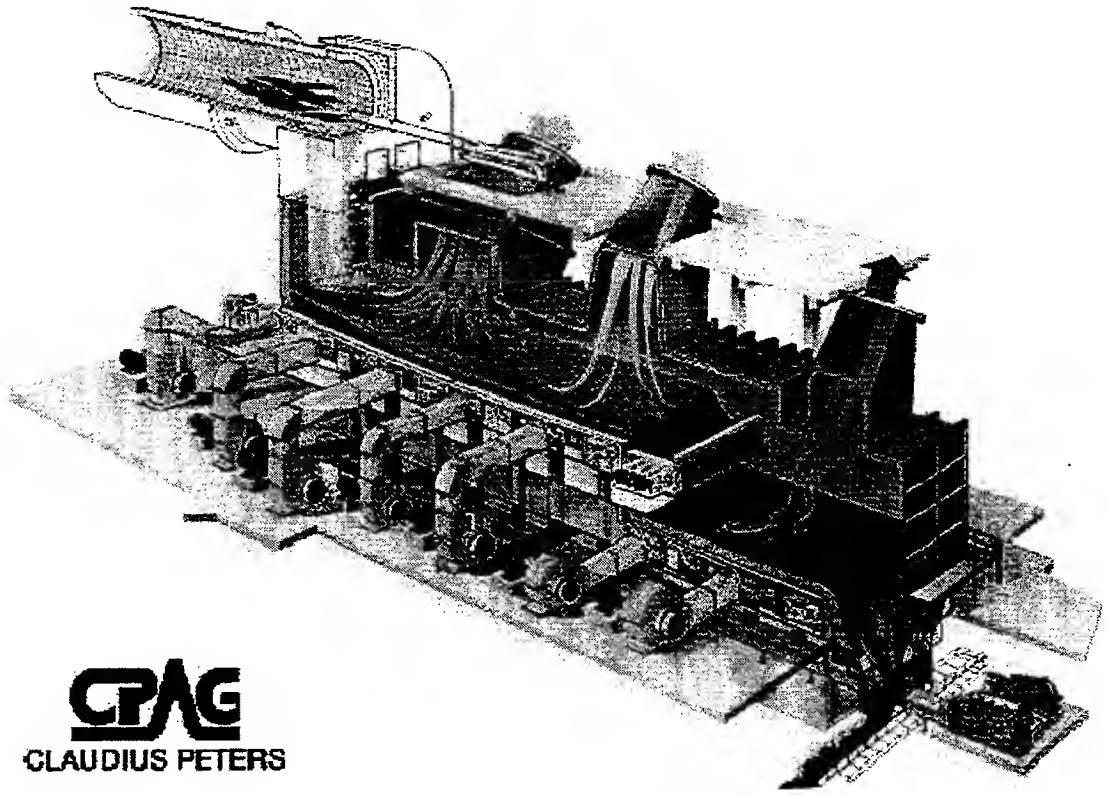

Figure 32 FLS: Coolax Grate Cooler

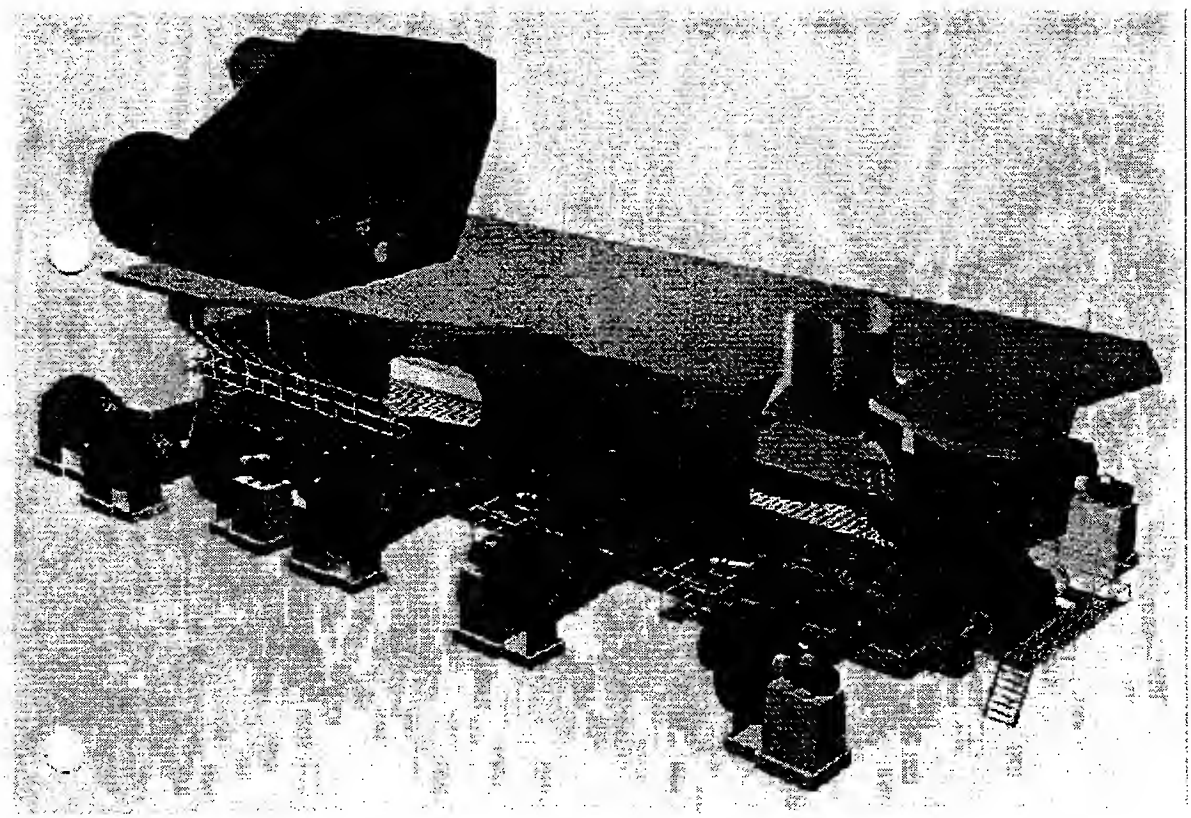

Figure 33 Fuller: Controlled Flow Grate (CFG) Cooler

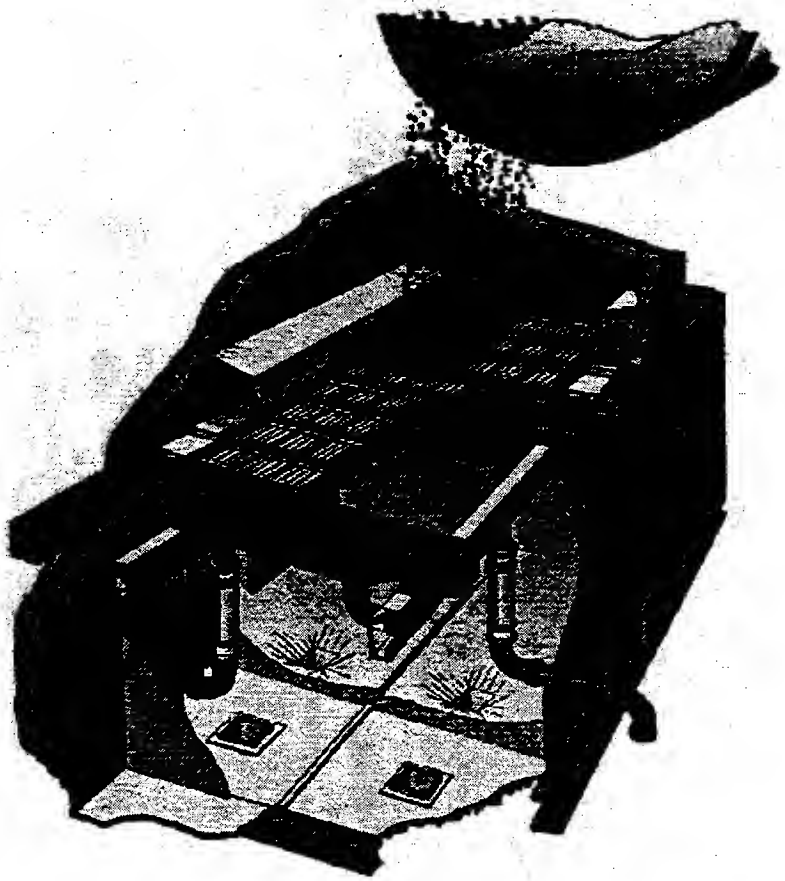

Figure 34 IKN: Pendulum Cooler

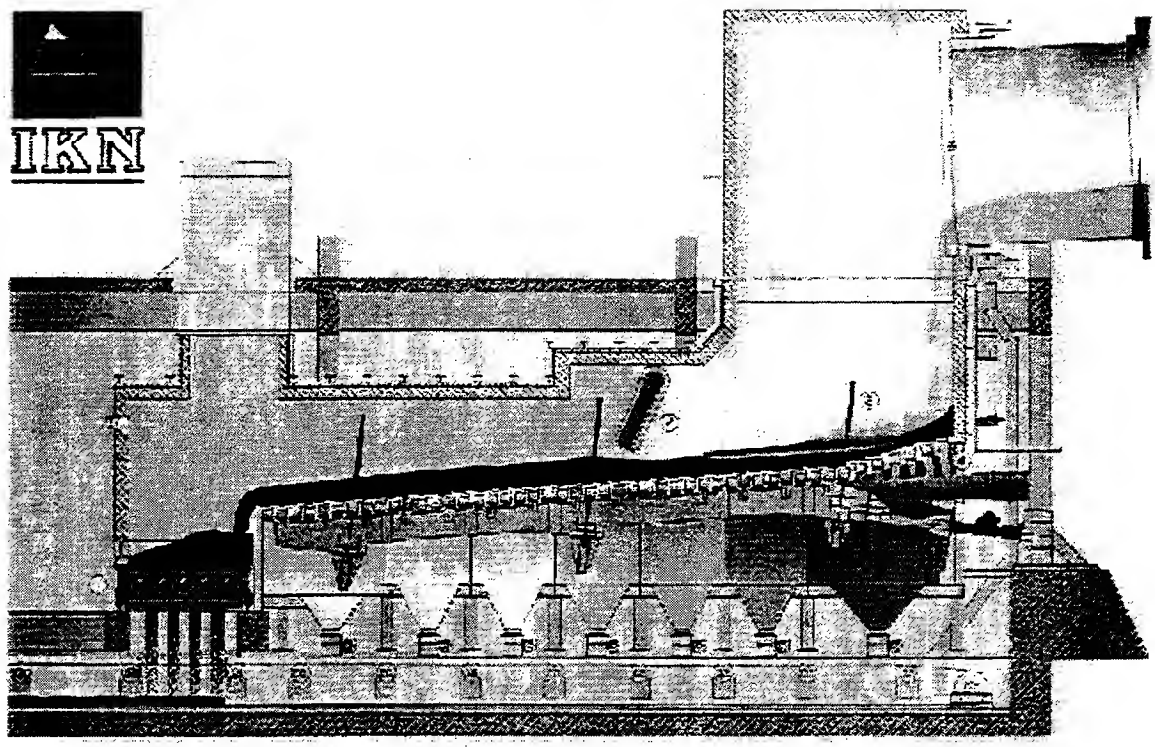

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Figure 35 KHD: Pyrostep Cooler

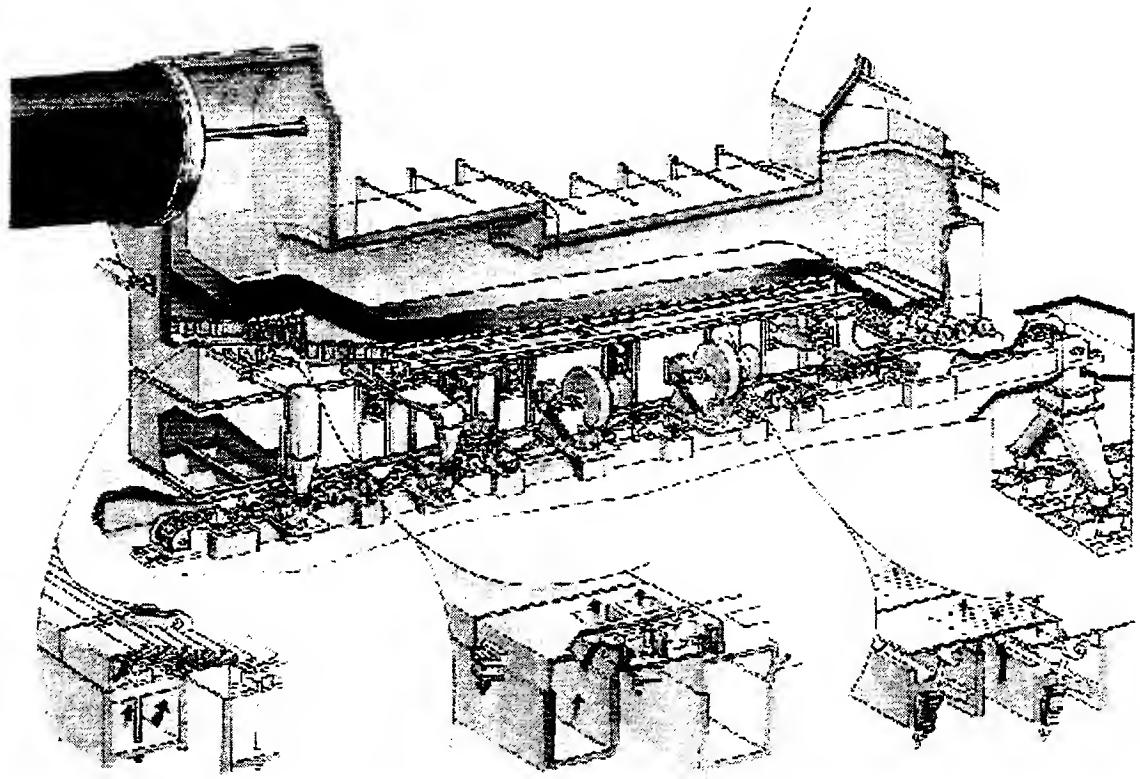
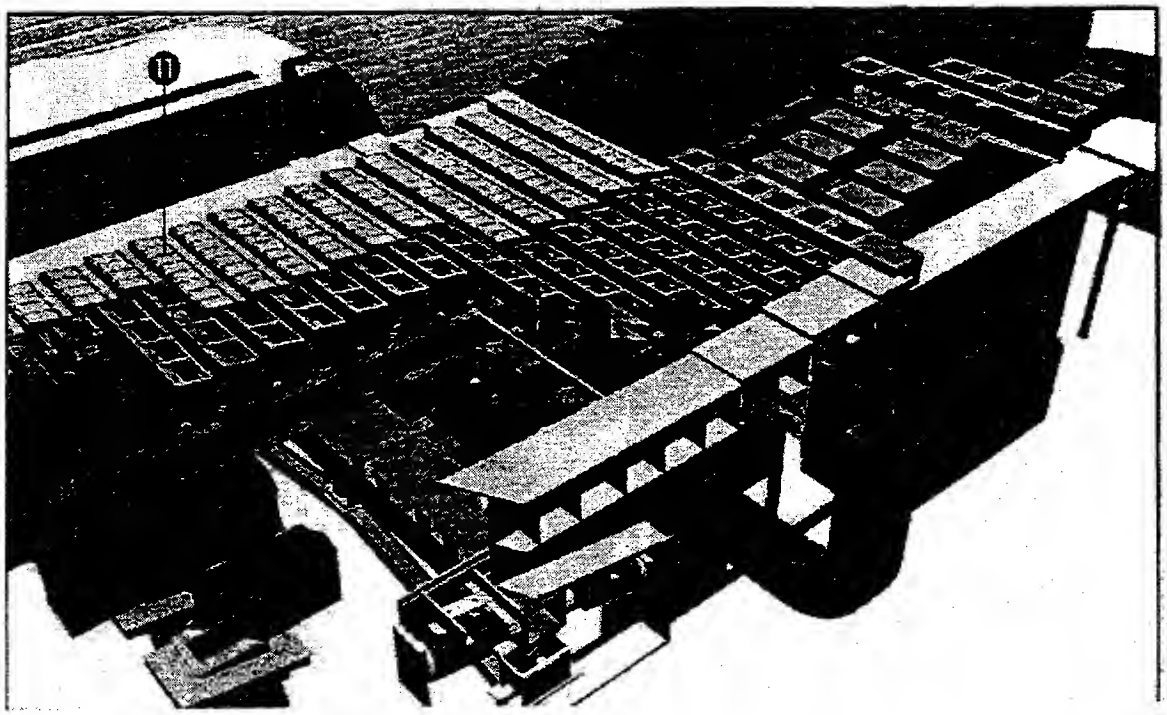


Figure 36 Polysius: Repol RS Cooler



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Chapter 2

Internal Kiln Fittings

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Kiln Chain Systems

A. Obrist
PT 96/ 14036 / E

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Summary:

A kiln chain system has four main functions:

- It helps to increase the heat exchange between gas and raw meal
- It keeps the kiln shell (lining surface) clean
- It assists the transport of material through the kiln tube
- It helps to reduce the dust emission

A properly designed chain system must respect the changing properties of material passing through the kiln tube. In a wet process kiln the material is fed as a liquid slurry and changes it properties subsequently in several steps inside the chain system to dry preheated granules. In accordance with the changing material properties different arrangement of chains (straight curtains, spirals, garlands, etc.) have to be used for individual parts of the system to satisfy the specific requirements. Also the chain densities and the height of the free tunnel below the chains have to be selected carefully in order to reach the maximum efficiency.

The chain links can have different shapes (round, long, oval etc.), preferably round links. The chemical composition of the chains' alloy and its physical treatment (hardening) strongly influence the life time of the system.

Different types of chain hangers can be used (single or multiple hangers, with or without shackles etc.). They have to guarantee a sufficient stability, to enable an easy installation and they should as far as possible assist the function of the chains.

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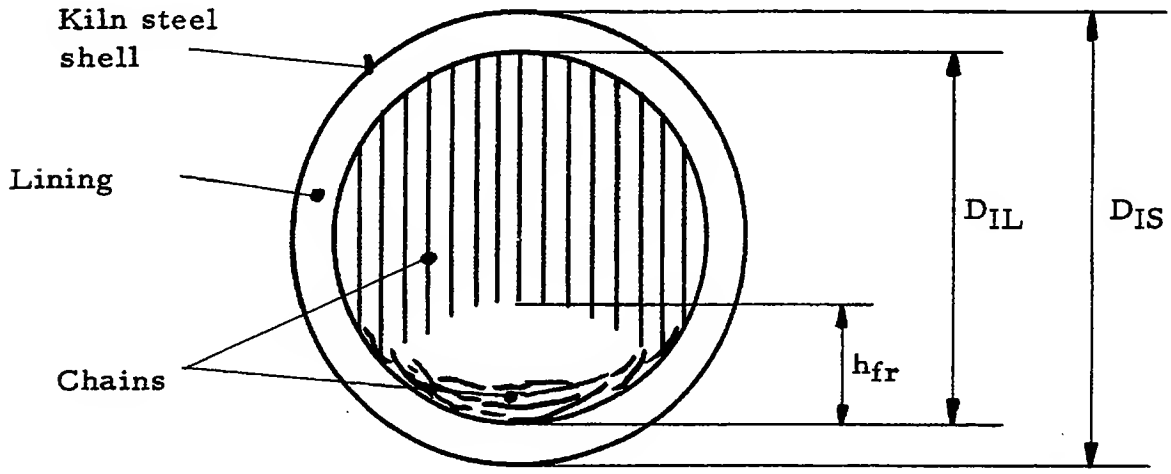
NOMENCLATURE

Just a few symbols and names are to be explained before starting this lecture, the other ones will be explained in the respective chapters.

Figure:



D_{is}	Diameter inside kiln steel shell
D_{IL}	Diameter inside kiln lining
h_{fr}	Theoretical free height under the chains (see attached sketch), expressed in mm or as % of D_{IL}
density of chains m^2/m^3	is calculated for individual parts (zones) of the system as the total surface area of chains in the respective zone divided by the volume inside lining of this zone
density of chains kg/m^3	similar to the above mentioned density, but concerns the weight of chains instead of their surface

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1. INTRODUCTION

Wet process kilns cannot be successfully operated without internal kiln fittings, among which the kiln chains are the most typical and most frequently used ones. The number of existing wet process kilns is still high (~33% in "Holderbank" Group) and a conversion from wet to dry process is very expensive. By improving the existing chain systems or, where necessary, by installing a completely new chain system, the kiln operation can be upgraded considerably with relatively moderate investment costs.

2. FUNCTIONS OF A KILN CHAIN SYSTEM

The kiln chain system has 4 main functions:

2.1 Heat Exchange

The heat exchange between hot gases and the raw material depends on the surface area exposed to the hot medium. In the parts of kiln where no chains are installed, this surface area consists of the surface of the material layer on the kiln bottom and of the surface of the remaining part of the kiln shell (resp. lining). By installing the chains a large additional surface area can be gained, exceeding that one mentioned above several times (up to 10 times and more) in the respective part of the kiln. By improving the heat exchange the specific heat demand is reduced and the kiln output is increased.

In Fig. 1 different positions of a chain during one kiln rotation are shown. In position 1 the chain is exposed to the stream of hot gases and thus heated up. The cooling of the chain (which passes its heat to the layer of material) starts in position 2, continues in position 3 and ends in position 4.

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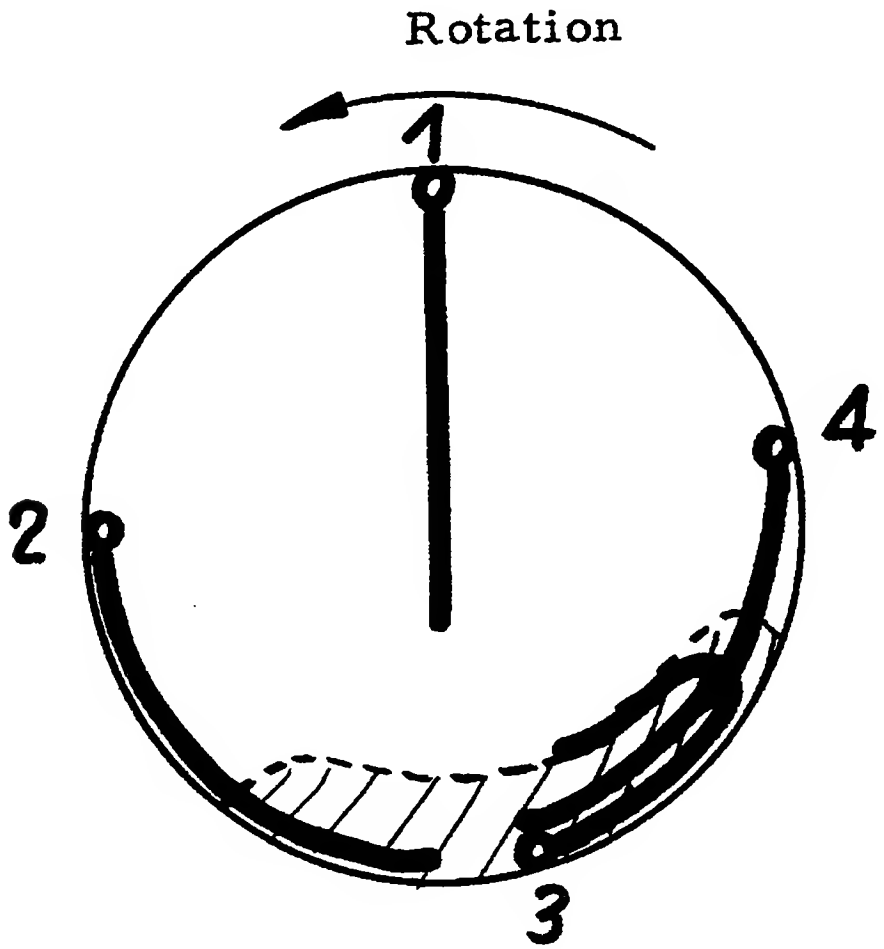
Figure 1:



2.2 Cleaning of the Kiln Shell

In the upper part of the kiln the characteristics of the wet, sticky raw material favors the formation of mud coating and mud rings. This would reduce the free kiln cross sectional area and thus obstruct the flow of material and gases. Growing mud rings make the kiln operation difficult. It is one of the main functions of the chain system to keep the internal kiln shell surface clean, free of coating or rings. Due to the kiln rotation the chains slide on the kiln shell (resp. lining) and destroy the rings and the coating. The sliding movement of a chain cleaning the kiln shell is shown in Fig. 1 (position 3).

2.3 Transport of Material

The properties of material in different parts of the kiln differ considerably. In some sections of the upper part of the kiln, where the material is sticky and plastic, its transporting is more difficult than in other sections.

As a regular flow of material is of an eminent importance for a smooth kiln operation, it is necessary to install material flow assisting devices in some sections. Some special arrangements of chains can help to draw the material through the critical sections. This can be achieved by chains moving in the desired direction (garlands) or by a screw shaped arrangement of the chain fastening points. Other arrangements of chains can be an obstruction to the flow of material and should therefore never be used in the critical sections.

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2.4 Reduce Dust Emissions

The gases leaving the kiln contain a certain amount of dust consisting mainly of partly calcined, hot raw material. The dust load of gases depends on the properties of the raw material and on the specific conditions of the kiln operation. Dust loss should be kept small, it means a loss of heat and material. The kiln chain system, mainly its upper part, can help to reduce the dust emission. Dust particles carried by the stream of gases stick to the wet surface of chains and later when these chains are emerged into the layer of material, this dust is passed over to the slurry.

3. INDIVIDUAL ZONES OF A CHAIN SYSTEM

The material passing the chain system changes subsequently its properties - it loses water and is heated up. According to the different material properties the total chain system can be divided into several zones. These zones are:

3.1 Free Zone of the Kiln Inlet

This short zone is considered to be a part of the chain system in spite of the fact that no chains are installed here. A sufficient amount of slurry should be accumulated in this zone in order to guarantee a constant and regular flow into the lower parts of the system.

Good results have been obtained with the zone length of 1 to 1.5 kiln diameters.

3.2 Dust Curtain Zone

The dust curtain zone is relatively short, its length does not exceed 0.5 D_{IL} under normal conditions.

The material entering this zone still has the relatively good flow properties of the kiln feed (slurry). When leaving this zone, the material has a lower water content and becomes more "plastic", essentially due to the inter-mixing of the dust previously retained by the chains in this zone.

In order to achieve a good dust catching efficiency, the density of chains must be high (some 8 to 15 m^2/m^3) and the free height below the chains should be 18 - 27% of D_{IL}.

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3.3 Plastic Zone

The length of this zone depends on properties of raw material, slurry moisture, characteristics of the kiln operation etc. and can vary in a wide range (approx. between 1.5 and 4 D_{IL}). The material in this zone is plastic and sticky, still relatively cold and wet and because of these properties it favors mud coating and mud ring formation. The transport of material through this zone is the most difficult one among all the zones of the chain system.

Due to the material properties mentioned above the chains in this zone must have a good shell cleaning and material transporting efficiency. The density of chains should be relatively low, some 5 to 8 m^2/m^3. As to achieve a big free tunnel under the chains, the free height h_{fr} should be approx. 30% or, if garland chains are installed in this zone, some 40%. Heavier (thick wire) chains should be installed.

In order to be sure that the zone of plastic material will always stay inside the zone of chains which can treat it successfully, the respective arrangement of chains should be slightly extended in the downstream direction as to obtain a sufficient safety.

3.4 Granular Zone (Preheating Zone)

The recommendable length of this zone depends on the desired material temperature and the rest water content at its discharge end. Good results have been achieved with a zone length between 2 and 4,5 D_{IL}.

The material entering this zone is not plastic any more, it forms granules which are easy to be transported and do not favor a mud ring formation. The granules should be dried and heated up in this zone.

The chains should heat the material gently without unnecessary dust generation, they should enable a good heat exchange. Lighter (thin wire) chains should therefore be installed. A chain density of approx. 6 to 10 m^2/m^3 and a free height of approx. 25 to 30% can be recommended. This zone is sometimes divided into an upper and a lower part. Both parts have the same (or at least a similar) arrangement, but the lower part has a higher density of chains than the upper one. The damming effect of the lower part, caused by the thicker layer of chains on the kiln bottom, helps to increase the material retention time and improves the heat exchange.

3.5 Heat Resistant Zone

This zone is relatively short, its length does not exceed 1,5 D_{IL}. The material, dry and hot granules, can easily be transported. A very gentle treatment of the material is required in order to keep the dust creation as low as possible.

The main function of the chains is to protect the upstream part of the system against heat radiation and too high a gas temperature. Chains made of heat resistant steel should be installed in this zone. Lighter (thin wire) chains should be preferred.

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3.6 Main Characteristic Data of the Individual Chain System Zones

Table 1

		Zone					
		Free	Dust Curtain	Plastic	Granular Upper	Granular Lower	Heat Resistant
Length	D_{IL}	≤ 1,5	≤ 0.5	1.5 to 4	1 to 3	≤ 1,5	≤ 1,5
h_{fr}	% D_{IL}		18 - 27	40 / 30	25 to 30		
Density	m^2/m^3		8 to 15	5 to 8	6 to 10		
Moisture	%	30 to 40		15 - 25			
Material temp.	°C	20		100			<600
Chain temp.	°C			<400			<1'000
Gas temp.	°C	160 to 240					<1'100
		→ Material flow					

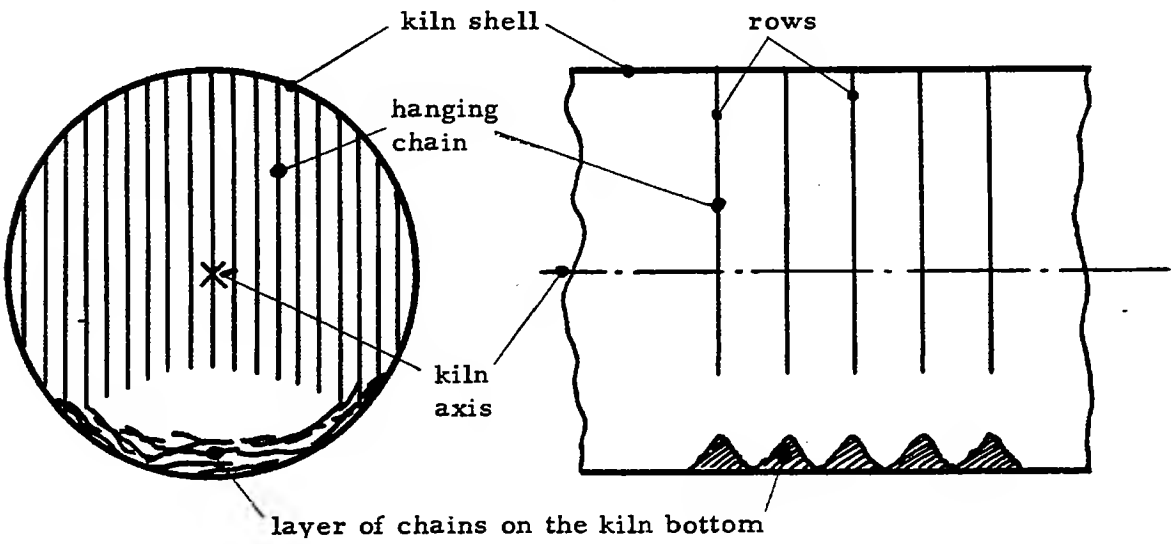
4. ARRANGEMENT OF CHAINS

Each individual zone of a chain system must have its own specific properties in order to satisfy the requirements mentioned previously.

4.1 Straight Curtain

As shown in Fig. 2, this arrangement of chains is very simple. The chain fastening points form a ring. Several straight curtains are combined to a straight curtain zone. The distance between the individual straight curtains (rings or rows) should not be too long, this would lead - in order to achieve a sufficient density - to too great a number of chains per ring and thus to big heaps of chains on the kiln bottom obstructing the flow of material.

Figure 2: Straight Curtain



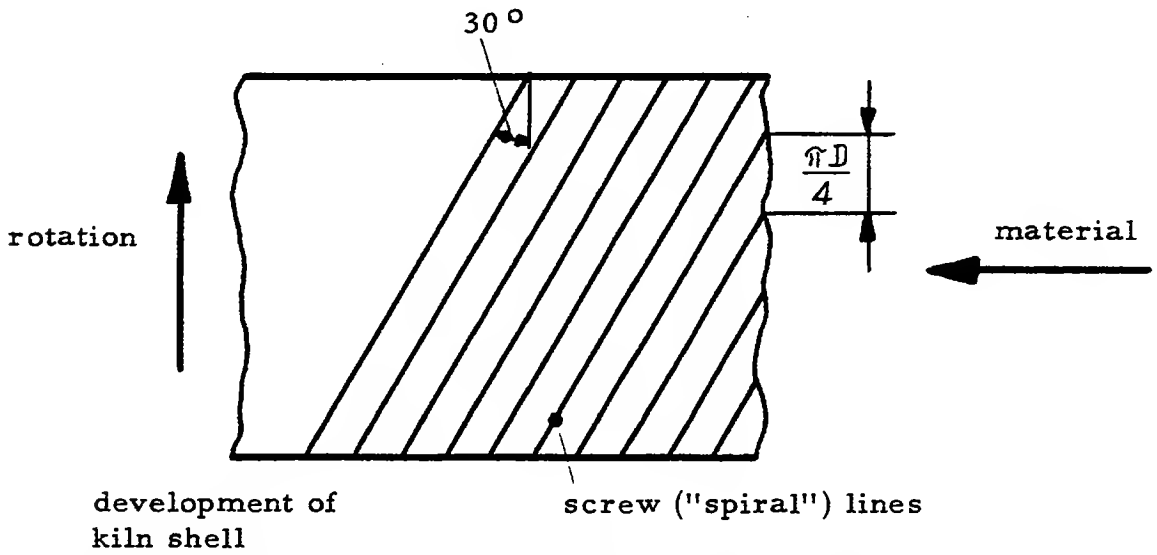
The main advantages of a straight curtain are its simple design and installation as well as an easy maintenance. Its main disadvantages are the poor shell cleaning efficiency and the fact that this arrangement does not assist the transport of material. The straight curtains should therefore not be used in the plastic zone or in the dust curtain, but they can be recommended for the granular zone.

4.2 Spiral Zone

In a spiral curtain the chain fastening points follow the screw lines on the kiln shell.

The spiral (screw) lines should have an inclination of approx. 30° (see Fig. 3). In order to assist the transport of material, the inclination must have the proper sense, i.e. the sense of rotation must be taken into consideration.

Figure 3: Spiral Curtain (4-start spiral)



The arrangement with 4 screw lines per circumference, called a 4-start spiral, is most frequently used. This arrangement allows for a good shell cleaning effect and at the same time overlapping of chains can be avoided, as shown in Fig. 4a. The recommended chain length is namely approx. 0.7 D, and the circumferential distance between adjacent screw lines is π D/4 = 0.78 D.

The benefits of this solution become clear if we compare it with other arrangements having a different number of screw lines. An arrangement with less than 4 screw lines per circumference (Fig. 4b) does not enable a full shell cleaning effect, because some parts of the shell have no contact with a chain. An arrangement with more than 4 screw lines leads necessarily to overlapping of chains (Fig. 4c) and thus to a higher layer of chains on the kiln bottom which is not favorable for the transport of material.

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Figure 4a-c:

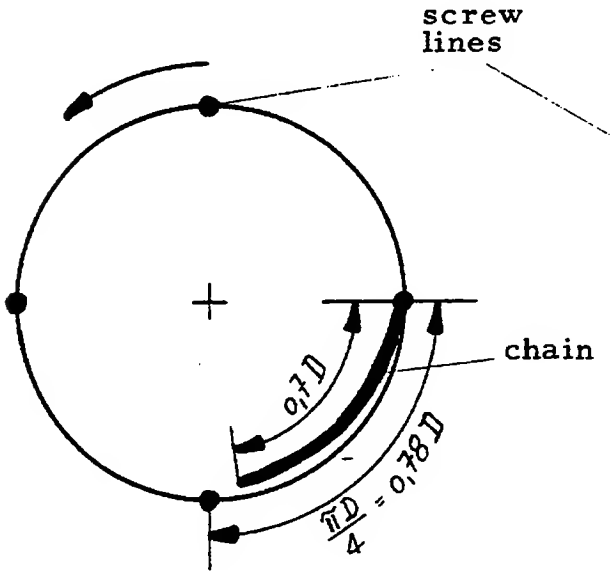

Fig. 4a

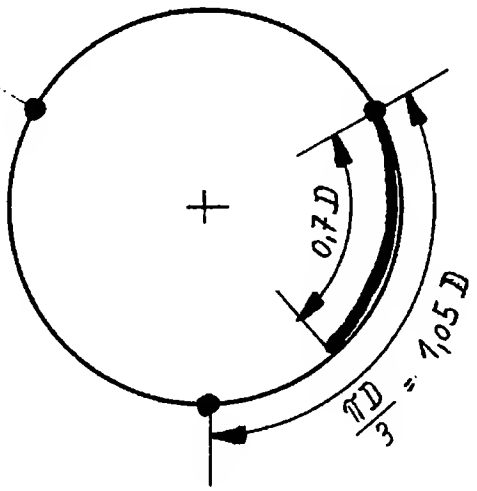

Fig. 4b

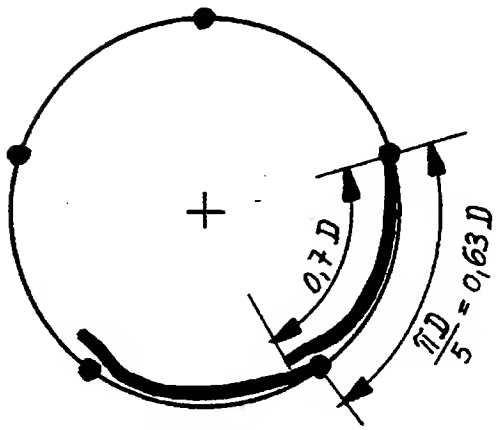

Fig. 4c

The very good material transporting and the good shell cleaning efficiency are the main advantages of a spiral curtain. Its disadvantages are a slightly smaller heat exchanging efficiency, a more difficult brick lining work and a limited maximum chain density. For these reasons, the spiral curtain arrangement should not be used in the granular zone but it can be recommended for the plastic zone.

4.3 Multiple Spiral Curtain

A spiral curtain having 8 or more spirals per circumference is called a multiple spiral curtain. Such an arrangement enables to achieve high chain densities and the passages between chains are narrow, therefore, it can be used in the dust curtain zone. its material transporting efficiency is not as good as that one of a 4-start spiral, but it is still better than that one of any other arrangement enabling a similarly high chain density. A multiple spiral arrangement can therefore be recommended for dust curtain zones in such cases where the kiln feed flow properties are poor.

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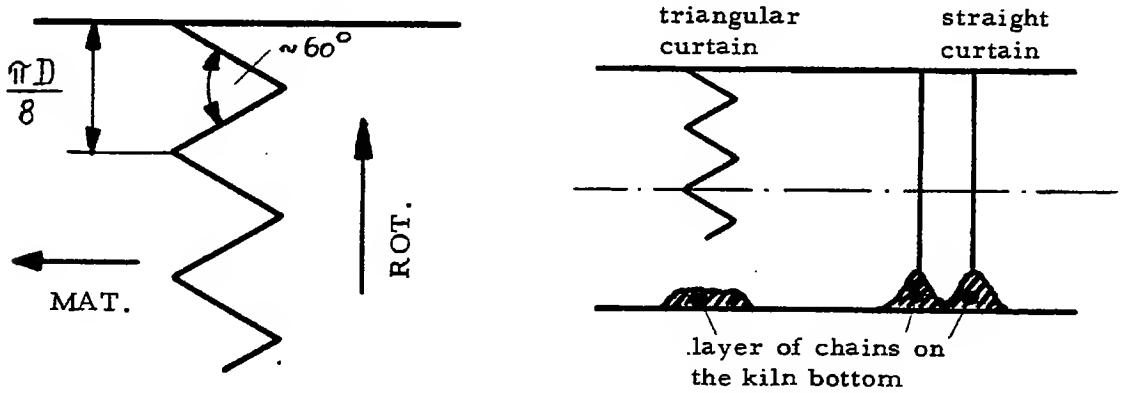
4.4 Triangular Curtain (Z-Curtain)

In a triangular curtain, also called Z-curtain, the chain fastening points follow a zig-zag line (see Fig. 5). This arrangement enables to reach high chain densities without obstructing the flow of material and gases too much. Because of the equal distribution of chains the heap of chains on the kiln bottom is not too high in spite of the high chain density (see Fig. 6).

Figure 5 & 6



The arrangement with angles of 60° and with 8 "triangles" per kiln circumference has proved to be the most successful one. In such an arrangement the layer of chains on the kiln bottom is only approx. 4 chains high (because 8 triangles have all together 16 fastening lines and the chain length of approx. 0.73 D corresponds to ≈ π D/4, therefore, each chain passes 16/4 = 4 lines).

The dust catching efficiency of such an arrangement is very good, it was therefore used in the dust curtain zone in such cases where the kiln feed flow properties are relatively good. Frequently one single triangular curtain was installed, but two are also possible.

Nowadays, for dust curtains a multiple start spiral arrangement is preferred to a triangular curtain because of its material transport efficiency.

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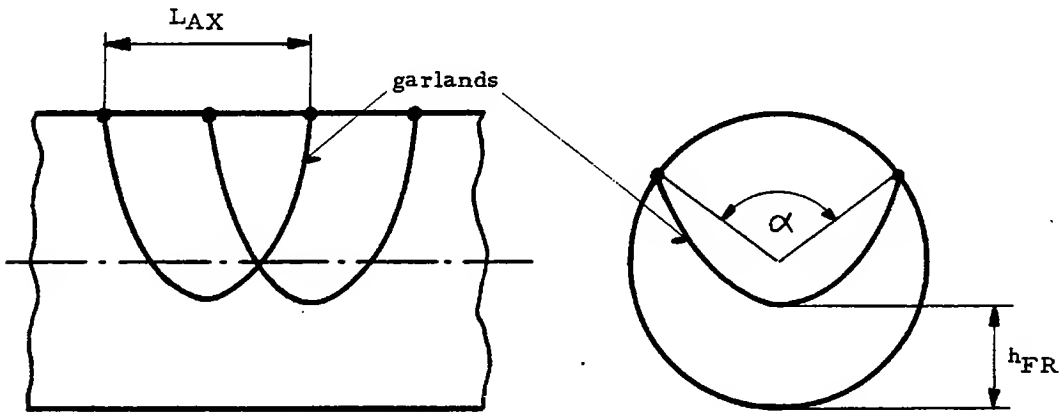
4.5 Garlands

Chains having their both ends fastened to the kiln shell are called garlands (Fig. 7) The fastening points form straight rings in the kiln shell. The shape of a garland is characterized by the chain length, by the axial distance L_{AX} and by the angle $\propto$ (see Fig. 7 and 8). Good results have been achieved with an angle $\propto \approx 90$ to 110° and a distance $L_{AX} \approx 0.5$ to 0.9 D.

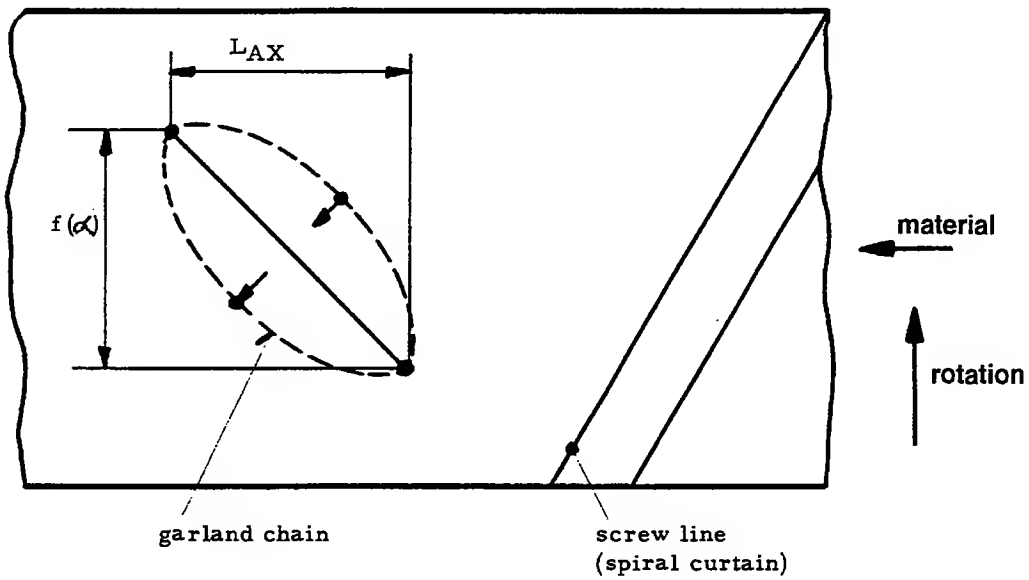
Figure 7 & 8:



longitudinal kiln cross section

Due to the sliding movement of the garland chains on the kiln shell (resp. lining) surface, their shell cleaning and material transporting efficiency is very good and their heat exchange efficiency is approx. 1.5 times higher than that one of pending chains (see Fig. 9).

Figure 9



As can be seen in Fig. 9 the garland chain should be hung in a "reverse" sense, i.e. in a sense differing from that one of a screw line, in order to assist the material transport.

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Due to the properties mentioned above, the garland chains have been used mainly in the plastic zone.

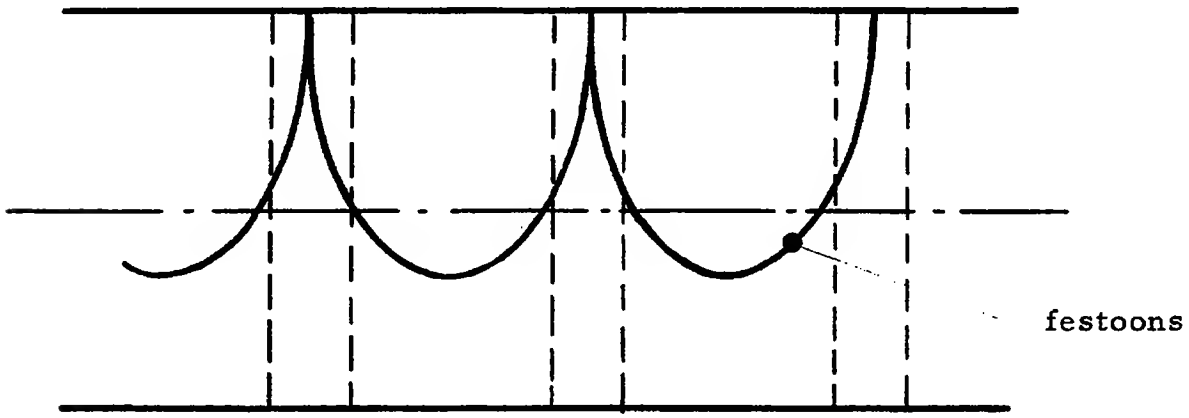
In spite of the advantages mentioned above CS/PT does usually not recommend the installation of garlands because of their disadvantages, namely:

- complicated installation
- difficult maintenance
- relatively short life time
- at the ends of the garland zone the shall cleaning efficiency is very poor (no movement of chain)

4.6 Festoons

Garlands without overlapping chains in the axial direction are called festoons (Fig. 10). The installation and maintenance of festoons are less difficult than that of garlands but their shell cleaning efficiency is poorer (the areas between the individual bays of festoons are not cleaned reliably).

Figure 10



4.7 Spiral Garlands

An arrangement of garlands where the chain fastening points form screw lines ("spirals") is called spiral garlands. Its material transporting efficiency is even better than that one of normal straight garland zone, but the erection and maintenance are more complicated.

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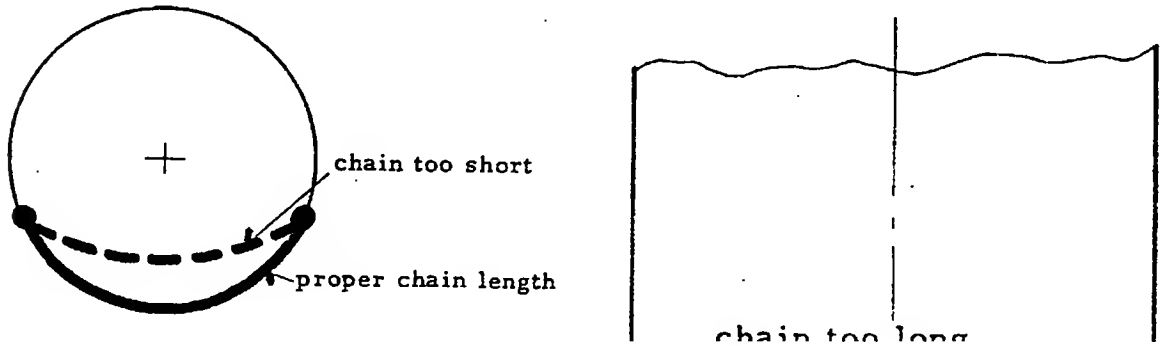
4.8 Thermochains

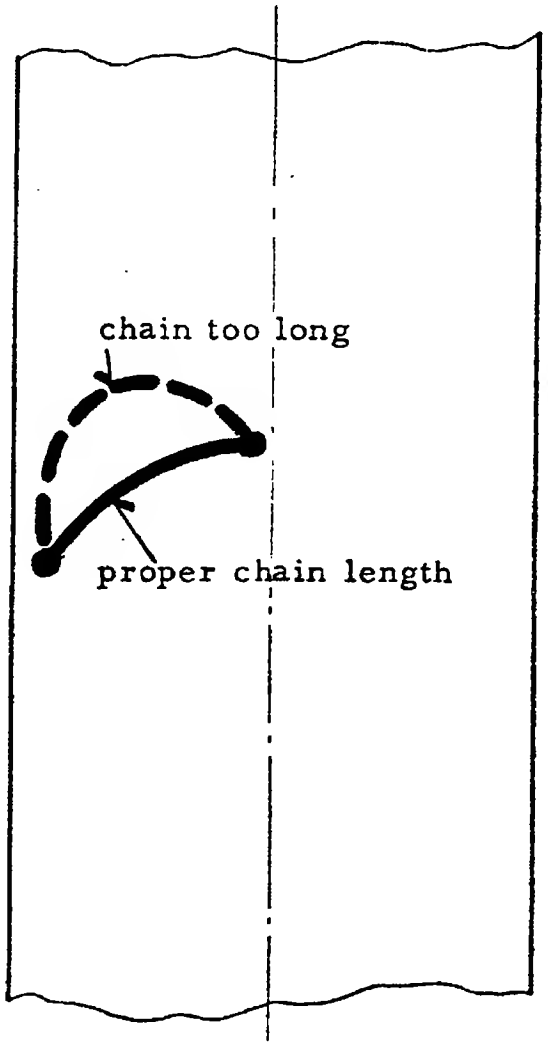
Thermochains are a special type of festoons, meeting the following conditions:

- The axial distance L_{AX} between the two fastening points is short (approx. 0.1 to 0.15 D_{IL}).
- The angle $\propto$ between the two fastening points is approx. between 60° and 120°, preferably 90° to 120°.
- The free height under the chain h_{fr} is approx. between 0.4 D_{IL} (for $\propto$ = 120°) and 0.6 D_{IL} (for $\propto$ = 90°).
- The chain (shackles included, if used) is approx. 1.1 times longer than the distance between the two fastening points, measured on the lining surface (see Fig. 11a and 11b).

A good heat transfer and at the same time a gentle treatment of material preventing an unnecessary dust creation are the main advantages of thermochains. Their disadvantages are a low shell cleaning ability and a very limited material transporting efficiency. Thermochains only have a limited sliding movement on the kiln lining compared to garlands and for this reason the sense of hanging (reverse or non-reverse) does not make too much difference. Thermochains cannot be used in the upper and central part of the chain system where the shell cleaning efficiency is of an eminent importance. They should be used in the lowest (hot) part of the system in such cases when another type of chain arrangement enabling the same heat exchange would lead to an excessively high dust emission.

Figure 11a & b:





development of kiln shell

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5. TYPES OF CHAIN LINKS

5.1 Round Links

The round links (Fig. 12a) can rotate slowly when kiln is in operation which has two advantages:

- the wear is distributed equally over the whole link circumference, and therefore, in comparison with other types, round links have a longer life time.
- due to the rotation of links their surface is kept clean which enables a good heat exchange (links are not surrounded by an insulating mud layer).

Because of the properties mentioned above round links should be preferred to other types mainly in those zones where the material is wet and sticky.

For calculations of the chain surface area and chain weight, the following formula can be applied:

- surface area of 1 link $O = \pi^2 S(s+d)$
- volume of 1 link $V = 0 \bullet \frac{s}{4}$
- weigt of 1 link $G = V \bullet \gamma \; (\gamma \approx 7.8\, t/m^3)$

The above formulas are valid for round chain links with a round wire cross section.

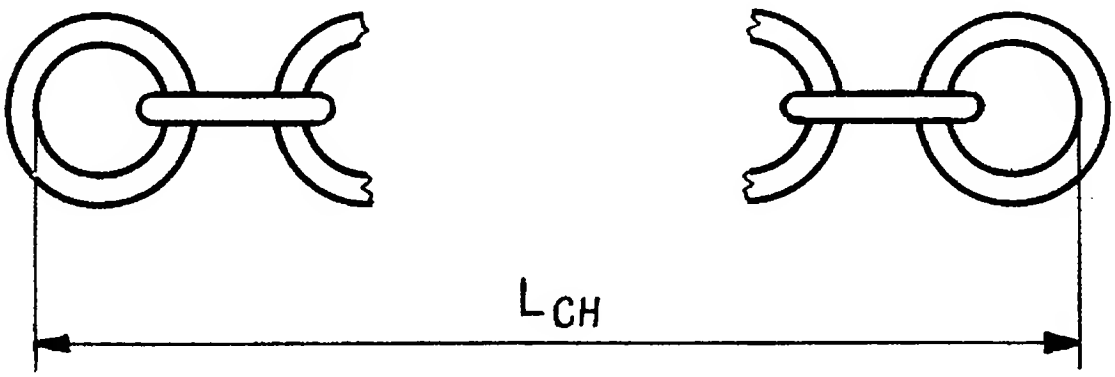
Some chain suppliers express the mentioned specific properties as chain surface area per 1 m of chain and chain weight per 1 m of chain. These values are formulated as follows:

$$O_{1m} = O_{1m\,link} \bullet \frac{1000mmm}{d(mm)}$$

$$G_{1m} = G_{1m\,link} \bullet \frac{1000mmm}{d(mm)}$$

Therefore, if the chain weight and surface area are to be calculated from the 1m specific data, the chain length L_{CH} should be measured as shown in the following sketch:

Figure



5.2 Long Links

The long links (Fig. 12b) cannot rotate like the round ones, their life time is shorter and their self-cleaning ability poorer.

For calculations of the chain surface area and chain weight, the following formulas can be applied:

- surface area of 1 link $O = \pi s\{2(l-d)+\pi(s+d)\}$
- volume of 1 link $V = 0 \bullet \frac{s}{4}$
- weigt of 1 link $G = V \bullet \gamma \; (\gamma \approx 7.8\, t/m^3)$

The above formulas are valid for long chain links with a round wire cross section.

5.3 Oval Links

The oval links (Fig. 12c) are similar to long links, they have similar properties and their surface area and weight can be calculated (with a negligible mistake) by means of the formulas mentioned in the part 5.2.

5.4 Other Types of Chain Links

Besides the links types mentioned above, the suppliers occasionally offer various other types of chain links such as asymmetric links, overlapping links (Fig. 12e), links with a non-round wire cross section (Fig. 12d) etc. These types are not to be recommended for cement kilns and are very rarely used, except the links with a non-round wire cross section.

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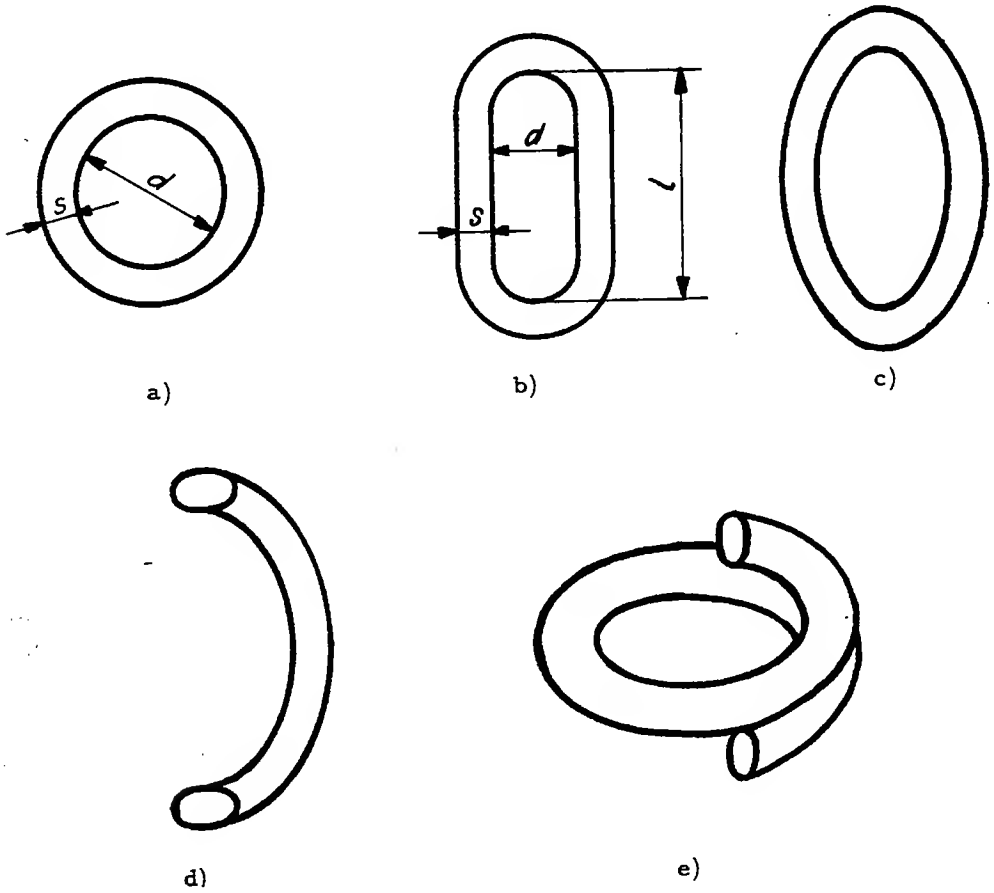
Figure 12: Type of Chain Links



6. CHAIN MATERIAL

6.1 Mild Steel Chains

A big majority of the kiln chains are made of mild steel. In order to withstand the friction between chains, between chain and raw material, between chain and hanger etc. they have to be made of a "through hardened" steel. This hardening (result of a thermal treatment) is one of the important chain properties and only experienced kiln chain suppliers are in possession of the necessary know-how for this procedure. For this reason, even the mild steel chains should be purchased from experienced suppliers.

6.2 Heat Resistant Alloy Chains

Due to the thermal load of the kiln and the chain temperature at the hot end of the system, the portion of the heat resistant chains should be some 15% of the total weight of chains. Mild steel chains should not be installed in that part of the system where a chain temperature of 450°C or more is to be expected.

The chain suppliers offer a lot of various heat resistant steel qualities. Besides the thermal treatment the chemical composition of the respective alloy is the most important criteria. The two main components are nickel and chromium.

Nickel increases the alloy resistivity against reducing kiln atmosphere, but a higher nickel content becomes dangerous if the raw material or kiln gases contain sulphur compounds which could react with it.

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Chromium increases the alloy resistivity against high temperatures, but a higher content of chromium makes the alloy sensible to sudden changes of temperature.

Above 20% Cr and at operating temperatures between 600 - 900°C an intermetallic compound can occur (-phase) which makes the alloy very brittle and causes destruction. Whether this phenomenon occurs or not depends also on the Ni-content and on other elements. Alloys which are sensitive to -phase formation must be used at working temperatures above 900°C.

Because of the properties mentioned above, it is necessary to find a compromise. Good results have been achieved with heat resistant alloys containing approx. 18 to 25% Cr and approx. 5 to 13% Ni.

7. CHAIN HANGERS

Chain hangers can be divided into 2 groups, namely single chain hangers and multiple chain hangers.

The single hangers have only one chain fastening point, the multiple hangers have several fastening points. The single chain hangers should be preferred in cases, where larger distances between the chain hanging points are desired - this solution enables to keep the weight of hangers as low as possible. (Under favorable circumstances, the weight of hangers should not exceed some 15% of the weight of the chains).

The chain can be fixed to the hanger by means of a shackle (Fig. 15). Specially designed hangers enable shackleless hanging of chains.

In Fig. 13 and 14 some examples of the chain hangers are presented, namely:

Fig. 13 a, b, c	single hangers with shackles
Fig. 13 d	single hanger, shackleless
Fig. 13 e, f	
Fig. 14 b, c	multiple hangers with shackles
Fig. 13 g	
Fig. 14 a	multiple hangers, shackleless

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Figure13:

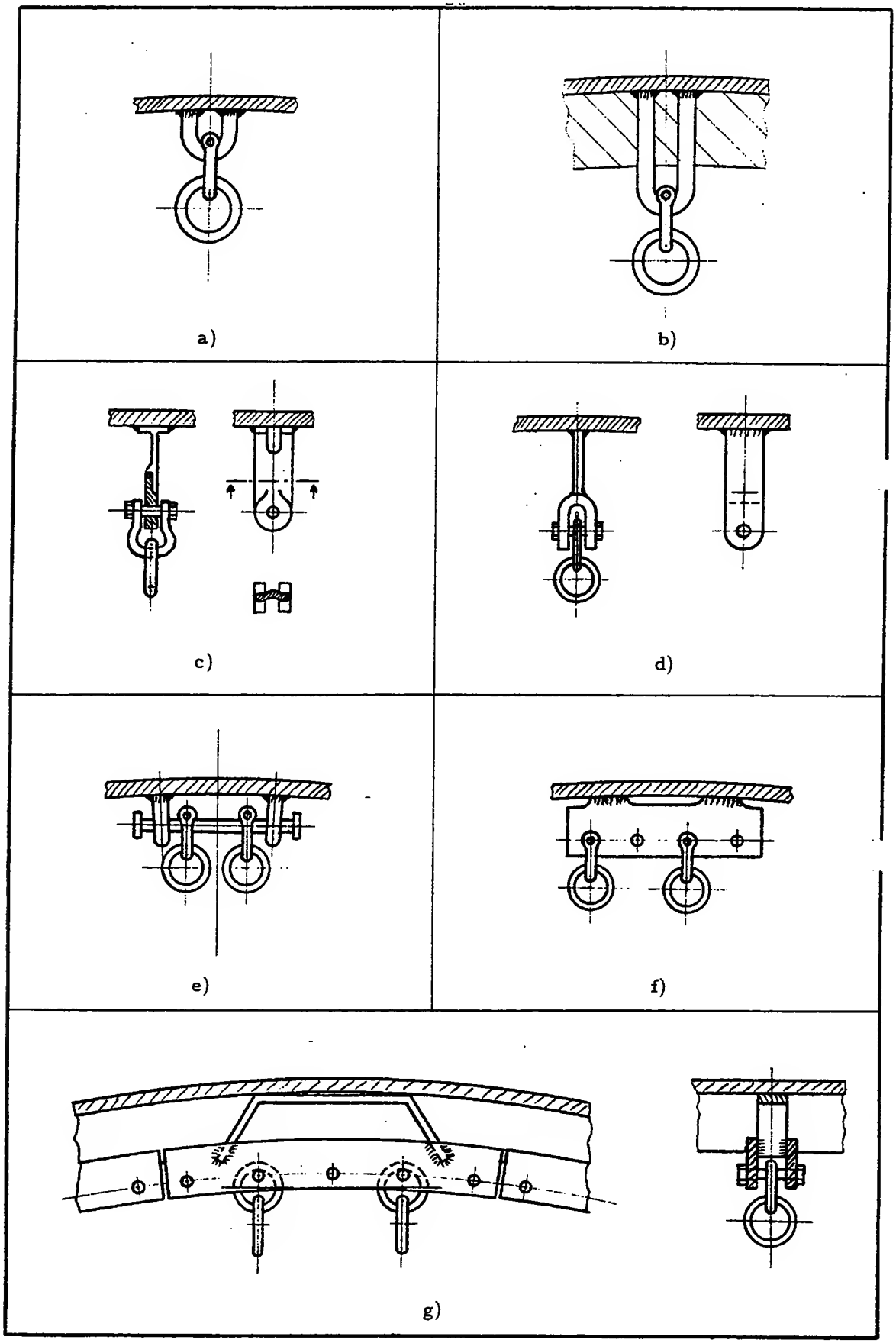

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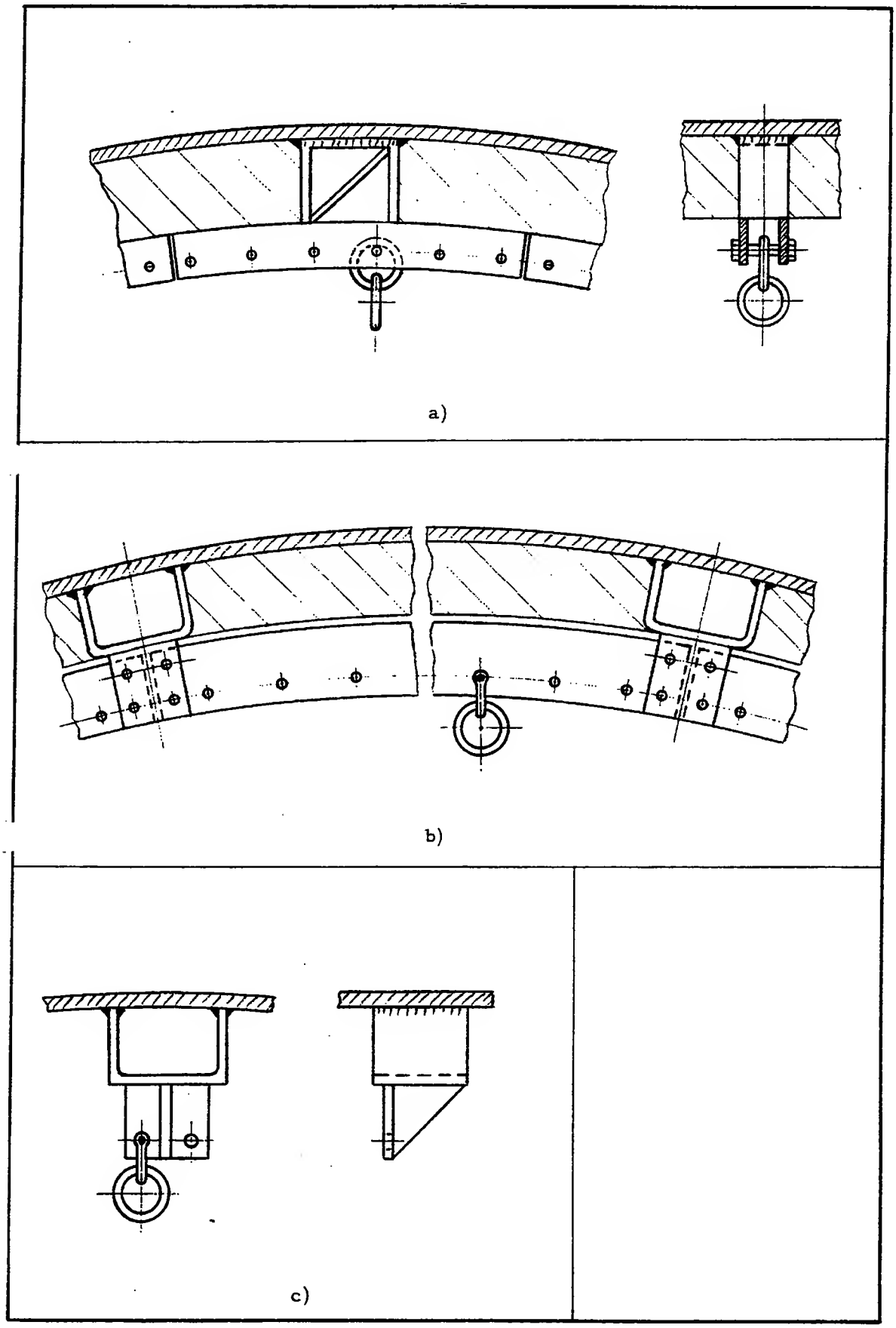
Figure 14:



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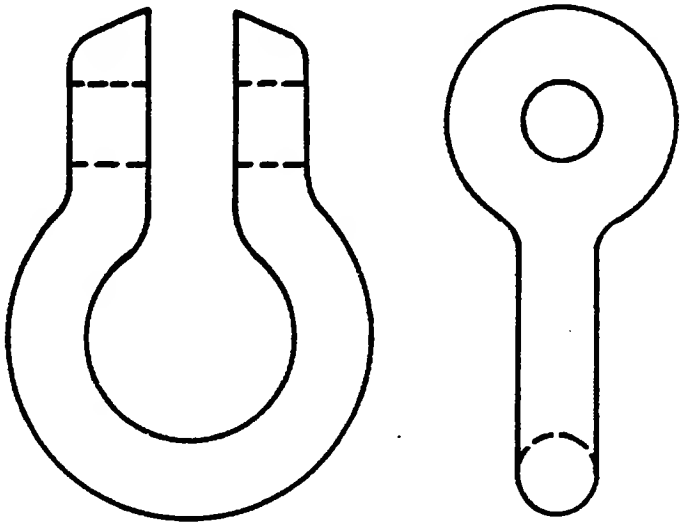
The gap between the plate of a multiple hanger and the steel kiln shell should not be too wide. Gaps exceeding some 20 mm enable, when the kiln lining is worn out, penetration of chains into the space between kiln shell and hanger. This leads to tangling of chains and finally to a destruction of chains and hangers.

Figure 15: Shackle



8. MAIN CHARACTERISTIC DATA OF CHAIN SYSTEMS

The following average data are to be understood only as a very rough guideline.

Length of the total chain system	18 to 35% of total kiln length (some 6 to 10 D)
Total weight of chains: - smaller kilns (< 1000 t/d) - bigger kilns	 9 to 12% of daily kiln output 11 to 20% of daily kiln output
Total surface area of chains: - smaller kilns - bigger kilns	 F_e/F_{ii}*) = 1.1 to 1.8 F_e/F_i = 1.5 to 2.6

* F_e = total surface area of chains
F_i = total surface are of kiln shell (inside lining)

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9. ANNEXES

Annex 1: Example of chain system record keeping (DA K5)

Annex 2: Example of material sampling port arrangement on wet kiln (BP K1)

Annex 3: Example material sampling and mass balance (BP K1)

Annex 4: Example of material sampling port on wet kiln (BS K1)

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Annex 1: Example of chain system record keeping (DA K5)

Darra Kiln #5 Chain System

				BZ	CZ	IZ			
GENERAL DATA	Production rate	750 tpd	Kiln diameter	3.9	3.4	3.9	m	Kiln length	141.7 m
	Heat consumption	6200 kJ/kg.cli	Mechanical load	78	106	78	t/m².d	Kiln surface (IL)	1460 m²
	Kiln feed moisture	36.6 %	Thermal load	5.6	7.6	5.6	MW/m²	Kiln volume (IL)	1204 m³
			Section length	32.9	62.6	46.2	m	Kiln load (IL)	0.62 t/m².d
									45 KW/m³

DESIGN VALUES BEFORE FEBRUARY 95 OUTAGE — Production rate 750 tpd

		Zone Start m	Zone End m	Zone Length m	Zone Dia IL m	Chain Mat'l	Chain Density m²/m²	Chain Density kg/m²	No of Links	Free Height %	Design Chain Inches	No. of Chain	Total Area m²	Total Weight Tons	Pattern
Free		141.7	135.9	5.78	3.90										
Dust	1A - 1C	135.9	133.7	2.23	3.80	MS	9.0	333	37	21%	0.75	342	227	8.4	9 start 30° spiral
Plastic	2 - 7	133.7	123.9	9.78	3.80	MS	5.0	247	37	21%	1	588	554	27.4	4 start 30° spiral
Preheat	8 - 10	123.9	119	4.89	3.80	MS	4.1	202	37	21%	1	240	226	11.2	1 start spiral
Hot	11 - 15	119	110.9	8.15	3.80	MS	5.7	283	37	21%	1	560	528	26.1	1 start spiral
Radiation	16 - 19	110.9	104.4	6.52	3.80	SS	4.1	202	37	21%	1	320	302	14.9	1 start spiral
				37.4								2,050	1,836	88.1	
	Percent of kiln length			26.4%											
	Length in D_{IL}			9.8								Fe/Fi =	1.26	11.8%	of 750 TPD Production

REMARKS — Average heat consumption in 1994: 6900 kJ/kg.cli; Average dust loss in 1994: 23 %

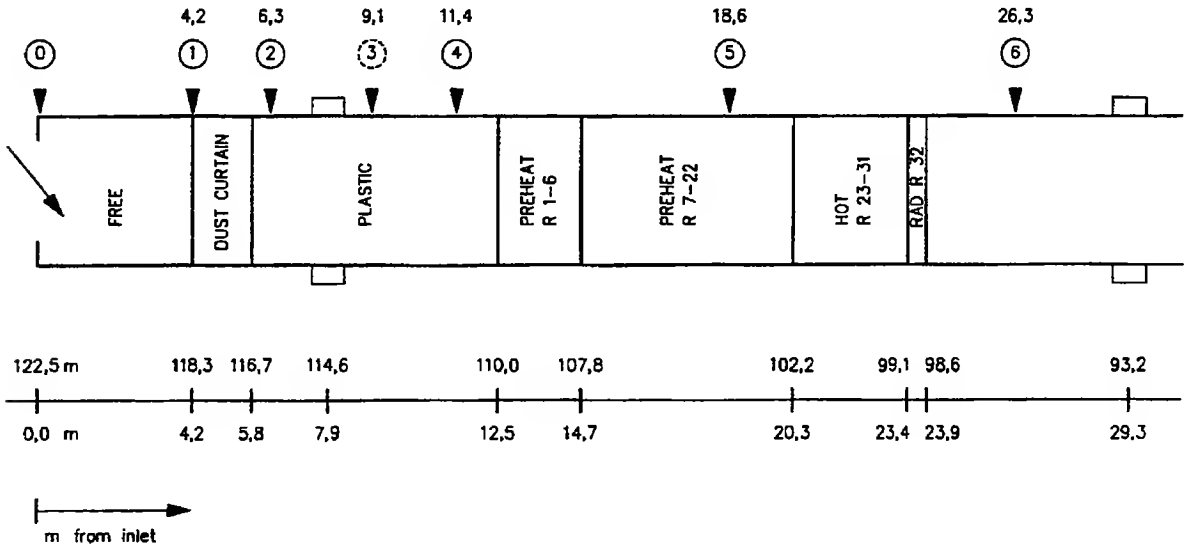
ACTUAL VALUES BEFORE FEBRUARY 95 OUTAGE — Production rate 750 tpd

		Zone Start m	Zone End m	Zone Length m	Zone Dia IL m	Chain Mat'l	Chain Density m²/m²	Chain Density kg/m²	No of Links	Free Height %	Actual Thick. %	Actual Chain Inches	Actual Length %	No. of Chain (Design)	Total Area m²	Total Weight Tons	Pattern
Free		141.7	135.9	5.78	3.90												
Dust	1A - 1C	135.9	133.7	2.23	3.80	MS	9.0	333	37	21%	100	0.75	100	342	227	8.4	9 start 30° spiral
Plastic	2 - 7	133.7	123.9	9.78	3.80	MS	4.5	210	37	21%	95	0.95	95	588	494	23.2	4 start 30° spiral
Preheat	8 - 10	123.9	119	4.89	3.80	MS	3.6	166	37	21%	93	0.93	97	240	200	9.2	1 start spiral
Hot	11 - 15	119	110.9	8.15	3.80	MS	3.3	145	37	21%	90	0.90	65	560	301	13.4	1 start spiral
Radiation	16 - 19	110.9	104.4	6.52	3.80	SS	3.7	173	37	21%	95	0.95	96	320	272	12.8	1 start spiral
				37.4										2,050	1,494	67.1	
	Percent of kiln length			26.4%													
	Length in D_{IL}			9.8										Fe/Fi =	1.02	8.9%	of 750 TPD Production

PROPOSED MODIFICATIONS FOR FEBRUARY 1995 OUTAGE — Production rate 750 tpd

		Zone Start	Zone End	Zone Length m	Zone Dia IL m	Chain Mat'l	Chain Density m²/m²	Chain Density kg/m²	No of Links	Free Height %	Actual Thick. %	Actual Chain Inches	Actual Length %	No. of Chain (Design)	Total Area m²	Total Weight Tons	Pattern
Free		141.7	136	5.78	3.90												
Dust	1A - 1C	136	134	2.23	3.80	MS	8.7	324	36	23%	100	0.75	100	342	220	8.2	9 start 30° spiral
Plastic	2 - 7	134	124	9.78	3.80	MS	5.5	261	35	25%	95	0.95	100	735	615	28.9	4 start 30° spiral
Preheat	8 - 10	124	119	4.89	3.80	MS	4.9	227	35	25%	93	0.93	100	336	274	12.8	1 start spiral
Hot	11 - 15	119	111	8.15	3.80	MS	3.8	141	35	25%	100	0.75	100	560	351	13.0	1 start spiral
Radiation	16	111	109	1.63	3.80	SS	3.9	191	35	25%	100	1	100	80	71	3.5	1 start spiral
Radiation	17	109	108	1.63	3.80	SS	3.4	169	31	33%	100	1	100	80	63	3.1	straight curtain
Radiation	18	108	106	1.63	3.75	SS	3.1	151	27	40%	100	1	100	80	55	2.7	straight curtain
Radiation	19	106	104	1.63	3.75	SS	2.6	129	23	48%	100	1	100	80	47	2.3	straight curtain
				37.4										2,293	1,896	74.5	
	Percent of kiln length			26.4%													
	Length in D_{IL}			9.8										Fe/Fi =	1.16	9.9%	of 750 TPD Production

Annex 2: Example of material sampling port arrangement on wet kiln (BP K1)



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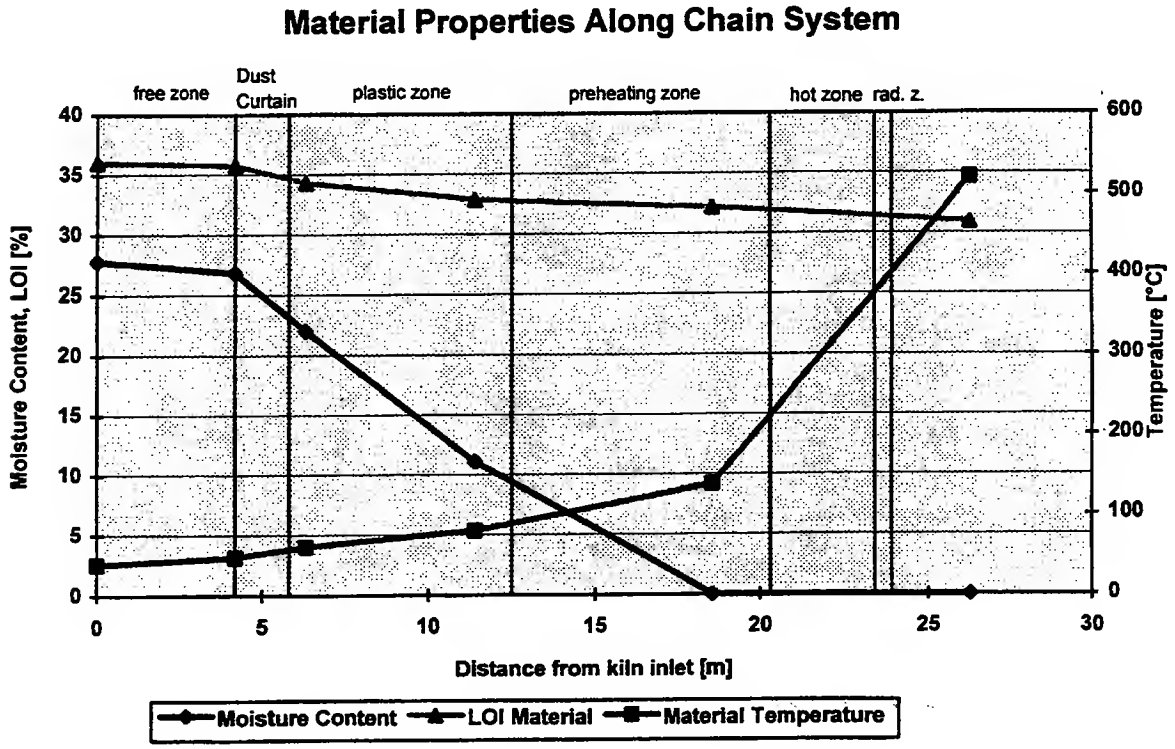
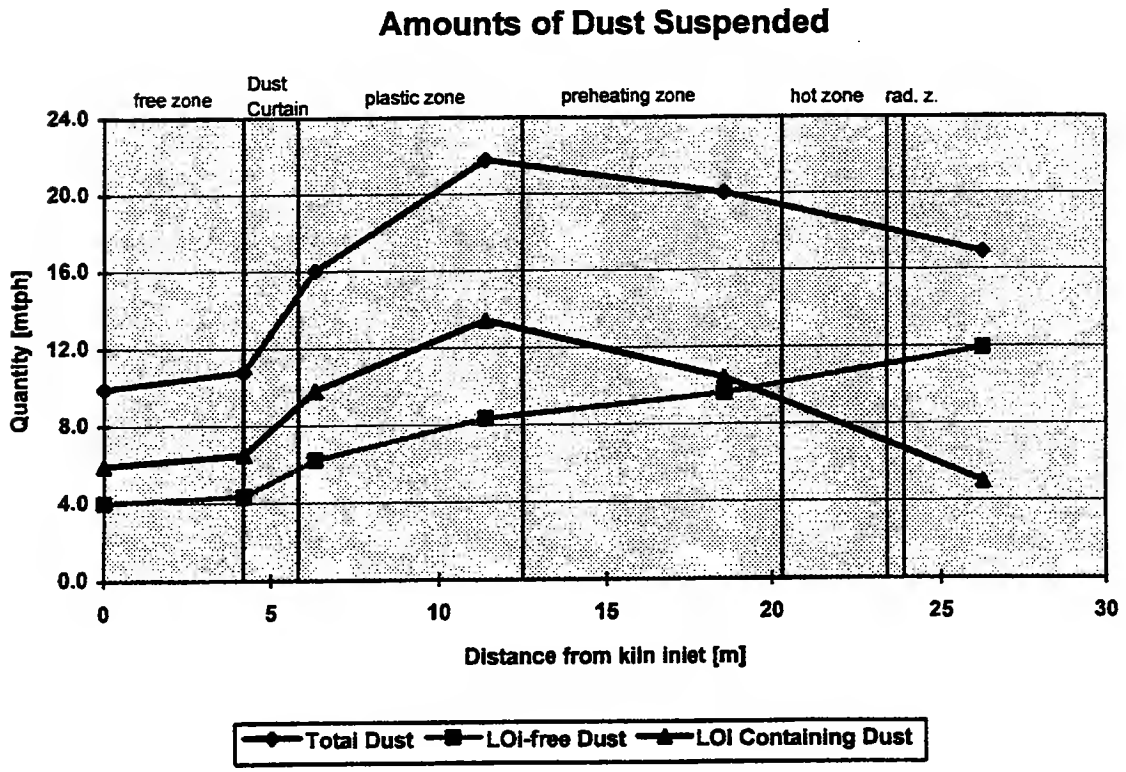
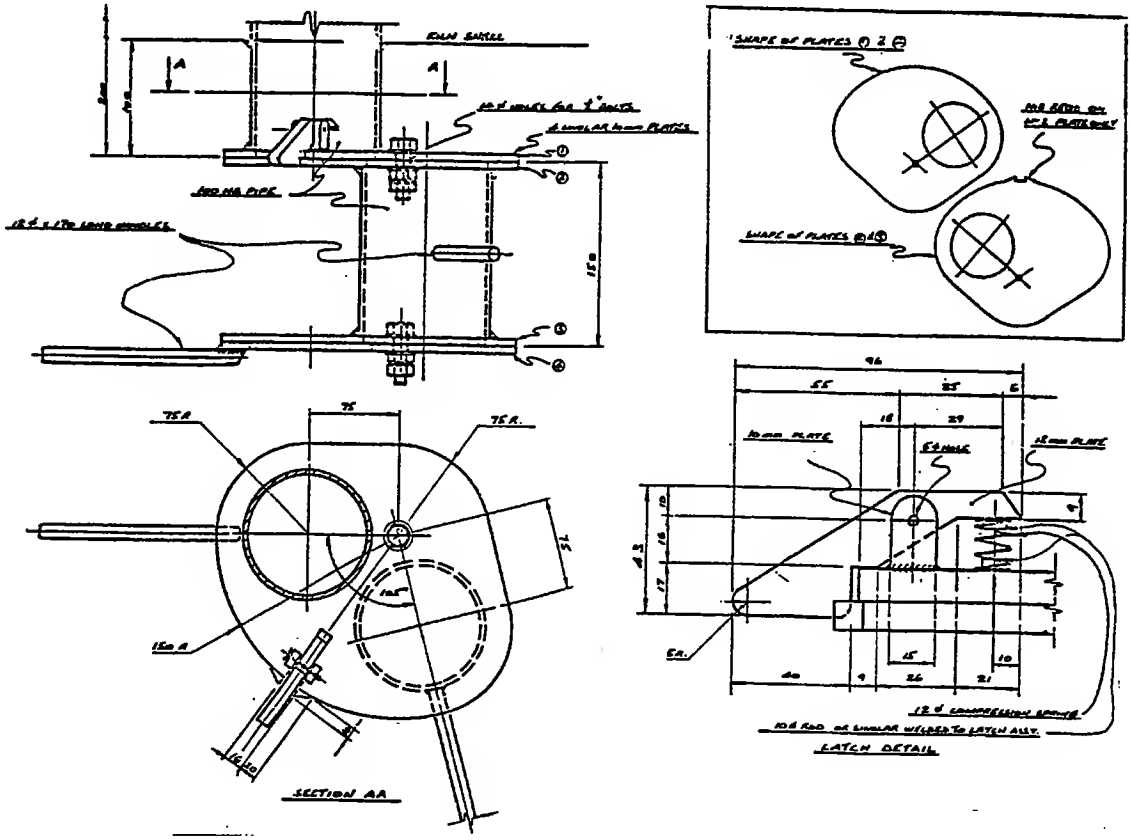
Annex 3: Example material sampling and mass balance (BP K1)





Annex 4: Example of material sampling port on wet kiln (BS K1)



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"Datenbank-Blatt" Collection of TC-VA

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11. TEST QUESTIONS

1) Name the main functions of a chain system and explain them!

2) According to the changing properties of material passing along the kiln tube, which individual zones do we distinguish inside of a chain system? Describe them, define the main requirements which the chains have to meet in each individual zone!

3) Name the different arrangements of chains and describe them!

4) Each arrangement has advantages as well as disadvantages, please list them!

5) Follow the stream of material in the kiln and define which chain arrangements can be used for the individual parts of the system! Explain why!

6) Explain the reverse sense of hanging garlands! Compare it to the sense of hanging thermochains

7) Would you recommend garlands for the downstream (hot) end of the system, thermochains for the upstream (cold) end? Please explain why!

8) What are the specific advantages of round link chains?

9) Describe the material and gas temperature profile along the system!

10) What portion of heat resistant steel chains would you recommend for a system? Define the main criteria for estimating this amount!

11) Which basic types of hangers do you know? In which case would you prefer single chain hangers?

12) Name the main characteristic data of a chain system: Length, total weight and total surface area of chains?

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Chapter 3

Refractory Linings

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Refractory Lining of Cement Kiln Systems

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SUMMARY

Since the sole costs of refractory materials per ton of clinker produced play only a minor role in the overall manufacturing costs, the prior task of refractory lining optimization is to prolong the useful life of the installed materials which fact in turn increases the kiln availability for production.

Selecting of the appropriate refractory qualities for the different kiln zones is of particular importance in various problem areas of the kiln system. As an example, alkali corrosion in calcining zones may be reduced by selecting acidic types of bricks, whereas eutectic reactions in the safety zone generally can only be countered by installing basic types of refractories.

Apart from selecting the correct refractory materials, the work quality of lining installation is a key element in obtaining a long refractory life. Tight lining and adequate joint dimensions are important features.

The operational influences on refractories materials, the work quality of lining installation is a key element in obtaining a long refractory life. Tight lining and adequate joint dimensions are important features.

The operational influences on refractories performance are of thermal, chemical and mechanical nature. By considering an adequate heating-up procedure, thermal and mechanical damage can be avoided during start up. Measures to provide an optimum raw material composition on one hand and to avoid kiln feed fluctuations on the other hand may be necessary to decrease wear of chemical nature and to improve coating stability. In the area of kiln tyres, mechanical destruction of bricks can be caused by excessive kiln shell deformation. Continuous control of the mechanical condition of the kiln shell in the tyre areas allows to detect possible deterioration and to take appropriate measures in an early stage in order to prevent damage to the lining from this side.

Refractory lifetime is generally not a matter of one single criterion. The influences described in this paper are nearly always jointly responsible for the results achieved and should be considered as an entirety.

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1. IMPORTANCE OF REFRACTORIES FOR CEMENT PRODUCTION

Cement plans represent quite big amounts of invested capital which fact demands for a production as continuous as possible in order to guarantee a corresponding return.

The sole cost of the refractory lining of a cement kiln, including the outlay for removal of the old and installation of the new materials amounts to barley 2 to 4% of the cement production costs. The losses caused by interruption of production, on the other hand, are already higher than the annual lining costs when the kiln has been idle for 10 to 15 days. Improvement or optimisation of the useful life of refractories with the object of impairing the availability of the kiln as little as possible by shut-down for relining, is therefore one of the major objectives for cement producers.

The durability of refractory linings is mainly influenced by three factors:

- The choice of the quality of materials employed in the various zones.
- The installation of the lining with due consideration of the methods of placing, size of joints and jointing materials.
- Due attention to operational criteria which affect the durability of the lining, namely the correct procedure for heating-up and for cooling down of a kiln system on the one hand and the minimisation of process fluctuations in order to maintain the continuous operation of the kiln on the other hand.

This paper contains accordingly remarks and information on how to consider the above criteria in order to improve refractory performance and kiln availability.

For types and classification of refractory materials and other aspects related to materials technology, reference is made to the MA Cement Course Documents Vol. 2.

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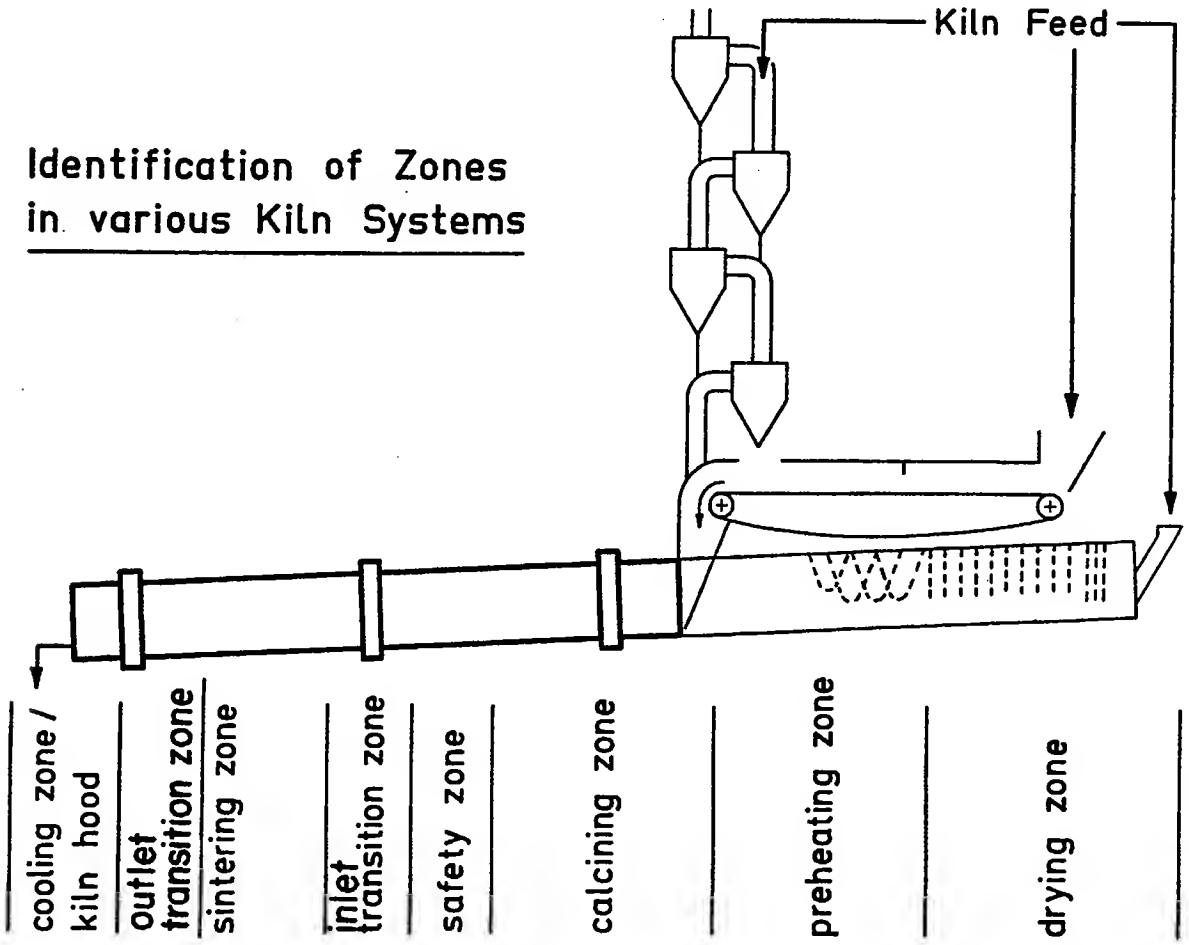
2. SUBDIVISION OF THE BURNING PROCESS AND SELECTION OF REFRACTORY QUALITIES

With regard to the requirements on refractory materials, the kiln system can be subdivided into various zones according to specific operating conditions (Fig. 1). The designation of the various zones refers to the respective reactions in the burning process which, however, are anything but clearly defined as they overlap in both directions. The point, at which the change from one refractory quality to another is made can usually be determined only by observing the behaviour of the lining in operation.

Fig. 1



The main points of the following explanations are also summarised in Table 1.

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Table 1

TABLE 1: SUBDIVISION OF BURNING PROCESS AND SELECTION OF BRICK QUALITIES

ZONE	APPROX. MATERIAL TEMPERATURE		ZONE LENGTH TO DIAMETER RATIO					BRICK QUALITY
	Beginning (°C)	End (°C)	suspension preheater kiln	grate pre-heater kiln	precalciner kiln	long wet kiln	long dry kiln	
Drying zone	20	100	-	preheater	-	chain system	-	Fireclay
Chain zone	20	200-400	-	-	-	7 - 9	3 - 5	Fireclay
Preheating zone	20 (100)	700	preheater	preheater	preheater	4 - 8 (excl. chain-system)	4 - 8 (ex. (chain-s.)	lightweight fi-rebricks, fire-clay
Calcining zone	700	900	4 - 6	4 - 6	1 - 3	6 - 9	6 - 9	lightweigth fi-rebricks, fire-clay
Safety zone	900		2	2	2	2	2	high alumina 50 - 60 (-80) % Al_2O_3
Transition zone		1050 - 1150	2 - 4	2 - 4	2 - 4	2 - 4	2 - 4	Magnesia-Spinell (Magnesia-chrome 60 - 70% MgO)
Sintering zone	1050 - 1150	1400 - 1500	3 - 5	3 - 5	5 - 8	3 - 5	3 - 5	Dolomite, Magnesia-Spinell (Magnesia-chrome 70 - 90% MgO)
Outlet transition zone	1400 - 1500	1350 - 1400	1 - 2	1 - 2	1 - 2	1 - 2	1 - 2	Magnesia Spinell (Magnesia-chrome 60 - 70% MgO)
Cooling zone (discharge zone)	1350 - 1400	1250 - 1350	0,5 - 2	0,5 - 2	0,5 - 2	0,5 - 2	0,5 - 2	high alumina. 80 - 90% Al_2O_3

2.1 Drying Zone (applies only for wet and semi-wet process kilns)

In the drying zone, the water content of slurry or nodules is evaporated. This reaction is almost terminated at material temperatures of 100°C.

In wet process kiln, the drying zone is generally a part of the chain zone. In grate pre-heater kilns, drying takes place in the first pre-heater chamber.

2.2 Chain Zone

In wet process kilns, the chain zone can be subdivided into a drying and a preheating zone. The material temperature at the outlet of the chain zone depends on arrangement and qualities of chains as well as type of system, i.e. nodule or dust kiln, and is in the order of magnitude of 200 to 400°C.

The length of the chain zone ranges from 7 to 9 kiln diameters in wet kilns and from 3 to 5 kiln diameters in long dry kilns.

The lining of the chain zone should be of a dense, low alumina firebrick with high abrasion resistance to withstand the abrasive action of the chains.

An air setting mortar should be used as there is insufficient temperature to vitrify a heat setting mortar. In wet kilns, furthermore, the mortar must be water resistant.

Since the arrangement of the chain hangers requires extensive modification work, sometimes dense fireclay castables are installed, which, however, should be carefully compacted in order to obtain high abrasion resistance.

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2.3 Preheating Zone

In the preheating zone, hydrate-water is driven off and the raw material is heated up to approx. 700°C. The length of the preheating zone in long kilns is approx. 4 to 8 kiln diameters (excluding chain zone). In short pre-heater kilns the preheating zone is a part of the pre-heater.

Regarding the lining of preheaters, reference is made to paragraph 4.1: Lining of Suspension and Grate Preheaters.

The preheating zone of long kilns is usually lined with low alumina firebricks or, for better thermal insulation, with lightweight firebricks. With respect of lining stability, two layer lining is generally not recommended in rotary kilns of more than 3.5 m diameter.

2.4 Calcining Zone

The calcining reaction starts already at material temperatures below 600°C and is completed at approx. 1200°C. Since, however, the most part of calcination takes place between raw material temperatures of 700 to 900°C, usually this range is designated as calcining zone.

The length of the calcining zone in long kilns is in the order of magnitude of 6 to 9 kiln diameters. In short pre-heater kilns calcining takes partially place in the pre-heater. There, the length of the calcining zone in the kiln amounts to approx. 4 to 6, in kilns with precalciners 1 to 3 kiln diameters.

From the refractory point of view, the calcining zone can still be lined with normal fireclay bricks or, for better insulation, with fireclay lightweight bricks. In case of alkali attack good operational results have been achieved with acidic light weight bricks with an SiO_2 content above 65%. These bricks form with alkalis a vitreous layer of 2 to 3 mm thickness on the surface, which prevents the further alkali infiltration.

Two layer linings are generally not recommended. There are, however, two layer bricks on the market, consisting of a dense working part and a porous insulating part. Such bricks are generally installed for heat saving purpose, if lightweight bricks show unsatisfactory operating results.

2.5 Transition Zone

The transition zones are located on both sides of the sintering zone. Since the length of the sintering zone varies with process fluctuations, the transition zones are characterised in particular by unstable coating formation.

Usually, the inlet-side transition zone is further subdivided into a colder section, so-called safety zone, and into a hot section, the proper transition zone.

The safety zone has a length of approx. 2 kiln diameters and is usually lined with alumina rich bricks with Al_2O_3 content of 50 to 60%. The bricks must have good thermal shock resistance and low porosity in order to have good resistivity against chemical attack. The application limit of alumina rich or high alumina bricks in the transition zone is generally determined by eutectic reactions in the system Al_2O_3 - CaO - SiO_2 or by alkali-spalling. In such cases, lining lifetime can be improved only by installing basic bricks.

The transition zones are often exposed to considerably higher loads than the sintering zone itself. Quite often no or only unstable coating is formed. Thus, the bricks are exposed directly to the flame radiation and considerable temperature variations. The lengths of the transition zones vary from 2 to 4 kiln diameters.They are usually lined with chrome free magnesia-spinell bricks produced of very pure, synthetic materials or with magnesia-chrome bricks containing approx. 60 to 70% MgO.

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2.6 Sintering Zone

Although this zone is often referred to as burning zone preference is given to the expression sintering zone on account that this better describes the mechanism of reactions taking place.

The sintering zone is generally covered with a more or less stable coating, formed with clinker and liquid phase. Liquid phase starts to form at raw material temperatures above approx. 1250°C. However, as the lining surface temperature is higher than the one of the raw material, coating formation starts already at raw material temperature above 1050 to 1150°C.

The term 'sintering zone' can also be explained as zone of increased material diffusion with formation of C_3S modifications, the latter starting at temperatures above approx. 1100°C.

The maximum material temperature in the sintering zone is in the order of magnitude of 1400 to 1500°C at the beginning of the cooling zone.

The length of the sintering zone is generally between 3 to 5 kiln diameters and depends largely on the shape of the flame and type of fuel. Coal flames give generally short, oil flames medium and gas flames longer sintering zones. In kilns with precalciner, the sintering zone length amounts to 5 to 8 kiln diameters due to the higher specific material throughout.

The bricks in the sintering zone are exposed to chemical attack by the liquid phase of the clinker and by alkali sulphates, high temperatures and, depending on coating stability, high thermal shocks. These conditions are best met by basic bricks due to their high refractoriness and good resistance against chemical attack.

Thus, generally, chrome free magnesia-spinell bricks, magnesia-chrome or dolomite bricks are installed. When using chrome-free magnesia-spinell bricks, qualities particularly developed to improve coating adhesion should be chosen (qualities based on natural sinter).

Dolomite bricks have generally good operating performance in zones with coating formation. The price for dolomite bricks is only approx. 60% of that of magnesite. A disadvantage of dolomite bricks is, however, its sensitivity to moisture. Thus, for longer storing times, these bricks are to be stored air-tight. During longer kiln stops the dolomite lining is to be protected against moisture by tightening the kiln tube and putting a hygroscopic agent (e.g. burned lime) in the kiln.

Raw material analysis and tests can give some indications on selection of brick quality to be installed. The final decision, however, is often to be based on operating experience, i.e. by trial and error.

Insulation of the burning zone with insulating back-lining is generally not recommended. By insulating, basically the hot face temperature of the bricks will be increased, resulting in reduced coating thickness and higher chemical and thermal load of the bricks. Furthermore, two layer lining is less stable and leads, particularly in big kilns, often to early failure due to relative movement and loosening of the lining.

In cases, where a coating does not form, insulation may be helpful in reducing heat losses and protecting the kiln shell, particularly in the tyre area. In such specific cases basic bricks with back-linings of 40 mm hard fireclay slabs are sometimes installed. However, generally installation of two layer linings is not recommended due to reduced lining stability.

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2.7 Cooling Zone

The cooling zone in the rotary kiln reaches approx. from the burner nozzle to the kiln outlet. In this zone, the clinker is cooled down from its maximum burning temperature of 1400 to 1500°C to approx. 1350°C in kilns with grate, rotary or shaft coolers and to approx. 1250°C in kilns with planetary coolers.

The cooling zone in kilns with planetary coolers has a length of approx. 1.5 to 2 kiln diameters. It consists generally of a cam lining for better cooling efficiency and a dam ring for equalising of clinker distribution to the individual cooler inlet openings. For camlining and damring, dense high alumina bricks with approx. 80% Al_2O_3 and considerable thermal shock resistance are generally used. The backing of the damring can be made of hard fireclay bricks with adequate mechanical strength.

The length of the cooling zone of kilns with grate, rotary or shaft coolers is generally 0.5 to 1 kiln diameter. It is usually lined with dense, abrasion resistant high alumina bricks containing 80% Al_2O_3. In case of heavy chemical attack it can be necessary to line this zone with basic bricks, which however, should have high thermal shock resistance (magnesia-spinell bricks). Since the outlet zone is often free of coating, kiln shell temperature will increase due to the higher thermal conductivity of basic bricks. Due to the high thermal and mechanical load of the bricks in this zone, insulating back-lining is not recommended.

The end of the cooling zone, the nose ring, is one of the most critical points of cement kiln linings. Its lifetime is often lower than the one of high loaded sintering zones. Shape and quality of nose ring bricks requires therefore careful consideration.

Basically, complicated special shapes should be avoided since special shaped bricks are often 'hand-made' and can have a much poorer quality than machine made bricks of the same composition. Nose ring design according to Fig. 2a and 2b would allow to use standard brick sizes with minor modifications and is to be preferred against the design according to Fig. 2c.

Fig. 2a Nose Ring Design Using Bricks

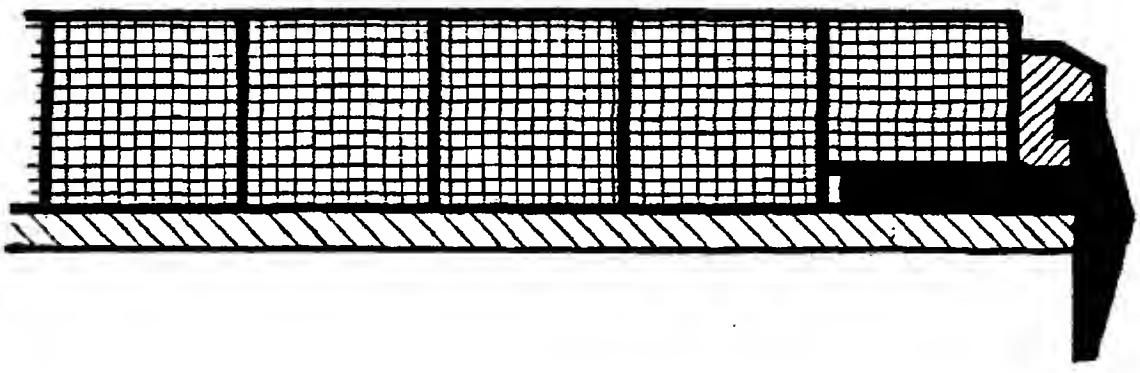

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Fig. 2b Nose Ring Design Using Bricks

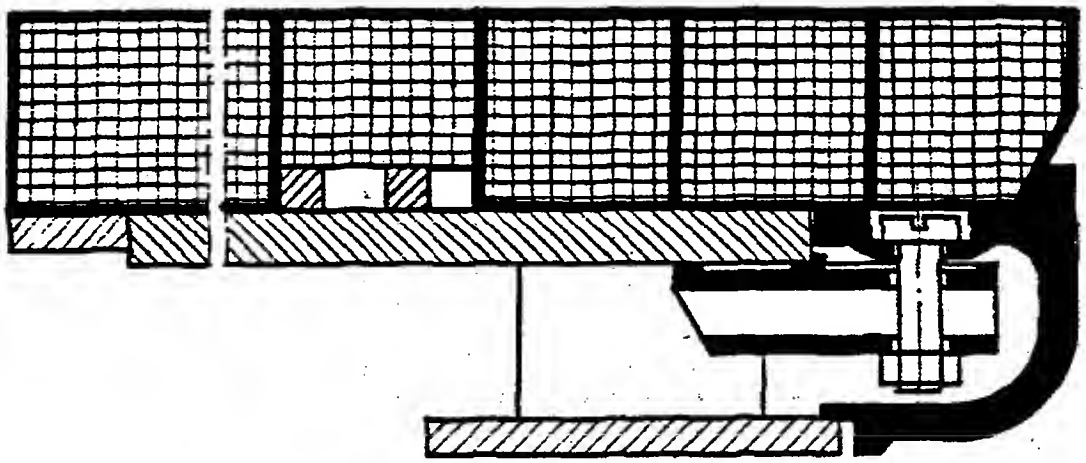
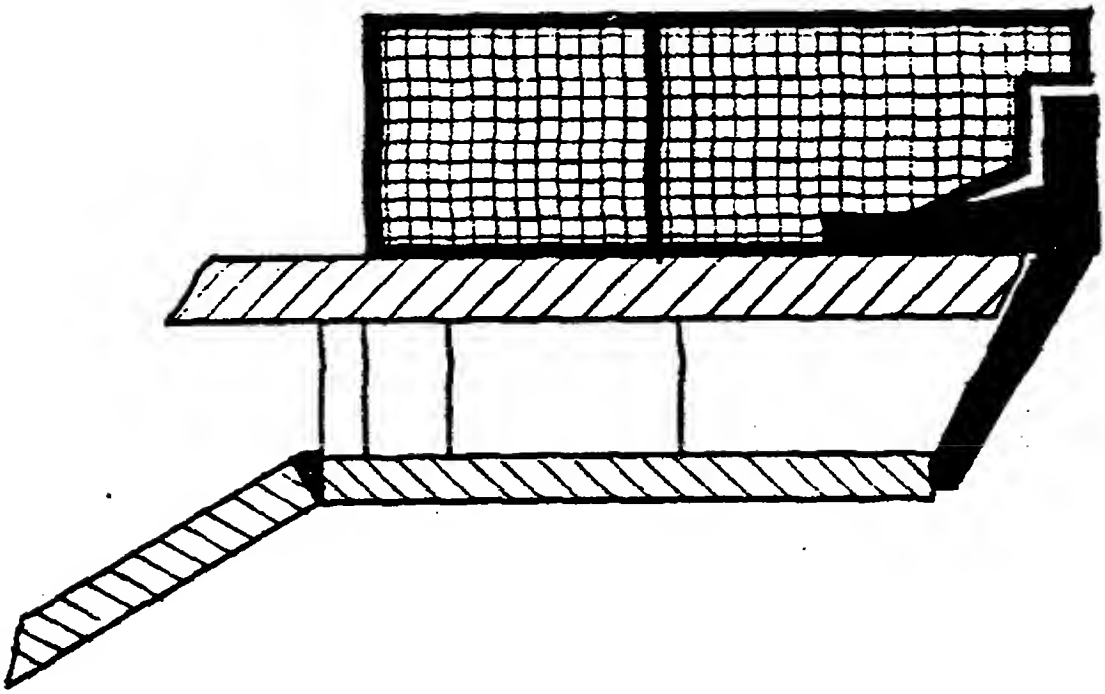


Fig. 2c Nose Ring Design Using Bricks



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3. IMPORTANT FEATURES OF REFRACTORIES INSTALLATION

3.1 General

Depending on specific requirements and local conditions, refractory brick linings in rotary kilns are installed dry or with mortar. For specific applications, unshaped materials are used, which are installed by casting, gunning or ramming.

Apart from selecting the correct refractory materials, the quality of the installation procedure is a key element in obtaining a durable refractory lining and long service life.

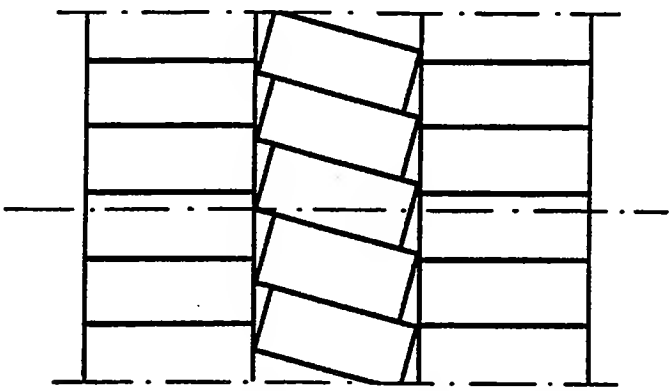
High refractory lining quality requires:

- Tight installation of brick
- Correct compensation for thermal expansion with adequate joints
- Selection of suitable mortar for mortar joints
- Selection of the optimum lining method, so that kiln rotation in the cold state can be minimised

Loose linings may result in spiral displacement of entire lining sections (see Fig. 3). The bricks jam in a twisted position and can no longer expand freely when hot. Result: spalling due to excessive mechanical stresses.

Fig. 3

Displacement of Lining



The lining ring must run true to the vertical kiln axis. Using a welding seam as reference, alignment lines can be drawn on the shell at regular intervals.

Installation of closure bricks requires special care.

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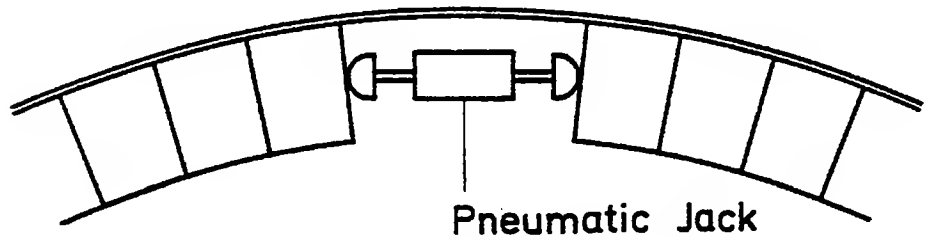
Procedure (Fig. 4)

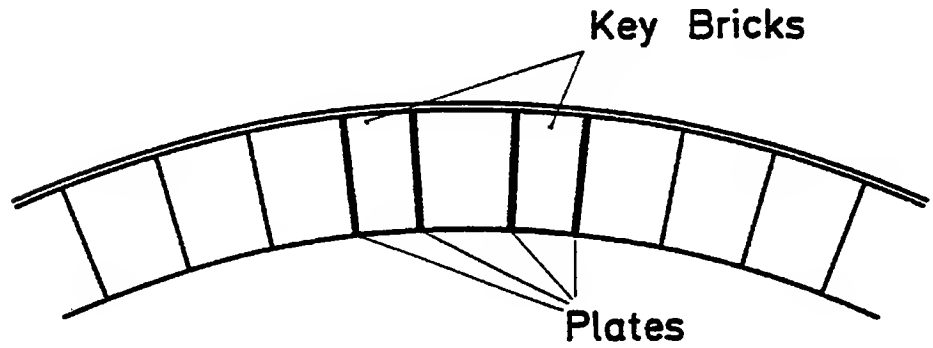
1) Tension the ring with a hydraulic spreader jack.
2) Insert the most tightly fitting combination of key bricks
3) With basic lining: drive in key plates between axial joints of the ring until ring becomes completely stable. Use only key plate per joint.
   With linings containing Al_2O_3: closing with key plates should be avoided because iron reacts with Al_2O_3 at high temperature to form a compound with a low melting point. For this lining quality, the combination of key bricks must fit particularly well.

Fig. 4

Closing of Lining Rings



Tightening of the Ring



Key Bricks installed

For closing lining rings, key bricks of various dimensions are available which can be combined to obtain optimum closures. If necessary, standard formats can be cut to the desired widths. The brick lengths of the last lining ring generally will have to be cut as well.

Rotary kiln bricks may only be cut to 2/3 of their original length or width; if necessary two rows must be cut.

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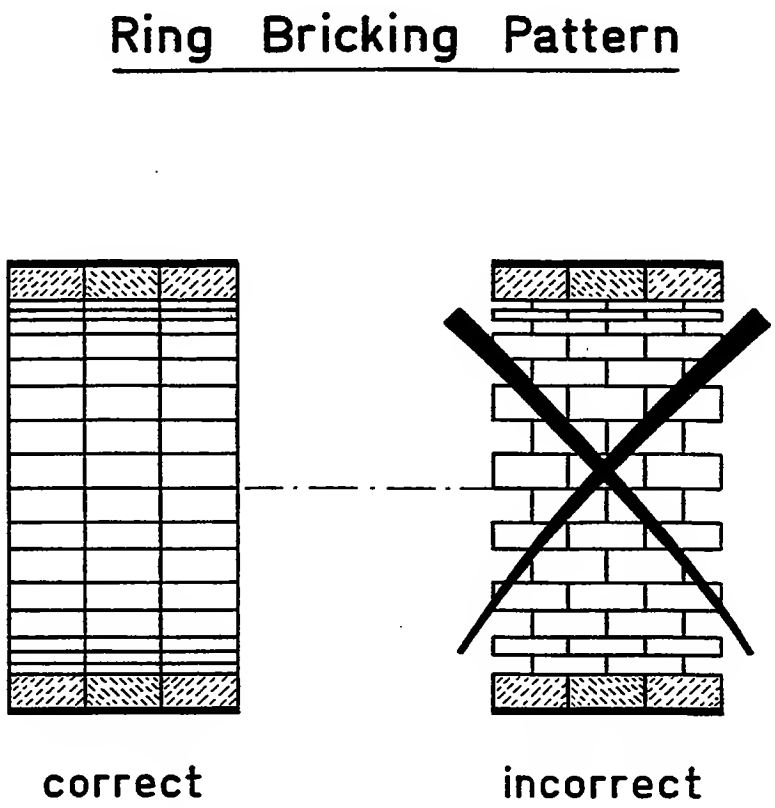
Offset brick linings as shown in Fig. 5 may not be installed in rotary kilns. Each ring must be self-supporting in order to avoid shearing stress occurring as a result of relative lining motion.

Fig. 5



3.2 Brick Joints and Jointing Materials

Bricks laid in mortar generally give better performance than dry placed bricks provided that the mortar used is of the correct quality. Mortar joints must be even and not thicker than 1.5 to 3 mm, preferably 2 mm.

To achieve the desired lining quality with mortar, qualified and experienced bricklayers are required, who regrettably are becoming a rarity in many countries.

In contrast, dry bricking can be executed in less time by less specialised personnel.

Dry lining is therefore the standard procedure for basic bricks. They are currently almost exclusively laid dry with or without steel plates in every radial joint. The steel plates react at high temperatures with the magnesia brick to form a highly refractory combination of magnesiaferrite which is claimed to give increased stability to the ring. Clench lining (dry lining without steel plates) is becoming more and more popular. Its main advantages are:

- tighter lining
- no handling of steel plates
- no separation of steel plates required when recycling old bricks

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Expansion allowances in axial and circumferential joints of the brick linings are sometimes necessary in order to compensate for the thermal expansion of the brick. With insufficient compensation of thermal expansion, the bricks can be mechanically overloaded at operating temperature. Excess compensation conversely leads to loose lining or even lining displacement and collapse of brick rings. Therefore the correct dimensioning of thermal expansion allowance is a most important feature of ensuring a stable lining.

3.3 Thermal Expansion Compensation

The thermal expansion of a refractory lining (re: Fig. 6 and 7) is partially compensated by the following factors:

- thermal expansion of kiln shell
- burning out of glue if sheet metal shims are glued to the brick surface
- lining 'inaccuracy'
- compressibility and pyroplastic behaviour of the bricks

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Fig. 6

Thermal Expansion of High Alumina Lining (typical)

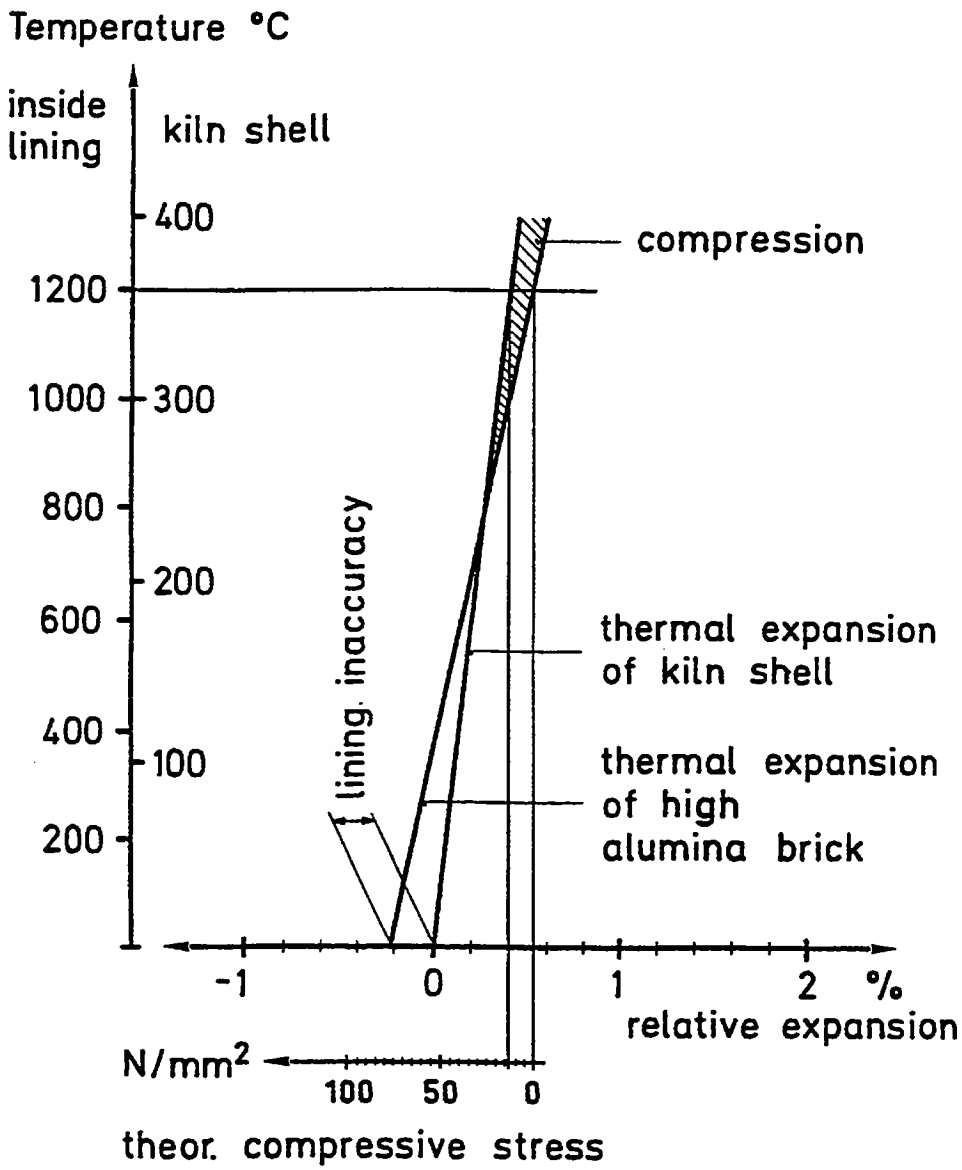

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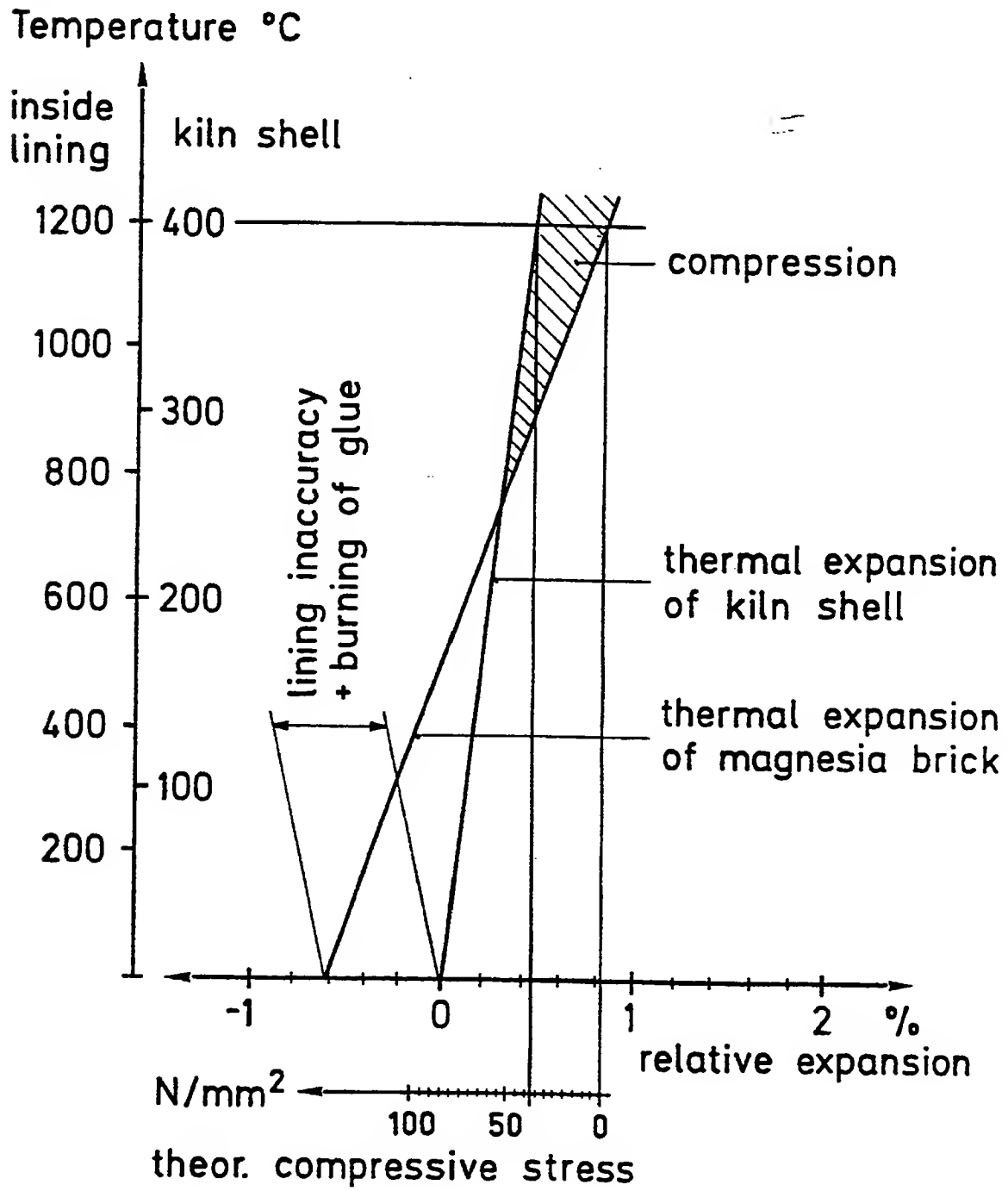
Fig. 7

Thermal Expansion of Magnesia Lining (typical)



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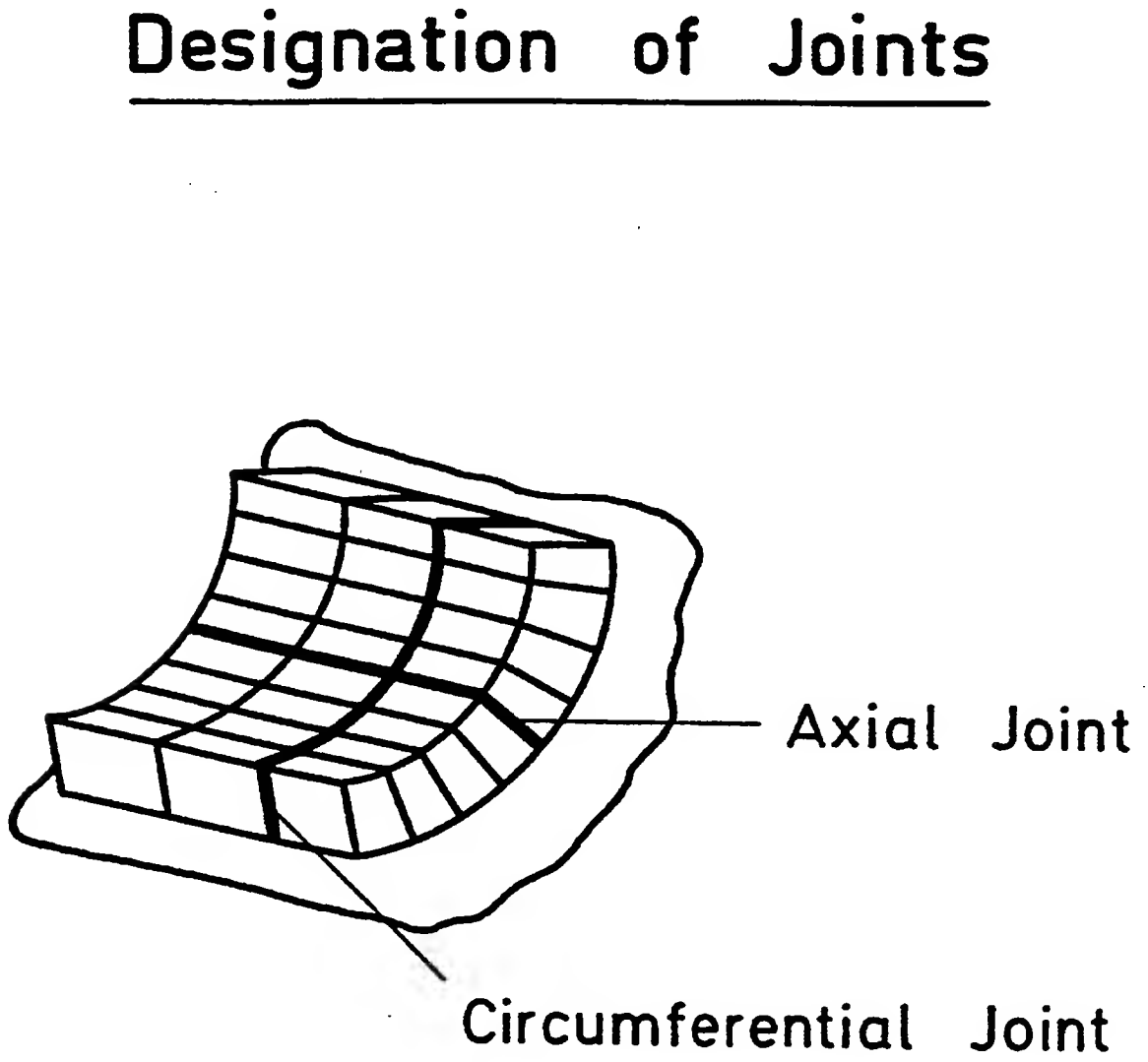
The thermal expansion which cannot be absorbed by the above factors has to be compensated by:

- the elasticity and pyroplasticity of mortars
- the elasticity and softening of steel plates
- combustible materials (cardboard shims)

The thermal expansion of refractory linings is strongly dependent on materials and operating conditions. For the correct dimensioning of the expansion allowance, the recommendations of the manufacturers of each brick type should be strictly followed.

For dry lining consisting of basic bricks, most manufacturers recommend a longitudinal expansion allowance of 1%, i.e. installation of a 2 mm cardboard in each circumferential joint for 200 mm brick length (for designation of joints see Fig. 8). Most qualities of basic bricks are normally laid without cardboards in the axial joints in order to achieve the tightest possible lining at working temperature.

Fig. 8



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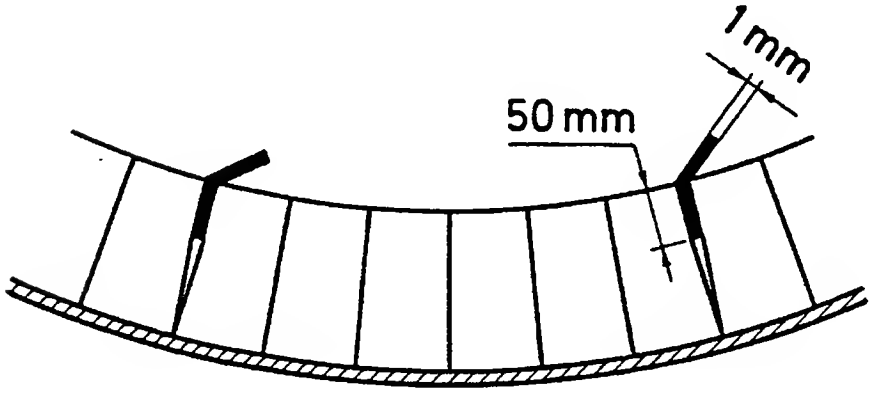
For some particularly dense, high fired qualities additional expansion allowances within the rings are recommended. In addition to the normal steel plate, a 1 mm thick, 50 mm wide cardboard is to be installed near the hot face of the axial joints after every 4th to 8th brick. To prevent the cardboards from slipping-in, they are bent at right angles to the brick face (see Fig. 9).

Fig. 9

Compensation of Thermal Expansion within Brick Rings for Direct Bonded Bricks



If basic bricks are laid in mortar, additional expansion allowance in the axial joints is not necessary. In the circumferential joints an allowance of about 0.2% is made, i.e. a 2 mm cardboard is inserted between each fifth ring without mortar.

Alumina and fireclay bricks as a rule are installed without expansion allowance in the interest of a stable lining. This is possible due to the lower thermal expansion of these bricks compared to the basic materials and due to good pyroplastic properties which permit accommodation of the stresses at high temperatures.

Steel plates are not used with Alumina bricks, since steel and aluminium silicates form low melting compound at high temperature, which can lead to lining damage.

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3.4 Lining Methods

The following factors have to be considered for kiln lining methods:

- safety
- tight installation of the bricks
- time required for lining

The methods applied can be generally classified as:

- rotating methods, i.e. the kiln has to be turned during lining
- formwork methods: the bricks are installed on a curved formwork without turning of the kiln

3.4.1 Rotating Methods

With the rotating methods, kiln zones of 5 to 20 m length are lined at one time. Work is carried out only in the bottom of the kiln, which is rotated intermittently as the work of lining proceeds. The bricks must be fastened to the shell, e.g. by means of screw jacks or glues, as the ring of bricks is rotated into the overhead position before being closed.

3.4.1.1 Screw-Jack Method (Fig. 10)

The lining is fastened by means of screw-jacks, usually of steel. The bricks are clamped in position by means of wooden wedges. The screw-jacks should be furnished with pressure control facilities as e.g. discsprings or load cells. The screw-jack method should not be employed for kilns more than 4 m in diameter.

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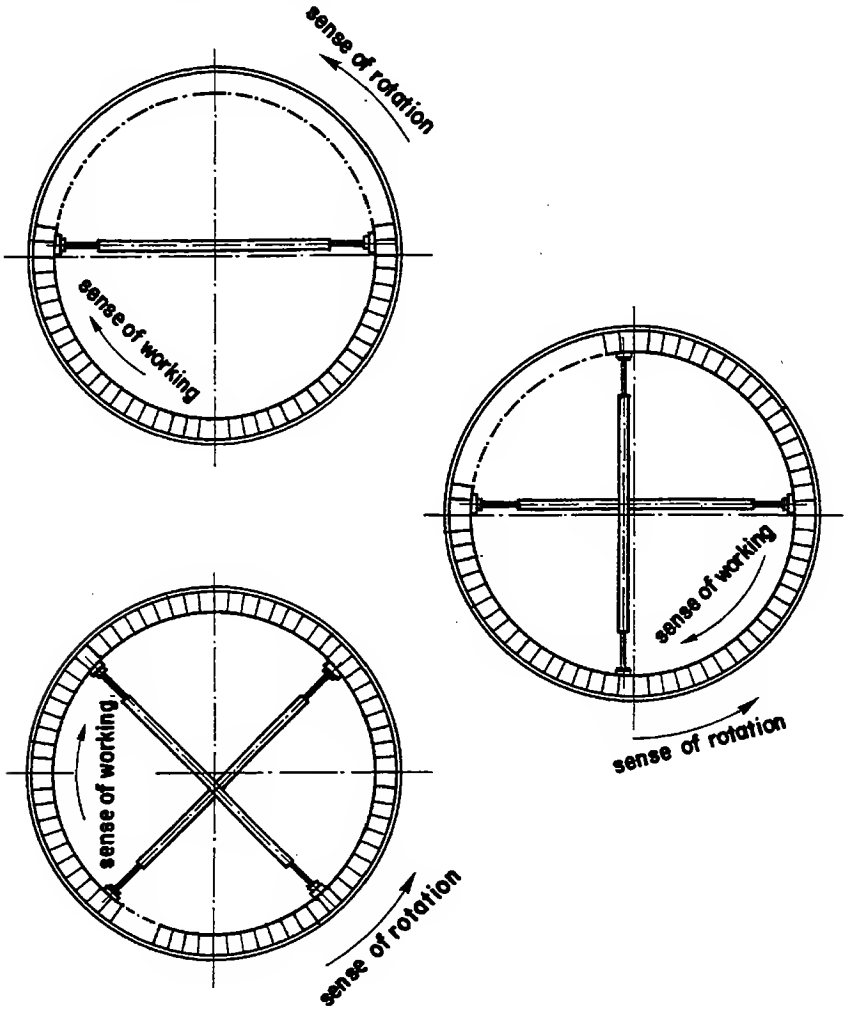
Fig. 10

Screw Jack Method



Lining Procedure

1) Lining of slightly more than the bottom half of the kiln.
2) Fastening of the lining by means of screw-jacks and wooden wedges. The distance of the screw-jacks is 0.5 - 1 m, depending on kiln diameter.
3) Turning the kiln by one quarter of circumference.
4) Lining of the third quarter.
5) Fastening according to point 2.
6) Turning of the kiln until the last section is on the bottom
7) Lining of the last section and keying with closure bricks.

Disadvantages of the Screw-Jack Method

- Kiln has to be turned during lining.
- With large kilns there is a danger of the screw-jacks slipping and the shell being distorted.

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Average Lining Speed with the Screw-Jack Method

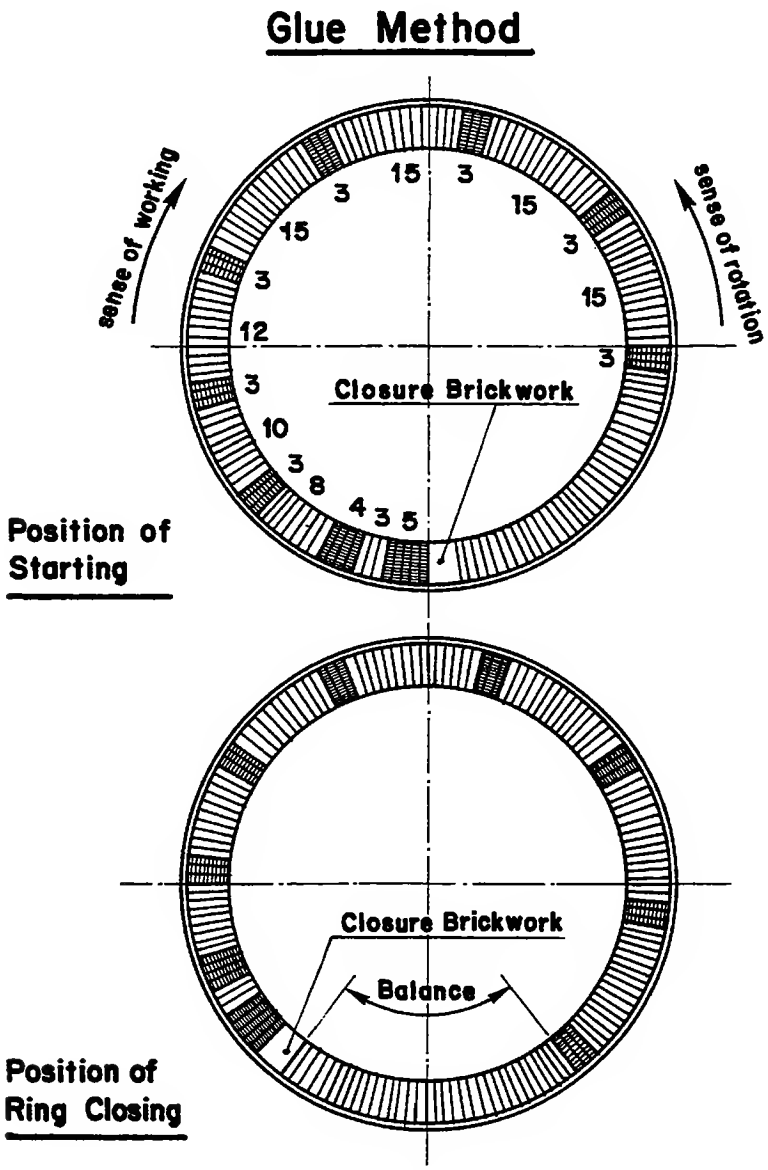
- 1 to 3.5 m^2/h or 0.17 to 0.25 m^2/man-hour

3.4.1.2 Gluing Method

With the gluing method the bricks are glued to the shell in an alternative manner, following a specific pattern according e.g. Fig. 11.

The glues used are two component expoxy or polyurethane resins and for very low temperatures down to minus 25°C there are synthetic resins based on polyesters or polyacrylates.

Fig. 11



The glues decompose after heating up above 200°C and lose their effect.

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Lining procedure:

1) Cleaning of the shell down to the bright metal, if necessary with the aid of a grinder. The surface must be free from dust or grease.
2) Gluing of the first lining sector of approx. 5 brick rows.
3) Installation of the next sector without glue etc. The number of bricks between the glued sectors depend on brick weight, kiln diameter, glue properties, etc. Generally the brick and/or glue suppliers specify the gluing pattern. As a rule approx. 20% of the bricks laid are glued. Glue consumption amounts to approx. 0.4 to 0.8 kg/m^2 of lined area or approx. 2 kg/m^2 glued area.

Using the gluing method, the following points require careful consideration:

- The start to be glued must be fitted together before the adhesives start to set.
- The kiln should not be turned before the glue has developed sufficient strength.
- The glue must be carefully prepared according to the suppliers specifications, particularly with respect to mixing ratio.

Disadvantages of the gluing method:

- Kiln has to be turned during lining.
- Tightening of the brick rings before keying is not possible, but only the bricks between two glued sectors. Thus, lining mistakes are not easy to be localised.
- Safety risk if kiln is turned before glue has developed sufficient strength.

With the gluing method, average lining speeds of 3 to 7 m^2/h or 0.4 to 1.2 m^2/man-hour can be achieved.

3.4.2 Formwork Methods

The formwork methods can nowadays be considered as the standard lining method. They are safe in operation and permit very accurate installation of the bricks.

With the curved formwork methods, first the lower half shell of the kiln is lined without any particular aid, and then, the upper half ring by ring with the aid of the curved formwork. There is a wide choice of equipment, form simple wooden forms to hydraulically operated mechanised forms.

3.4.2.1 Pogo Stick Method

The main equipment for the pogo stick method is:

- supporting construction
- working scaffold
- wooden curved drum
- spring loaded pogo sticks

The working scaffold is installed upon the lined bottom half of the kiln. Then the bricks of the upper half section are installed and fixed by means of the pogo sticks. Generally two bricklayers and two helpers line from each side towards the top position. After keying, the pogo sticks are removed and the formwork is put into the new position.

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3.4.2.2 Wooden Curved Form Method

The main equipment for the wooden curved form is:

- supporting framework
- working platform
- wooden curved formwork
- wooden wedges

The supporting framework is so constructed to permit a fork lift vehicle to move beneath the working platform, lifting the brick pallets to the platform. It is installed on the lined bottom half of the kiln. The bricks of the upper section are then laid on the wooden arch and pressed against the shell by means of wooden wedges. Work is done from both sides towards the top by two men on each side. After keying of a ring the curved formwork is moved into the next position, thereby, normally the wooden wedges are loosening themselves.

3.4.2.3 Mechanised Curved Forms (Fig. 12)

Mechanised curved forms are mechanised variations of the simple arch forms. They basically consist of a travelling working platform on a supporting framework. The curved form is installed on the working platform and is equipped with mechanically or pneumatically operated pressure tools by which the bricks are pressed against the shell. The supporting frame is so constructed as to allow the passage of a fork lift vehicle.

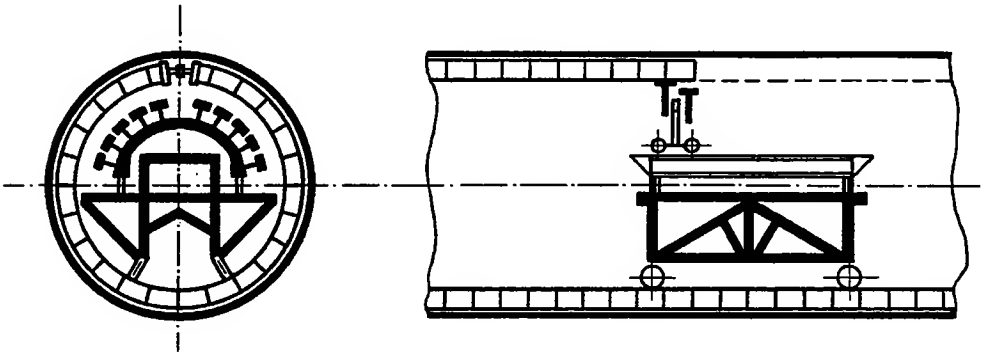
The most used mechanised lining machines are the Mult-O-Ring, the DAT-Appartus and the Reintjes lining machines. The Mult-O-Ring is equipped with two parallel rows of pressure tools which almost doubles the speed of work.

The lining procedure is similar to that with simple curved forms, except that the fastening of the bricks is mechanised.

With mechanised lining machines very high lining speeds of up to 6.5 m^2/h or 0.5 m^2/man-hour for mortared linings and up to 9.5 m^2/h or 0.75 m^2/man-hour for dry lining can be achieved.

Fig. 12

Arch Method



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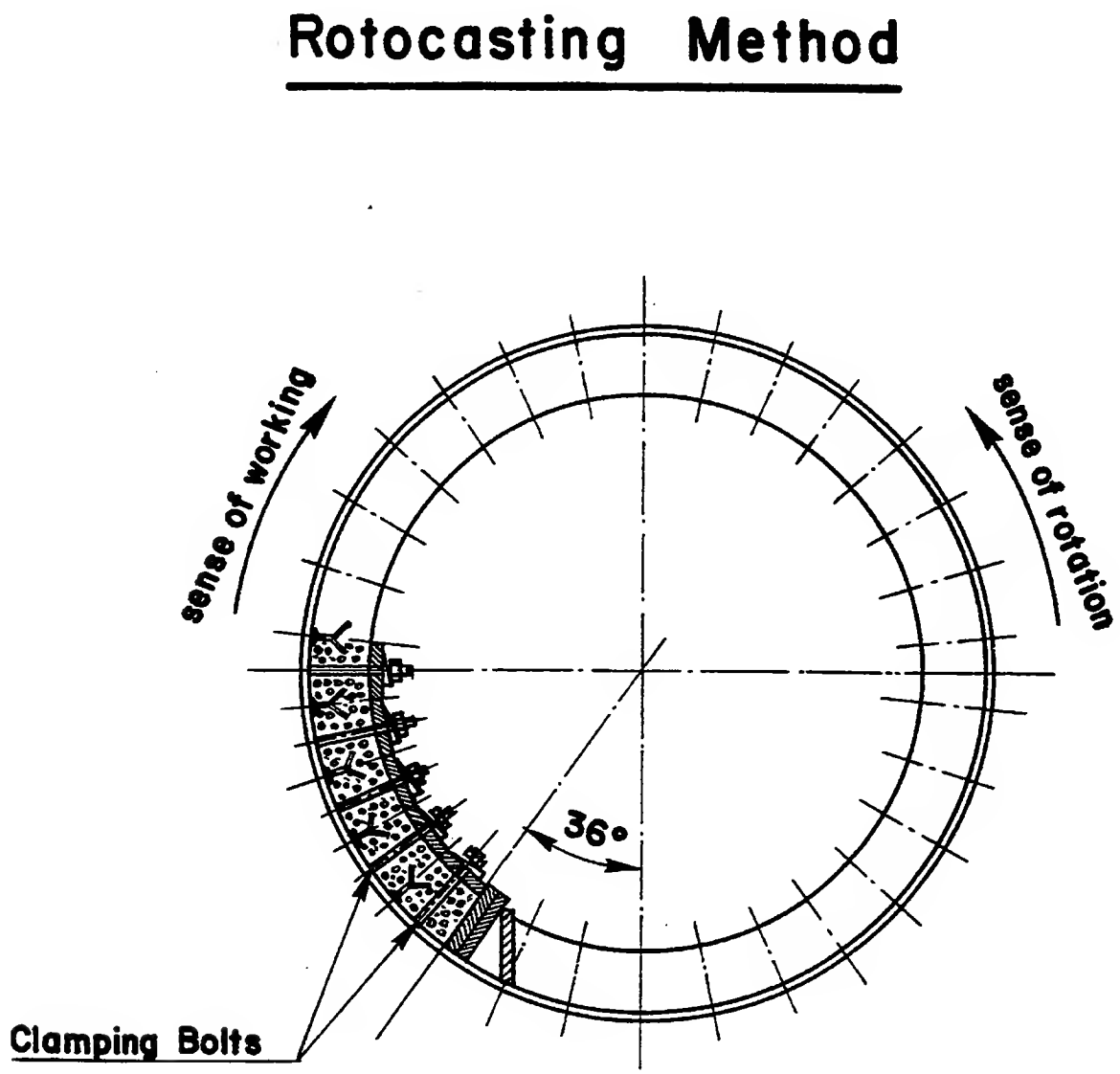
3.4.3 Lining with Unshaped Materials

Unshaped refractory materials can be installed by casting, gunning or ramming. The installed masses are fixed by metallic anchors which provide a good lining stability even in cases of local damages or spallings. Usually V-type anchors, generally delivered by the refractory supplier, are welded in a cross pattern on the shell. About 15 to 25 anchors per m^2 are required, depending on lining thickness.

3.4.3.1 Castables

Castables or refractory concrete contain a hydraulic setting agent (high alumina cement). They are installed generally in suitable formworks, e.g. by the rotocasting method (Fig. 13) and compacted by vibrating. In order to avoid spalling due to quick water evaporation during heating up, it can be advisable to perforate the surface by pushing steel wires of 5 mm thickness into the freshly placed castable at a distance of approx. 5 cm. The steel wires are to be removed after hardening of the castables. During the hardening period just after installation of the castable, its surface should be protected against too quick evaporation of water by means of wet cloths or water spray.

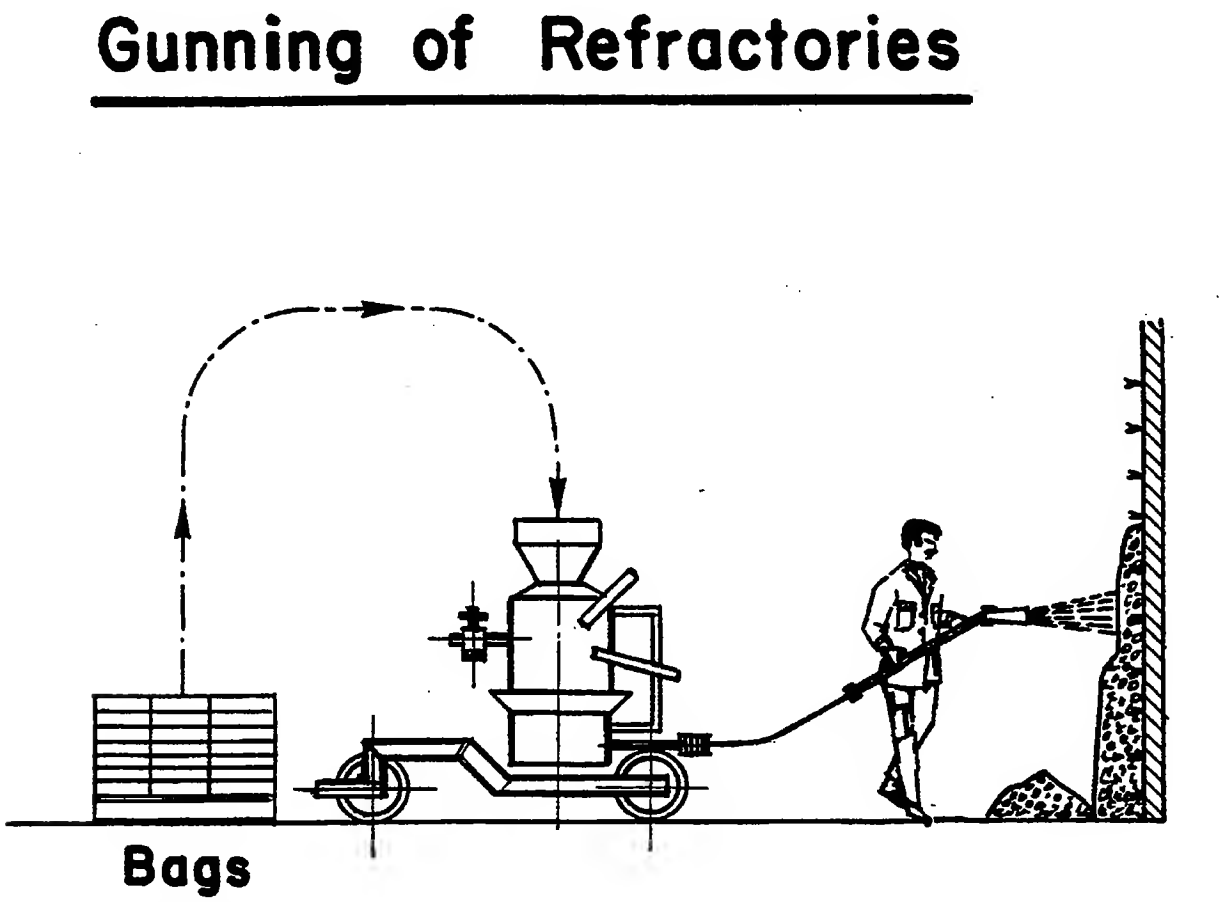
Fig. 13



3.4.3.2 Guniting Materials

Guniting materials are supplied either with chemically or with hydraulically setting bonding agents. The materials are placed pneumatically, passing through a hose to a mixing nozzle where water is introduced (Fig. 14).

Fig. 14



Guniting should always be done by specialised personnel, since quality of the lining as well as amount of losses in rebounding is highly depending on the ability of the operator. Depending on shape and position of the working surface losses in rebound amount from 10 to 50%.

During the hardening period, the gunned surface must be protected against water evaporation similar to castables. Also perforation of the surface by means of steel wires can be advisable.

3.4.3.3 Ramming Material

Ramming materials are plastic or granular materials, generally chemically, seldom hydraulically bonded.

Due to the low water content, rammed refractories generally have low porosity, high volume stability and strength and good resistance against chemical attack.

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The masses are placed by compacting layer by layer of approx. 25 to 50 mm thickness each by means of a vibrating hammer fitted on a compressed airhammer. Appropriate and homogeneous compression is necessary for good operating performance of rammed materials, therefore, highly qualified personnel is required for placing.

Chemically bonded materials should be installed only shortly before heating up. Spare parts lined with chemically bonded rammed materials must be tempered at approx. 250°C. After tempering, the lining surface must be protected against humidity by bituminous paint, in order that they are not damaged by absorbing water during storage.

3.5 Stripping and Removing of Old Lining

Stripping and removing of old lining and coating manually requires approx. the same time as the relining of the same zone with new bricks. By use of mechanised stripping machines, this dangerous and time consuming operation can be done much safer in considerably shorter time with less personnel.

Manually, 6 to 7 men remove approx. 3 to 5 m^2/h of old lining. By means of mechanised stripping machines, 2 to 3 men remove approx. 15 to 18 m^2/h.

The dismantling unit consists generally of a drilling machine chassis with an all-round pivoting arm on the end of which a compressed airhammer or drilling device is mounted (Fig. 15). For removal, small front-end-loaders are used. The shovel is shaped to fit the curve of the kiln shell.

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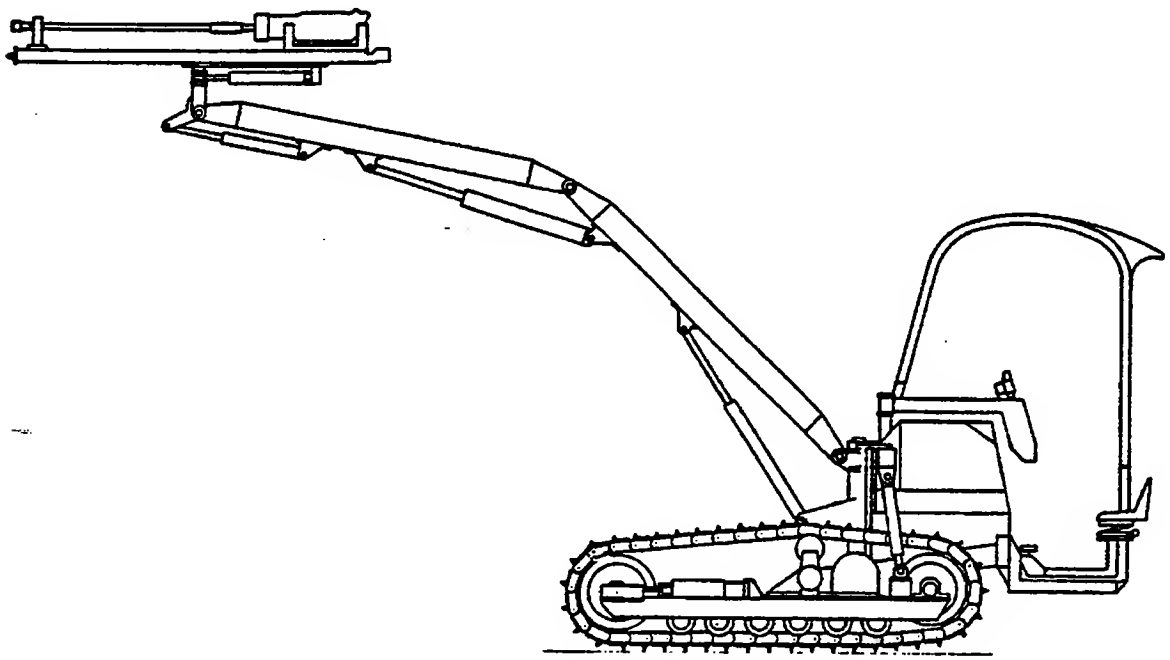
Fig. 15



Atlas Copco Refractory Peeler

The stripping procedure is generally done in the following steps:

1) If the coating is more than 25 cm thick it should be cleared away before the actual lining is to be opened up.
2) In small kilns of < 4 m diameter the removed coating must be transported away before the stripping of lining starts.
3) The process of opening up a slit should be done very careful in order not to damage the steel shell.
4) When the lining has been opened up, actual stripping can commence. Due to the opening already made in the lining, the stripping tool can more or less work parallel to the shell, thereby causing no damage.
5) After the upper half of the kiln is stripped, the kiln is rotated so that the remaining brickwork fall down.
6) The rubble can now be carted away by means of the front-end-loader.

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4. LINING OF PREHEATERS, KILN HOODS AND COOLERS

4.1 Suspension and Grate Preheaters

The lining of preheaters should have good insulating properties, therefore, basically two or three layer linings are installed. The working lining generally consists of abrasion resistant fireclay bricks with low porosity. In case of alkali attack acid firebricks with SiO_2-content above 65% are installed. Monolithic linings are used particularly for complicated shapes where the installation of bricks is difficult and expensive. For plane walls and gas ducts lining with bricks generally is easier and cheaper than lining with unshaped materials.

A three layer insulating lining is generally composed of a backing of ceramic fibre board, an intermediate layer of insulting firebricks and the dense working lining.

The bricks of the working lining are usually placed with chemically setting mortars in order to obtain gas tight linings. For compensation of thermal expansion, expansion joints are installed horizontally and vertically at distances of approx. 2 m, as well as in corners. The expansion joints have a width of approx. 15 to 20 mm and should be staggered and filled with ceramic fibre mats in order to avoid penetration of dust (Fig. 16).

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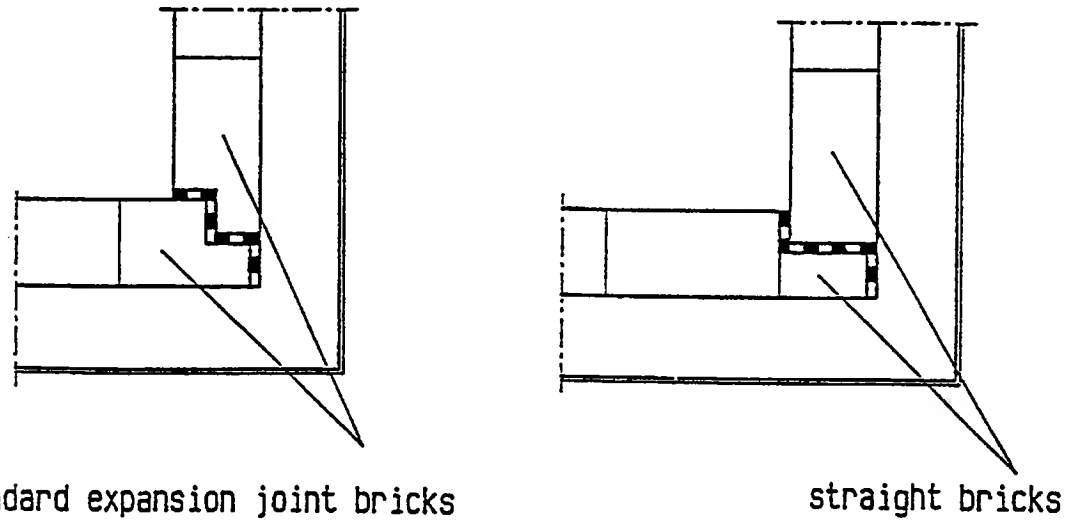
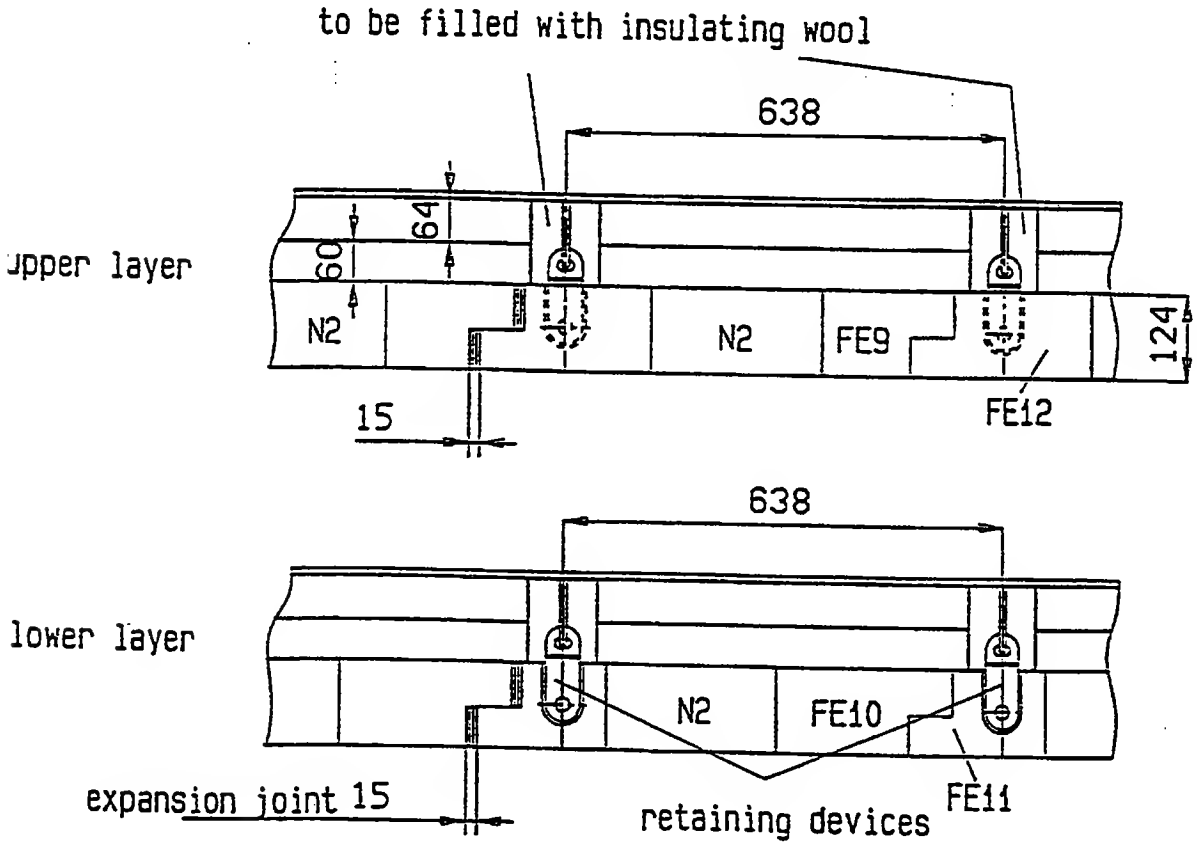
Fig. 16





Expansion Joints in Straight Walls

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For stabilising of the linings, anchor bricks and holding irons which are flexibly fitted with the steel shell are used.

Roofs are either lined vaulted without use of anchors or flat with hanger bricks and holding irons. Suspended roofs are often lined combined with hanger bricks and unshaped materials.

Pre-heater cyclones usually are lined with bricks and insulating back-lining, except the complicated shaped parts as e.g. cyclone inlet and control openings, which generally are lined with unshaped materials.

Meal chutes should be lined with highly abrasion resistant materials. Usually dense castables with metallic anchors or prefabricated materials are installed.

4.2 Kiln Hood

For kilns equipped with planetary coolers, refractory lining in the hood section is limited to the hinged or otherwise movable kiln door with openings for burners, measuring instruments and observation. The most frequently used lining material is dense castable fireclay together with insulating backing.

The lining of the kiln hood of kilns equipped with grate or rotary coolers generally consists of a dense working face and an insulating backing. The working lining must possess high thermal shock resistance and abrasion resistance. In the upper region, where the lining is exposed to direct flame radiation, alumina refractories with Al_2O_3-contents of up to 70% are used. The lower region is usually lined with dense fireclay. As insulating linings conventional insulating bricks as well as insulating gunning mixes or castables are used.

4.3 Clinker Coolers

(Fig. 17)

Approximately 15 to 35% of the length of planetary coolers is refractory lined. This lining should have a high resistivity against thermal shock and abrasion. For the cam lining section dense, high alumina bricks with high hardness and strength are used. The inlet bends are usually lined with dense high alumina, or even pure Corundum castables. For simpler geometric shapes conventional bricks of suitable quality can also be used.

In rotary coolers, about 50% of the tube length is lined with refractory materials. In the inlet zones, similar as in planetary coolers, refractories of high strength fireclay are normally installed.

The refractory lining of grate coolers, with the exception of the areas immediately above the grate (where direct contact with clinker occurs), is exposed to far a lower extent to thermal fluctuations and abrasion. Normally a two or three layer refractory wall and roof is installed, consisting of an insulating rear and a dense fireclay working lining.

The wear zones immediately above the grate consist of particularly abrasion resistant bricks or castables.

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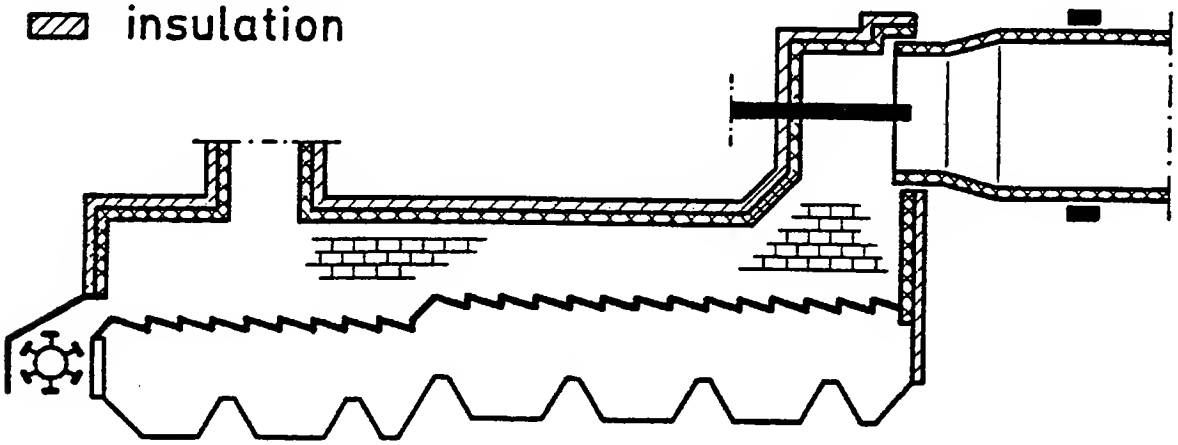
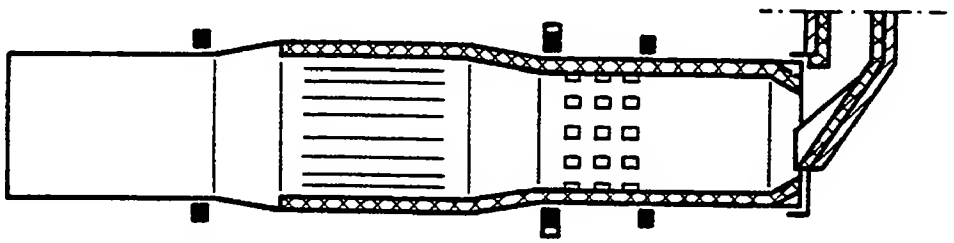
Fig. 17

Clinker Coolers

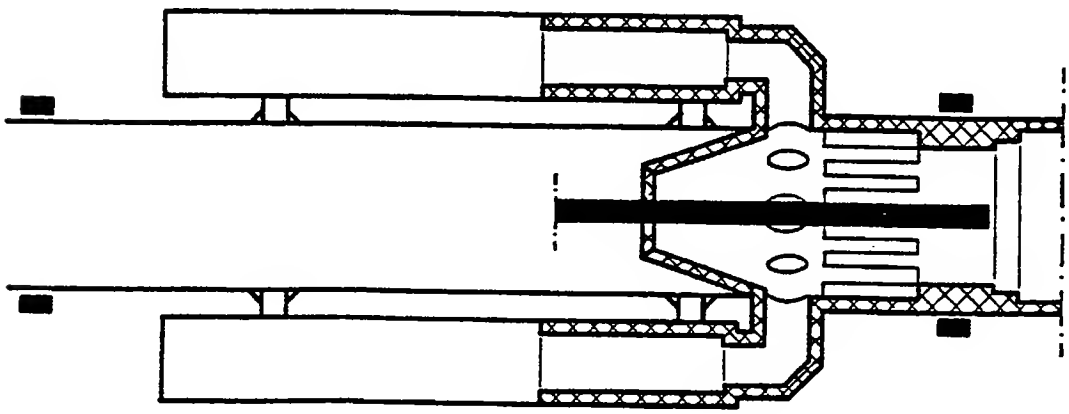
working lining
insulation



Grate Cooler



Rotary Cooler



Planetary Cooler

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5. OPERATIONAL ASPECTS REGARDING KILN LININGS

5.1 Heat-Up of Rotary Kiln Systems

The heating-up program, which mainly determines the amount of kiln temperature increase per unit of time and the timing of the raw material feed, must consider a number of factors such as type of refractory material, design of kiln system, mechanical systems etc.. The optimum heating program constitutes a compromise which best satisfies the various requirements.

Optimum Heating Period:

- Refractory Materials
  - Material quality
  - Type of bond
  - Installation method
- Riding Ring
  - Riding ring clearance
  - Riding ring dimensions
  - Monitoring capabilities
- Exhaust Gas Flow
  - Temperature profile in kiln
  - Permissible temperatures
  - Environmental regulations

5.1.1 Requirements from a Refractory Materials Viewpoint

The minimum heating-up duration from the refractory materials viewpoint is given by the type of special bricks and castables installed. Generally, the heating-up specifications of the suppliers must be respected. The minimum heating-up time is indicated by that material which requires the longest heating-up period because the minimum heating-up time can generally be extended but not shortened without causing damage. Exception: basic brick linings tend to hydrate when kept at temperatures between 100 and 450°C for several days.

Generally, the heating-up time is influenced by the following materials:

- Castable refractories with hydraulic bonding (Fig. 18)
  - Approx. 24 hrs setting time at room temperature
  - Approx. 24 hrs drying time at 100 to 200°C
  - Heating-up at the rate of 25°C/h up to 500°C (expels crystal water)
  - In subsequent heating-up cycles, operational temperatures may be reached more rapidly. The speed of reaching operational temperatures might be limited by other factors as e.g. the tyre situation.
- Castable refractories with chemical bonding
  - Approx. 8 to 16 hrs drying time at 100 to 200°C
  - Heating-up at the rate of 20 to 30°C/h up to 700°C (until chemical bonding is completed)
  - In subsequent heating-up cycles, operational temperatures may be reached more rapidly, as explained above.

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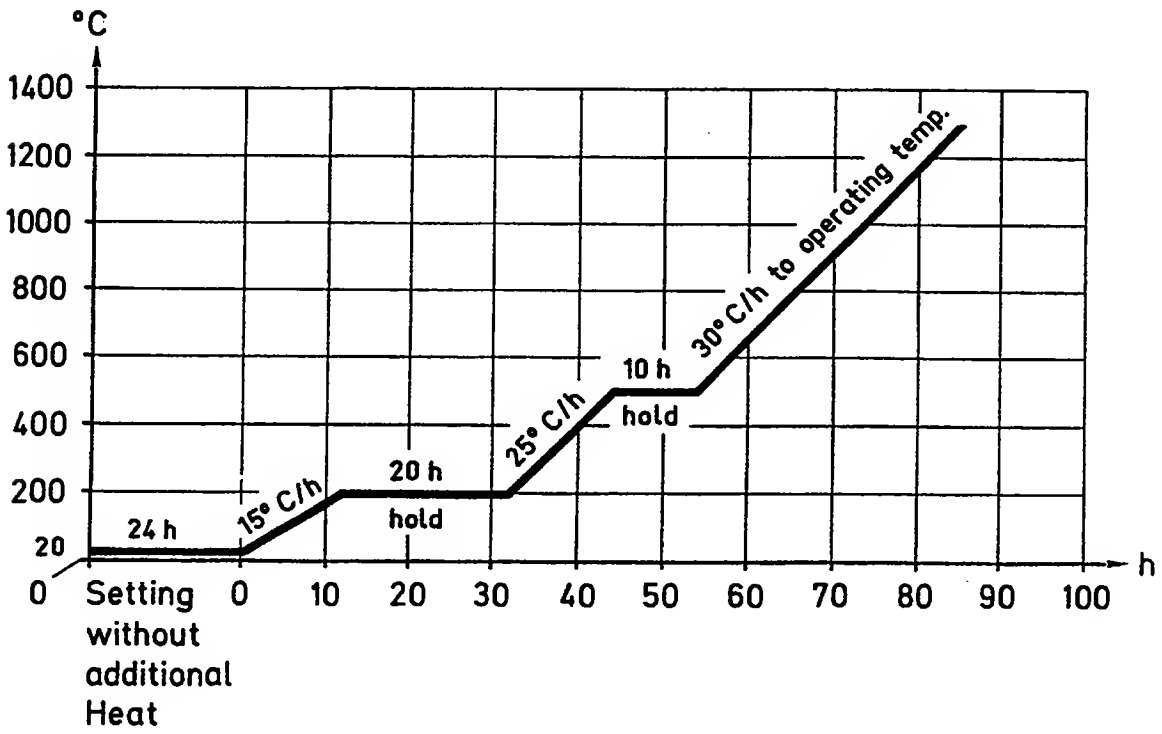
- Untempered, phosphate bonded bricks
  - Heating-up to 700°C (termination of chemical bonding) at the rate of 30°C/h
  - In subsequent heating-up cycles, operational temperature may be reached more rapidly, as explained above
- Jointing materials (mortars)
  - Theoretically the same rules would apply as for casting mixes; however, the situation is not critical because of the limited joint thickness.

Fig. 18



Example of Heating Curve for Castable

Based on the above listed criteria, most suppliers of refractory materials recommend for the start-up of new kilns, and after major overhauls with monolithic linings, a maximum heating-up rate of approx. 25°C/h, in the sintering zone, up to a sintering zone temperature of 900°C. (Whereby a previous drying of heat exchanger systems and monolithically lined kiln zones is assumed.)

After reaching 900°C, heating -up can continue at the rate of 50°C/h up to working temperature.

After minor repairs and after shut-downs not caused by defective lining, refractory materials suppliers generally recommend a heating-up rate in the sintering zone of 50°C/h.

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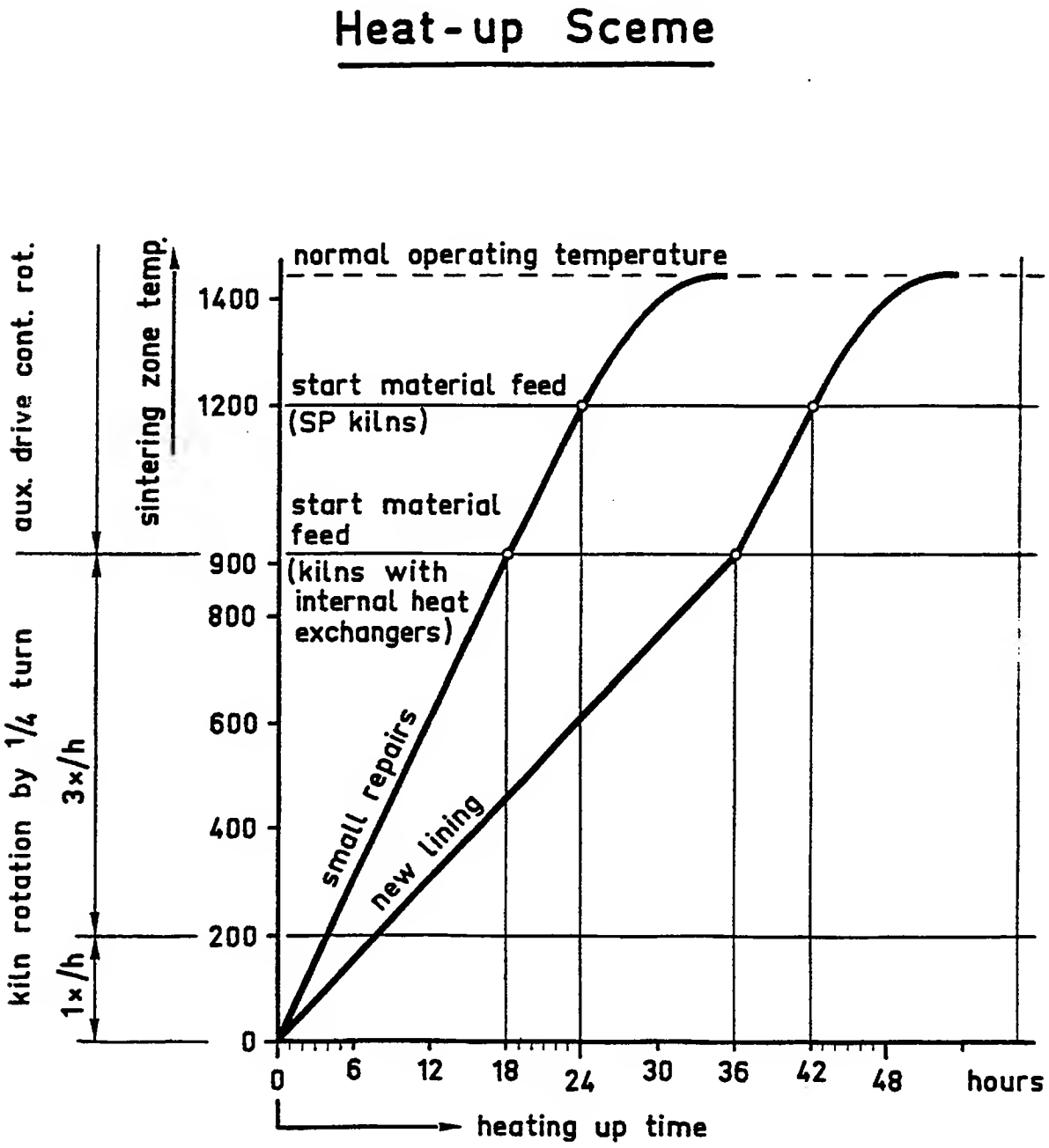
An example of heating-up scheme is shown in Fig. 19.

Fig. 19



Faster heat-up times are sometimes reported. Some Japanese suppliers heat-up new kilns in 5 to 8 hours without any damage to lining and kiln system. Such procedures can, however, not be recommended in general, i.e. without considering the entire situation of lining quality and mechanical details of kiln design. Since expansion joints are dimensioned to produce optimum lining stability under standard working conditions, rapid heat build-up may cause spalling due to excessive mechanical stress of the brick surface because the lining face heats-up more rapidly than the kiln shell.

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5.1.2 Riding Rings (Floating Type)

From a riding ring point of view, the minimum heating-up period is determined by the time required to stabilise the temperature difference between the kiln shell and riding rings. The shell will heat-up more rapidly than the tyre. The critical temperature difference results in a constriction of the kiln shell, which will cause permanent deformation if the yield strength of the kiln shell material is exceeded. This in turn will cause excessive play in the riding ring after the normal working temperature is regained, as well as increased ovality of the kiln shell, a factor which may contribute to excessive refractory lining wear.

As a general rule, the riding ring creep, i.e. the relative motion between riding ring and kiln shell should be monitored at regular intervals or, even better continuously, at least during the heat-up cycle. Various measures may be taken to protect the riding ring section from overheating.

With imminent danger of seizure, the heating-up process should be slowed down or interrupted until a measurable amount of relative movement is again present. For this reason, the tyre creep may become the limiting factor in determining the heating-up rate.

5.1.3 Exhaust Gas Control

Temperature Gradient in Kiln System

Already during the heating-up process a temperature gradient is sought which at the start of the raw meal feeding to the kiln is equivalent or similar to the temperature gradient prevailing under normal operating conditions. This condition can be approximately reached under ideal heat-up conditions and with correct flame patterns, because in a dynamic heating process, the cold kiln system is heated along the flow direction of the exit gas while the exit gases are simultaneously cooled. With optimum fuel addition, this procedure is ideally continued with material feed by preheating the raw meal in the counterflow. In practice, however, such a smooth transition is not possible because the amount of material fed can, for mechanical reasons, not be reduced to indefinitely small amounts and because the heat transfer to the meal is governed by endothermic and exothermic reactions.

For correct temperature gradients, fuel combustion at lowest possible amount of excess air is a necessity. In this manner, the temperature gradient between gas and lining and the dwell time of the gases within the kiln system will result in optimum absorption of the available heat by the refractory lining.

5.1.4 Shut-down of Kiln Systems

The procedure for shutting down a kiln depends mainly on whether it is a planned shut-down or an emergency stop due to a breakdown.

To protect the lining, the cooling rate should, if possible, not exceed the maximum permissible heating rate, i.e. approx. 50°C/h measured in the sintering zone. Excessively rapid cooling may cause damage to the refractory lining due to thermal shock.

After stopping raw meal and fuel feed, the exhaust gas damper is closed and the exhaust gas fan is stopped. Closing the exhaust gas path prevents gas circulation within the kiln and the temperature profile is more or less maintained. The cooling rate of 50°C/h will initially be reached by radiation alone. Only after a dull red heat is reached in the sintering zone cooling should continue with a light draft until the end of the cooling phase is reached.

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5.2 Kiln Shell Deformation

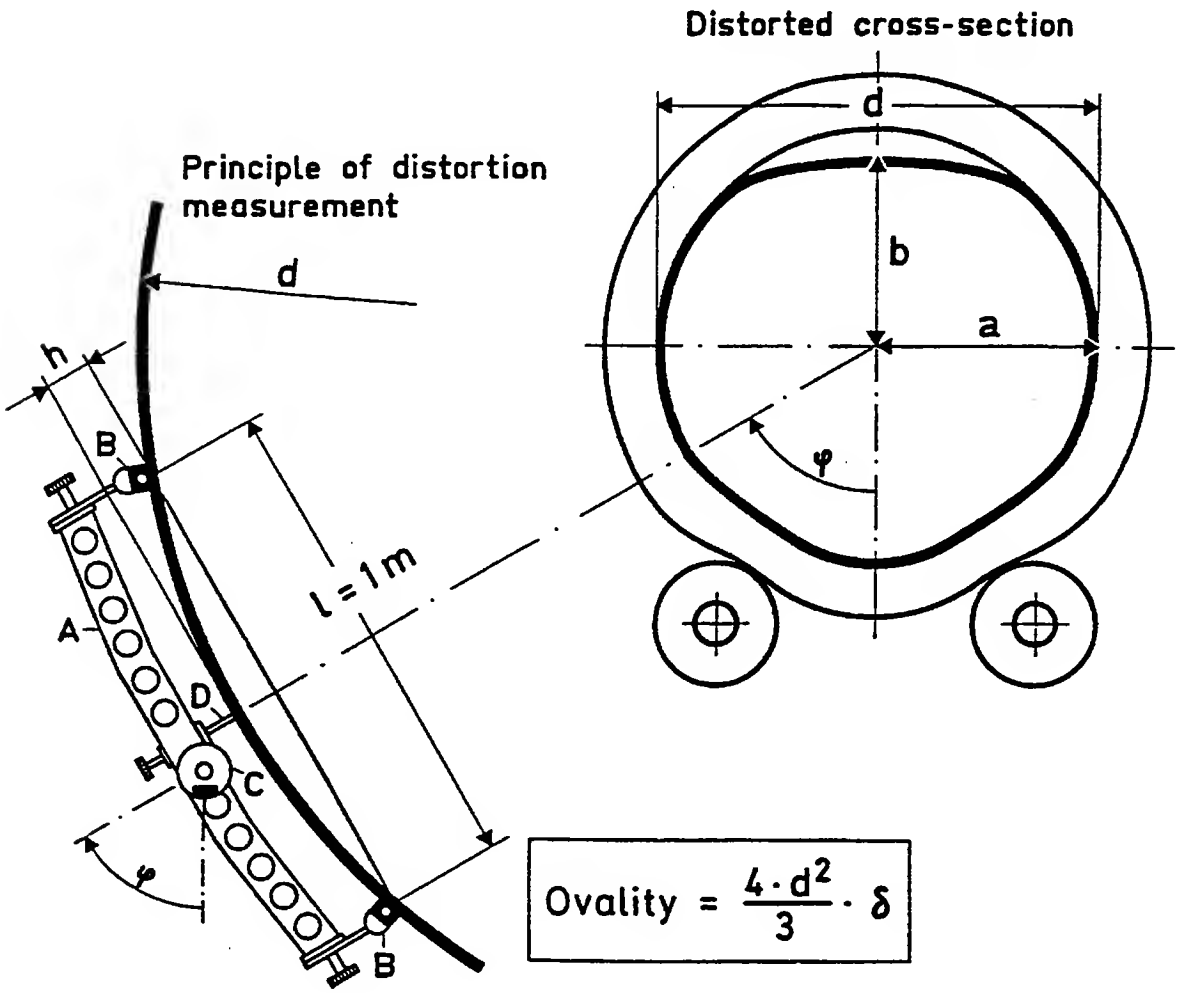
The refractory life and coating stability in the area of kiln tyres is significantly influenced by kiln deformation. As a result of changes in the kiln shell radius, there is a movement between the bricks which results in local surface pressure, leading to rapid wear and possibly total lining collapse.

The measurement of kiln shell deformation can be made during kiln operation with the aid of the 'Holderbank' Shell-Test Gauge.

The measuring principle is illustrated in Fig. 20.

Fig. 20

Shelltest measuring unit
«Holderbank» System



The yoke (A) of the Shell-test gauge covers a circular section with a chord of length of 1 m and height h.

During kiln rotation the chord height h changes with the continuously changing curvature radius r of the kiln shell.

This change is transferred via a plunger (D) and recorder (C) to produce a polar diagram (shell-test diagram).

The closed line drawn on the diagram represents a geometrically similar figure to the deformed kiln shell cross section.

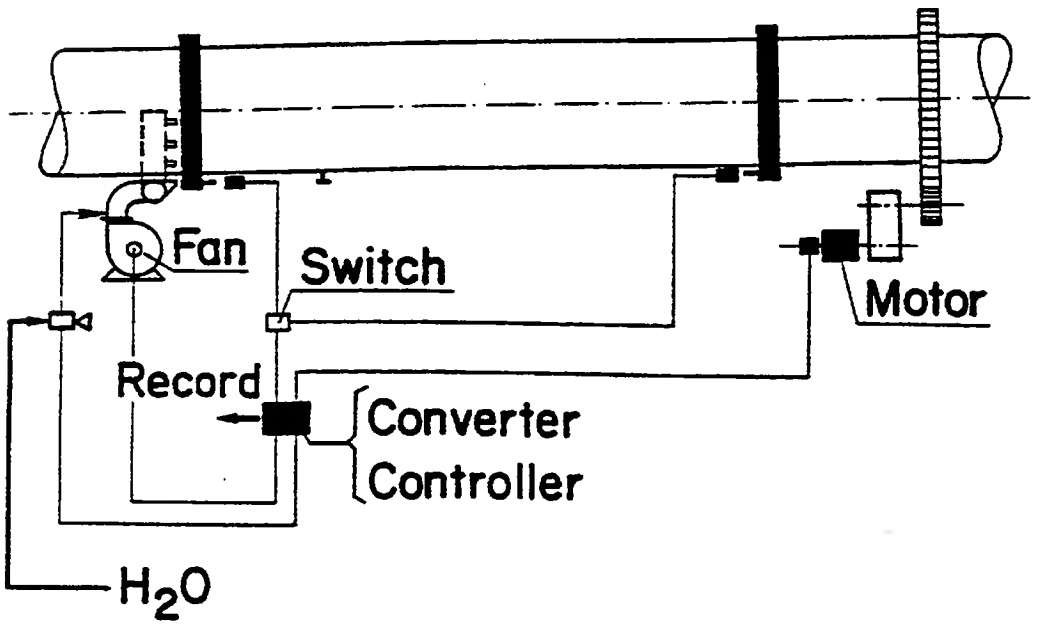
The subsequent calculation is based on determining the difference between the longest and the shortest radius of the shell-test diagram after which the ovality is calculated according to the method of G. Rosenblad.

The mechanical condition of kiln shell and kiln tyres with respect to allowable ovality should be regularly checked.

For kiln with loose tyre design, an indication on the mechanical condition of the tyre section can be gained by continuous measurement of the difference in rotational speed of the kiln tyre and kiln shell. This method permits continuous indication/recording in the control room (Fig. 21).

Fig. 21

Tyre Creep Recorder



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5.3 Fluctuating Process Parameters

Unstable burning conditions of cement kilns belong still to the most frequent reasons of refractory lining failures. All types of used sintering zone refractories show structural changes between hot and cold face, infiltrated circulating elements of faded brick sections by overheating, do not immediately result in refractory failures. However, if kiln operating conditions change, followed by changed thermal profile or changed coating conditions, the bricks break off in large lumps since, due to its altered structure, their mechanical characteristics have deteriorated.

Unstable burning conditions can be caused by

- Unstable kiln feed due to insufficient material flow control
- Fluctuating kiln feed composition due to insufficient homogenising of raw material
- Fluctuating flame length due to insufficient fuel flow control or fluctuating fuel quality
- Frequent kiln shut-down due to maintenance problems of auxiliary equipment, power failures or refractory failures.

Without going into details it is evident, that these points need to be carefully watched and, if required, improved, in order to establish the conditions which make the improvements in the field of refractory lining selection, installation and operation effective.

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6. CONCLUSIONS

Improvement of refractory lining lifetime is not a matter of one single criterion. There exist several fields of influence which often are jointly responsible for the results obtained.

The following list contains suggestions concerning the most important and general consideration for achieving improvement in lining life.

Since the major refractory problems occur in the sintering and transition zone, most of the considerations will concentrate on these kiln areas.

Most important factors:

1) Optimisation of kiln feed (qualitatively and quantitatively)
   * by selective quarrying procedures
   * by selection of optimum raw material composition
   * by appropriate homogenising of raw materials
   * by improvement of kiln feeding equipment
2) Optimisation of flame shape and length
   * by evaluation of the influence of adjustable flames
   * by the most constant burning possible with the optimally established flame shape
3) Improvement of mechanical condition of kiln shell in the tyre area
   * by adjustment to minimum possible tyre clearance
   * by continuously monitoring tyre creep
   * by cooling of kiln tyre areas when danger of seizure exists
4) Optimisation of brick quality in the sintering zone
   * by methodical investigations (trials) with alternative brick qualities
5) Thermal load reduction
   * by introduction of secondary firing (if applicable)
6) Reduction of the number of kiln shut-downs
   * by improving the mechanical and electrical reliability of the system
   * by installation of emergency power supply
7) Optimisation of the bricking process
   * by refining existing or selecting better methods
   * by improvement of working conditions or methods
   * by introducing clear instructions, control (bonus systems)
   * by optimised bricking policy
8) Prevention of rapid heating or cooling
   * by observing the important factors on kiln light-up
   * by prevention of rapid cooling on shut-down

This list is not claimed to be complete and some of the suggestions might not be realisable for a specific case on hand. As a check-list, however, it allows the identification of points of weakness and the establishment of further procedures in order to be able to define the actual problem and finally to solve it as effectively as possible.

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7. TEST QUESTIONS

1) Please make a sketch of the various zones of a suspension pre-heater-, Lepol -and a long wet kiln and indicate what type of refractories are used in each zone.
2) Explain the different methods you know of refractory brick installation in a rotary kiln (sketches).
3) Where are the unshaped refractories mainly used in cement kiln system?
4) What are the main features of brick joints (jointing materials) for rotary kiln linings in the case of alumina and basic bricks, considering dry and mortar lining?
5) Establishment of a heating-up program for cement kilns is mainly based on considerations regarding three main factors. What are these considerations?

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Chapter 4

Firing Systems

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2. Safe Handling of Coal and other Combustible dusts 275

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Firing Systems: Handling and Preparation of Noble Fuels

D. Pauling
PT 98/14353/E

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1. INTRODUCTION

Noble fuels are coal, fuel oil and natural gas. Handling and preparation of those fuels has to fulfill certain requirements in order to produce similar combustion conditions for those different fuels and avoid incomplete combustion, e.g. CO at kiln inlet or local reducing conditions due to combustion of fuel particles in the clinker bed.

For coal firing the main types of firing systems are described (direct, semi-direct and indirect firing). For pulverized coal dosing and transport to the burner the important design criterias are outlined. The required coal dust qualities for a good combustion in the cement kiln are described.

For fuel oil firing, preparation and heating systems are outlined. The required fuel oil qualities (pressure, viscosity and temperature) are given.

For natural gas firing, preparation and safety precautions are described.

Burners, injection characteristics and flames are not subject of this paper (see paper: "Burners and Flames").

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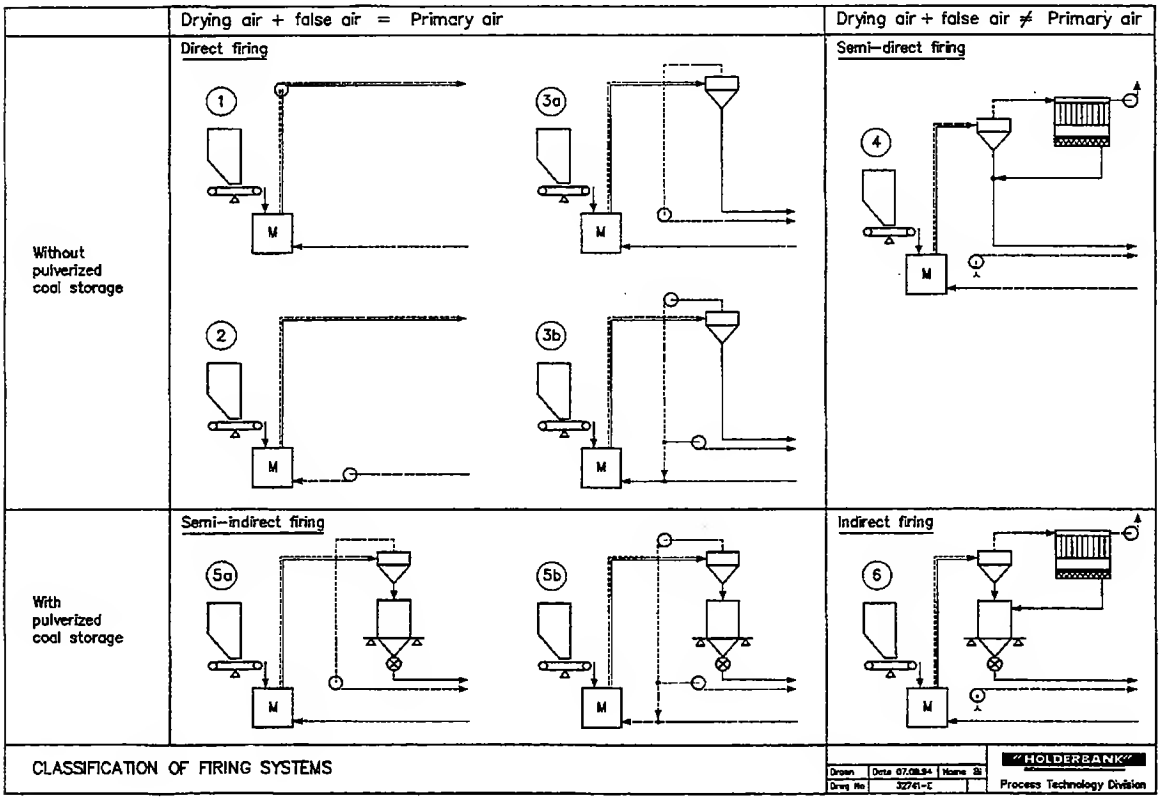
2. COAL FIRING SYSTEMS

Before the coal is fired, it has to be prepared according to the required fineness. The coal has to be dried to 0.5 - 1.5 % residual moisture content, since moisture in the coal means loss of calorific value, as the water has to be evaporated and heated up to flame temperature. Coal drying is done simultaneously with the grinding.

2.1 Classification of Coal Firing Systems

With reference to gas and material flow, the coal firing systems can be classified into four main groups which in total sum up to six individual systems (Fig. 1).

Figure 1: Classification of Coal Firing Systems



System 1 - Direct firing

Represents the most simple case. The coal is ground in the mill, dried and blown into the kiln together with the drying gases.

System 2 - Direct firing

Basically describes the same solution with the exception of the mill working under positive pressure. This solution is generally applied to protect the fan when processing abrasive coal.

System 3 - Direct firing

In system 3, the fan is protected by separating the pulverized coal in a cyclone and feeding it after the fan into the primary air stream.

System 3a - Direct firing with recirculation

Same as system 3, but with recirculating drying air. This arrangement allows reduced primary air ratios.

System 4 - Semi-direct firing

Has little technical significance since the solution with intermediate storage of coal would generally be given preference.

System 5a - Semi-indirect firing

With system 5a, the kiln can be operated independently of short mill shut downs since the pulverized coal is stored in an intermediary storage bin. The exhaust air from the mill enters the kiln as primary air.

System 5b - Semi-indirect firing with recirculation

Same as system 5a, but with recirculating drying air. This arrangement allows reduced primary air ratios.

System 6 - Indirect firing

In system 6, the grinding installation is completely separated from the kiln. The pulverized coal is stored in an intermediary storage bin and the exhaust air from the mill is released through a filter into the atmosphere. By this way, the kiln operation is completely independent from the combined drying and grinding operation.

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Major Advantages / Disadvantages of the Different Coal Firing Systems:

	Direct firing		Semi-indirect firing		Indirect firing
	Conventional	modified	Conventional	Modified	
	System 1 and 2	System 3a	System 5a	System 5b	System 6
Advantages	• Simple design • Low risk of explosions • Simple extinction of fire in the in the grinding system by stopping coal feed. No spread of fire into silos	• Lower primary air ratios and thus lower heat consumption compared to conventional • Independent primary air fan	• No exhaust gas, therefore no filter required, thus lower risk of explosions than with indirect firing • Short mill shut down not = kiln shutdown • Only one mill required for several kilns • Easy sampling for fineness control	• Lower primary air ratios and thus lower heat consumption compared to conventional • Independent primary air fan	• Simple flame control • Low primary air ratio • Water vapour from coal drying is not introduced into the kiln • Short mill shut down not = kiln shutdown • Only one mill required for several kilns • Easy sampling for fine-ness control
Disadvantages	• Combined operation with the kiln, therefore often not optimal operating conditions. • Mill shutdown = kiln shut-down • Number of kilns = number of mills thus reducuing the advantage of lower investment cost if several kilns are installed • High primary air ratio up to 30% • Slow control loops, long dead time • Sampling for fineness control difficult	• More complex installation requiring additionally: primary air fan, longer ducting	• More complex installation requiring additionally: cyclones , pulverized coal silos, pulverized coal feeders, measuring and control system • High primary air ratio up to 30%, during start up and shut down of grinding plant disturbed kiln operation • Investment cost higher than with a direct firing system (valid for one kiln only) • Risk of self-ignition of the pulverized coal in the storage silo	• More complex installation requiring additionally: primary air fan, longer ducting	• More complex installation requiring additionally dedusting filter • Investment cost higher than with a direct firing (valid for one kiln only) • More vulnerable to fires and explosions in gas ducts and filter • Risk of self-ignition of the pulverized coal in storage silo

Impact of Firing System on Kiln Operation:

- Direct firing systems tend to enhance coal fluctuations and therefore disturb combustion.
- If the mill vent air enters the kiln as primary air as it is the case with the direct- and semi-direct firing, the primary air ratio is higher than required for optimum combustion. At a given excess air factor, the primary air ratio has a direct influence on the heat recuperation efficiency of the cooler and finally on the overall kiln heat consumption. If the heat consumption can be reduced, the exhaust gas quantity is automatically decreasing, which offers potential for a capacity increase.
- Another very important advantage of lower exhaust gas quantities is the effect of decreased gas velocities in the kiln. This on the other hand has the benefit of lower dust generation for wet kiln systems.
- With a direct firing system the water vapor from coal drying enters the kiln with the primary air. The water vapor has no direct influence on the combustion process, but increases the kiln exhaust gas quantity accordingly. A water content of 15% in the coal increases the exhaust gas quantity of a dry process kiln by approx. 1.5% and of a wet process kiln by 1.2 %.
- At the same time, the flame stability may decrease as a result of dilution of the primary air.
- Flame shape is strongly influenced by the type of firing system. An indirect system will not only support a more stable flame, but also a shorter one, which results in smaller, more even distributed alite crystals with higher reactivity. Benefits are better quality of the clinker and a lower energy demand for the cement grinding.
- For new projects direct firing will not be selected anymore because of the above mentioned disadvantages. Today indirect firing systems are "State of Technology". Conversion projects from direct to indirect firing for existing installations can not always be financially justified on the basis of reduced thermal energy consumption. In countries with low coal prices, pay back times of several years must be expected. However, what can make a conversion project interesting, are the positive effects on kiln operation and thus product quality.

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2.2 Quality of Coal Preparation

Inadequate coal preparation (fineness) can result in both burn-out problems (CO formation) and the presence of fuel in the material bed (increased volatility of sulfur).

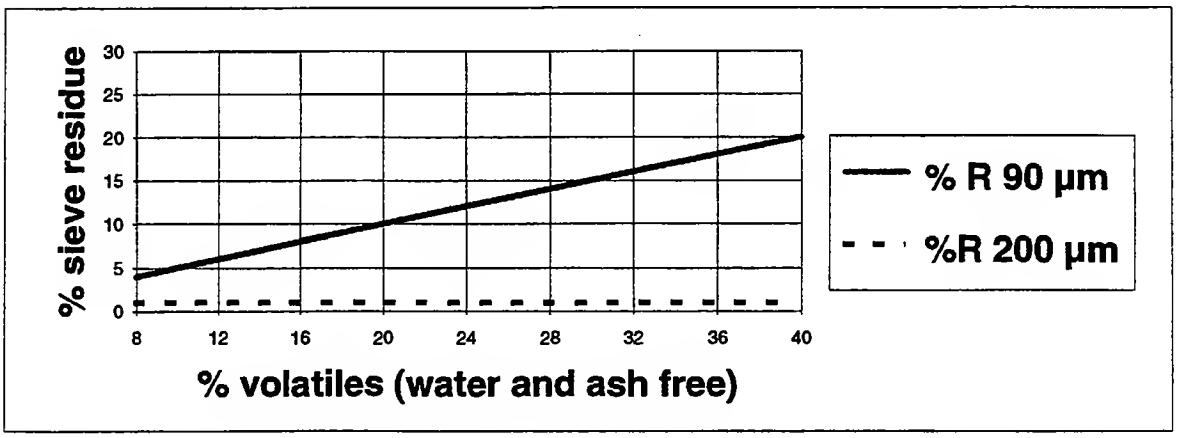
The combustion time of coal depends on the content of volatile elements.

Fig. 2 shows the principal requirements for coal fineness in function of the volatile content.

Figure 2: The Grinding Fineness of Coal in Function of its Volatile Content

Rule of thumb:

% R 90 µm = 1/2 (% volatiles)



The aim is to comply with the following simple rule as an upper limit:

- **Residue on the 90 µm sieve < ½ (% volatile components)**
- **Residue on the 200 µm sieve < 2%**

For low volatile and difficult to burn coal types such as petrol coke and anthracite, the above mentioned rule has to be tightened:

- **Residue on 90 µm sieve for petrol coke and anthracite < 5 %**
- **Residue on 200 µm sieve for petrol coke and anthracite < 1 %**

It has to be pointed out, that both values, the residues on 90 µm and on 200 µm are important. The 90 µm values influence flame length and CO formation, excess residues on 200 µm create reducing conditions in the material bed and can be responsible for increased volatilization of sulfur.

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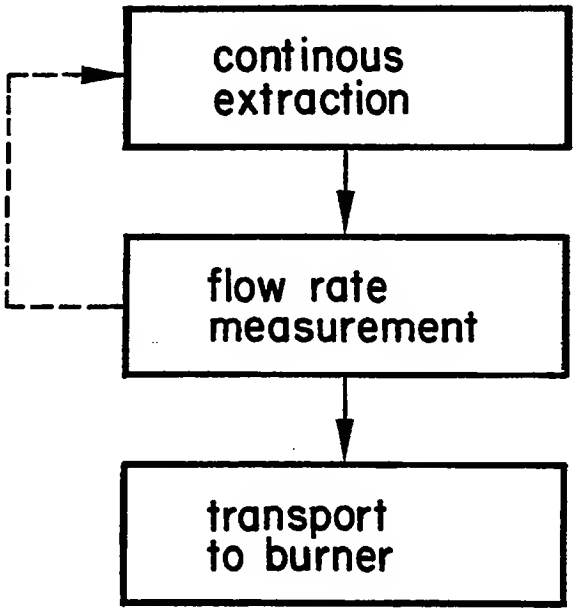
2.3 Pulverized Coal Dosing

For coal firing, in order to obtain perfect fuel feed, the entire feed system - from discharge from the coal dust silo, through weighing and dosing, to coal dust transport to the burner - must function as well as possible (Fig. 3).

Figure 3: Pulverized Coal Dosing



2.3.1 Feed Bins for Pulverized Coal

The feed bin design has a decisive impact on feed rate control. A feed bin design ignoring a product's flow characteristics may result in inconsistent discharge rates due to problems such as arching, erratic flow and flushing, conditions that can not be corrected by any feeder system.

Design of feed bin, activation and discharge:

- The capacity of the feed bin should be sufficient for at least 15 but not more than 60 minutes of kiln/precalciner operation
- The bin has to be designed for mass flow.
- The activated discharge opening section must be large enough to prevent bridging (at least 1200 mm in diameter for circular outlets and 600 x 1800 mm for slotted discharges).
- The discharge opening should be activated preferably by using mechanical discharge device such as paddle or agitator.
- Pulsed aeration systems for flow activation are only suitable for bins feeding loss-in-weight dosing systems. As a compromise aeration can help to solve discharge problems at existing bins, but should be avoided for new bins.

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2.3.2 Weighing and dosing of pulverized coal

Proper weighing and dosing requires a uniform coal dust supply (feed bin discharge; see above). It is necessary to distinguish between accuracy of weighing and short term variations.

The dosing system should meet the following requirements:

- Weighing accuracy: +/- 2% is normally accepted.
- Short term variations (referring to 10 sec. measurements): < +/- 1% (short term variations are responsible for CO peaks)
- Long term variations (approx. 10 min. to 1 hour): < +/- 0.5%
- Sensibility: < +/- 0.5%
  (Example: A dosing system with a maximum capacity of 5 t/h has to be capable to handle set point changes of +/- 25 kg/h).
- Adjustment range: 1:20 (of the maximum capacity).

The best indicator for the accuracy of the dosing is the oxygen level at kiln exit. Poor dosing of coal dust leads to big fluctuation of the oxygen concentration.

2.3.3 Most Common Pulverized Coal Dosing Systems

At present two systems are on the market which offer the best solutions for dosing pulverized coal.

- Rotor Feed Scale (Pfister) Fig. 4
- Coriolis Scale (Schenk) Fig. 5

Only second choice are the following systems:

- Loss-In-Weight System (complex setup requiring skilled maintenance)
- Impact-Flow Meter (limited accuracy)

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Figure 4: Rotor Feed Scale (Pfister)

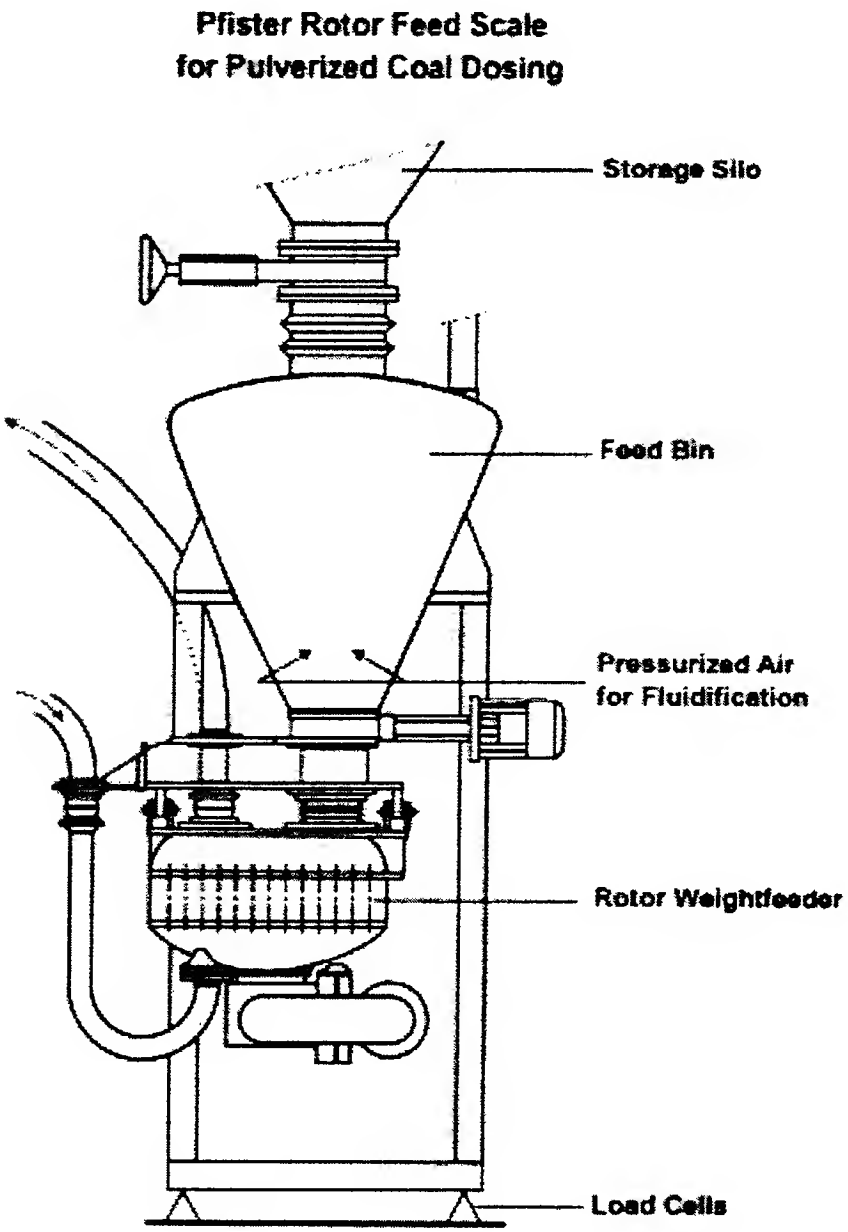

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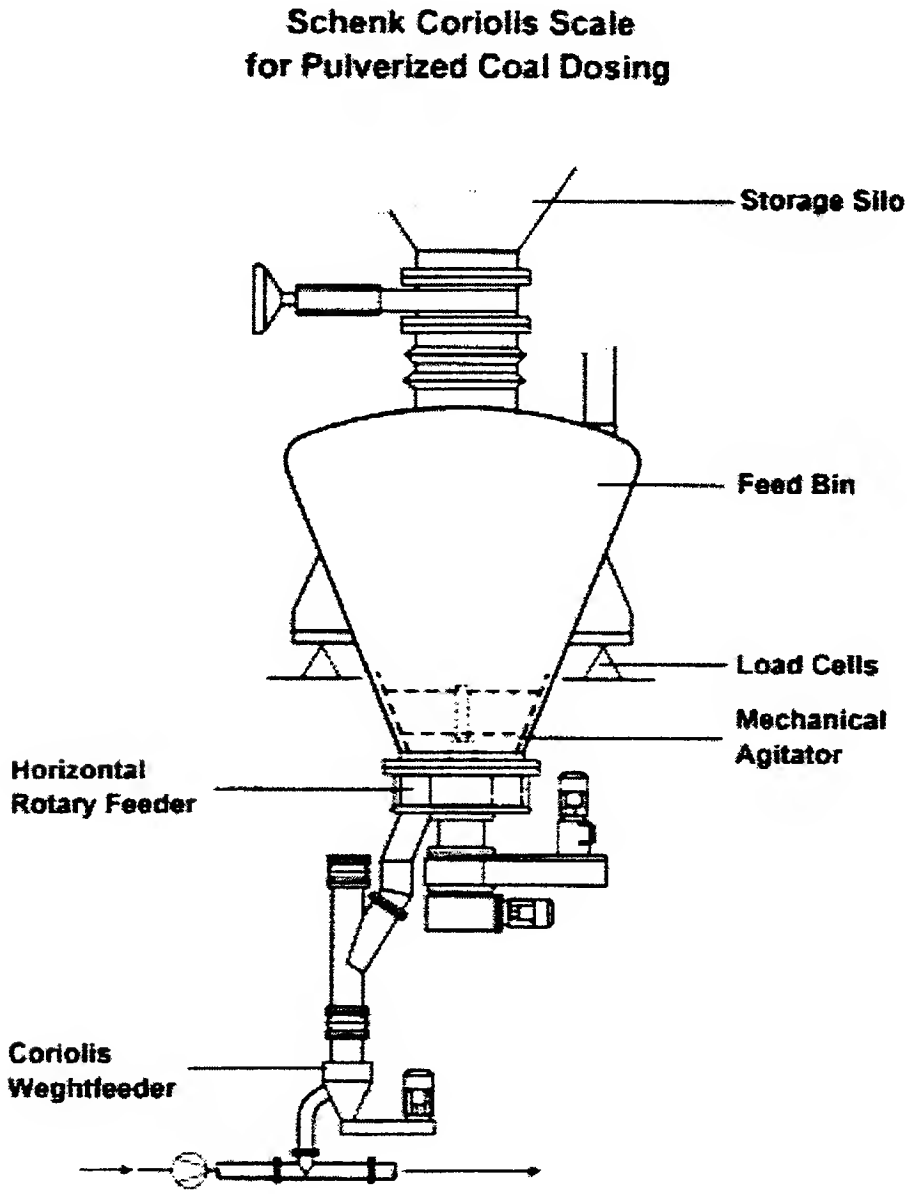
Figure 5: Coriolis Type Feed Scale (Schenk)



2.4 Pneumatic Transport of Pulverized Coal to the Burner

The highest accuracy of the dosing and feeding system is not useful for the kiln operation if the transport to the burner is not designed well.

What is required is a high accuracy at the feed point to the process. This means that a careful design of the pneumatic transport of the pulverized coal to the burner is of utmost importance too.

The coal dust transport should meet the following design criteria:

- Pneumatic transport velocity to burner is one of the most critical items for regular coal flow. To avoid pulsations caused by pocket formation in the pneumatic transport line, the transport velocity (from feeder to burner) should be in excess of 32 m/s.
- The fuel load carried by the air is not a critical value. Normal loads lie at about 5 kg/m^3, but values of up to 12 kg/m^3 are found without any operation problems.
- Fluctuations caused by the feeding device of coal to transport air (pneumatic pump, rotary air valve) have to be avoided by adequate design of the feeder (sizing, number, arrangement of rotary feeder cells, dedusting).
- Pressure fluctuations in the pneumatic transport: < +/- 5 mbar.
- Transport lines should run horizontally and vertically (no in-/declining sections). Long curves should be avoided because they lead to segregation of the coal dust through centrifugal forces and this in turn leads to plugging. Diversion pots have proved the best solution in three respects: 1. low wear and tear 2. low loss of pressure and 3. the coal dust is remixed with the transport air at every turn (Fig. 6).
- Maximum number of turns: 5 (preferably by diversion pots); first turn after the dosing no diversion pot
- Total length of the pneumatic transport line: < 120 m

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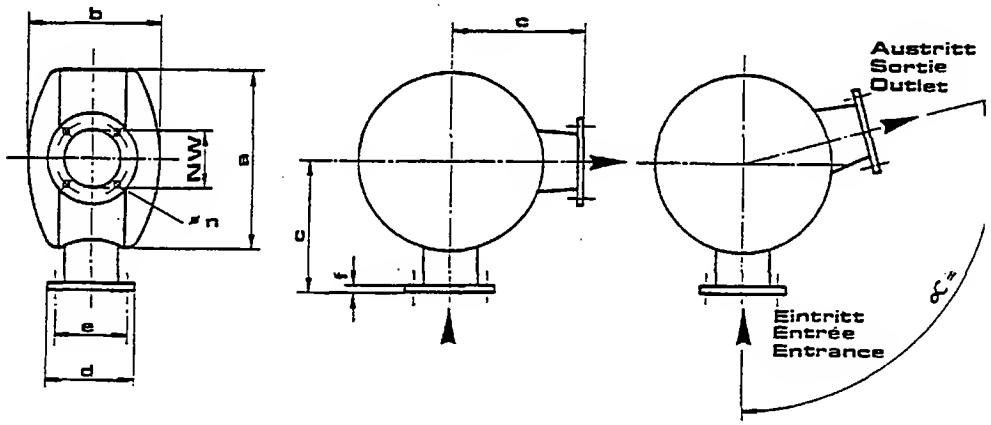
Figure 6: Diverting Pots for Pulverized Coal Transport (Units: mm)



NW	a	b	c	d	e	f	n	ca.kg
80	400	280	280	190	150	16	18 4	20
100	400	330	325	210	170	16	18 4	24
125	400	400	330	240	200	18	18 8	26
150	800	450	440	265	225	20	18 8	50
175	600	450	440	295	255	22	18 8	54
200	700	550	490	320	280	22	18 8	76
250	700	830	450	375	335	24	18 12	80
300	800	690	680	440	395	24	22 12	145
350	800	750	495	490	445	26	22 12	160

3. OIL FIRING SYSTEMS

The handling of fuel oil in a cement plant can be subdivided into the following steps:

1) Transfer to the storage tanks
2) Storage and extraction from storage tanks
3) Preparation, measuring, dosing
4) Atomization and combustion

The last point will be delt with in the separate paper: "Burners and Flames".

3.1 Fuel Oil Transfer from Delivery Point to the Storage Tank

For easy handling, fuel oil must have a temperature of about 50 to 60°C. If it is delivered at lower temperatures, which - due to the insulation of the wagons - is rather seldom, it has to be heated up. This can be done by circulating saturated steam (8 to 12 atm), thermal oil or electricity through the heating coils at the bottom of the railway wagons or trucks. Heating time depends on the boiler output, on the capacity of the wagon, on delivery temperature of the oil and on ambient temperature and lies between 2 and 6 (12, 24) hours (200 to 250 kg/h of steam is needed for a 20 tons capacity wagon). It is therefore common practice to do this - whenever required - in the afternoon, to heat up the oil during the night and to empty the wagons in the following morning. Via coarse strainers (for pump protection) the fuel oil is then pumped to the storage tanks (Fig. 7).

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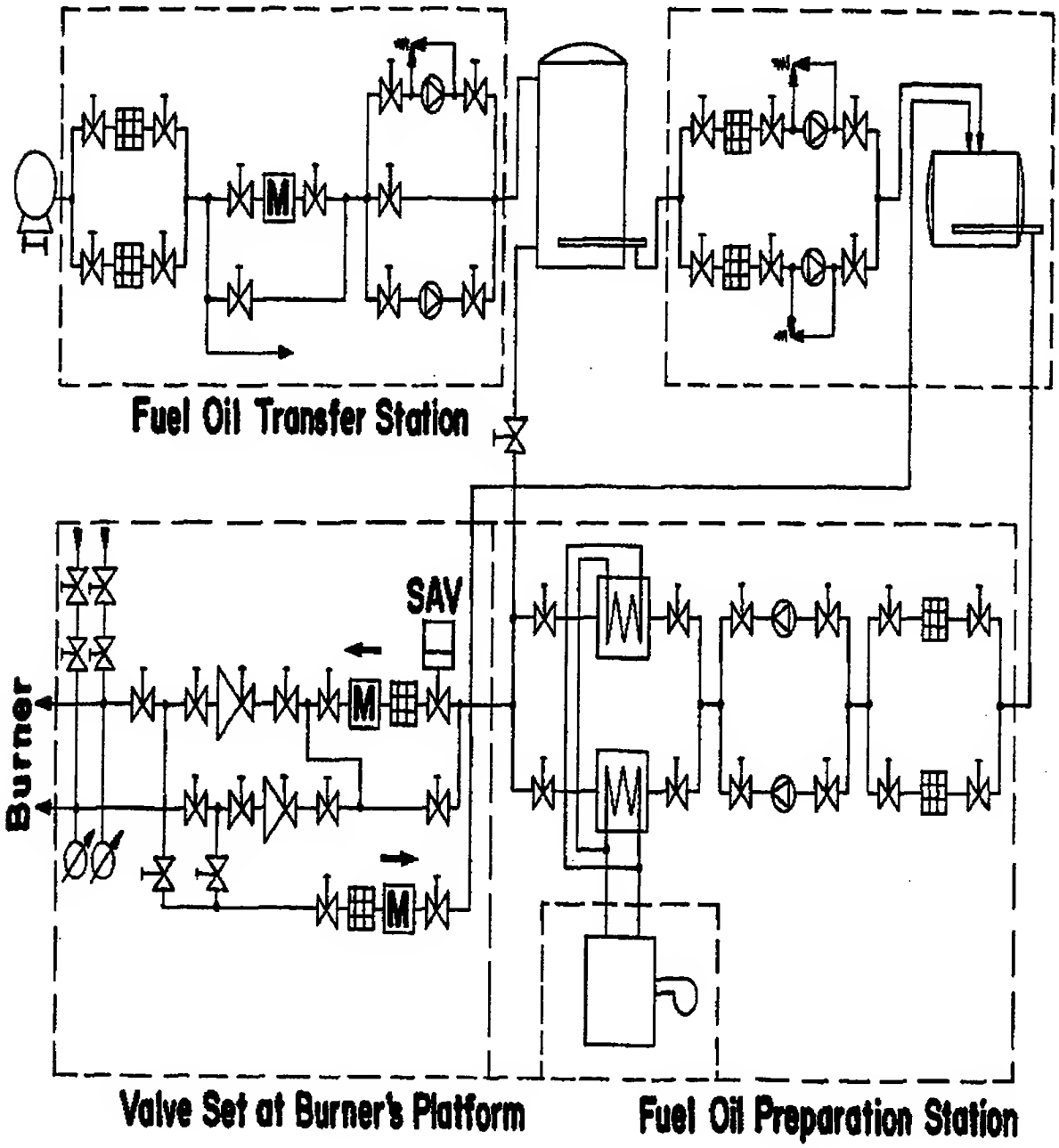
Figure 7: Fuel Oil Handling



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3.2 Fuel Oil Storage

The main storage requirements depend on the situation of the plant with respect of the fuel oil supply possibilities. A few plants are located sufficiently close to a refinery so that the oil is received by pipeline, directly from the refinery. Such cases require a minimum storage capacity.
Where oil is delivered by truck or by rail, typical main storage capacities allow a kiln operation of 2 to 10 weeks. Tanks are usually designed as welded steel constructions. Due to the fuel oil forming an insulating layer on the walls, any particular insulation efforts are unnecessary.
Suction heaters are used to maintain the fuel oil locally - i.e. in the area of the tank suction point - in a pumpable condition, i.e. at temperatures between 50 and 60°C. This is done in order to minimize the rate of deposit forming reactions, which doubles with each 10°C increase in fuel oil temperature.

3.3 Fuel Oil Preparation

Successful burning of oil requires that it is heated to approx. 140 - 170°C (see Chapter 3.4) in order to reduce its viscosity enough to allow it to be properly atomized by pressure atomization.

Heating up of the fuel oil is usually accomplished through an assembly of equipment all contained on a common base. This minimizes expensive piping and valving and centralizes the equipment for ease of maintenance and control.

Due to the foreign matter that all residual oils contain and the high rate of deposits that form at elevated temperatures, resulting in frequent maintenance, all equipment associated with and on the final heat and pump set is duplicated.

Such a set would contain (see Fig. 7):

- 2 strainers with coarse meshes for pump protection
- 2 oil pumps (gear pumps or screw pumps)
- 2 heat exchangers for heating up the fuel oil to atomization temperature
- 2 strainers with fine meshes for control equipment and atomizer head protection

The supply of heat mainly to the heat exchangers of the fuel oil preparation set, but also to the storage tank suction heater as well as to all oil carrying piping can be accomplished by:

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3.3.1 Heating with Steam

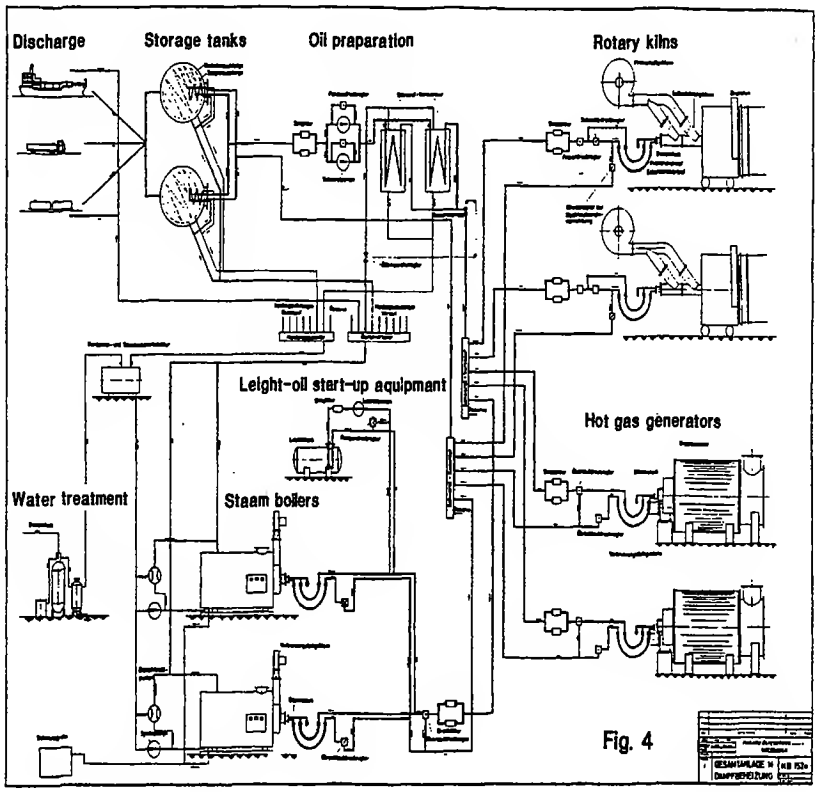
Steam has certainly been the most popular heat carrying medium for oil heating in the past (see Fig. 8). The principal problems associated with steam generation and its use are:

- feed water treatment
- steam trapping
- condensate handling
- high pressure operation
- freezing problems during plant stop

Steam can be produced by:

- conventional oil fired steam generators
- electrical submersion heaters in a pressure vessel
- waste heat based steam generators (e.g. cooler exhaust air)

Figure 8: Fuel Oil Preparation System Based on Steam



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3.3.2 Heating with Thermal Oil:

The essential advantages of these inorganic, low flammability oils as a heat transfer medium are:

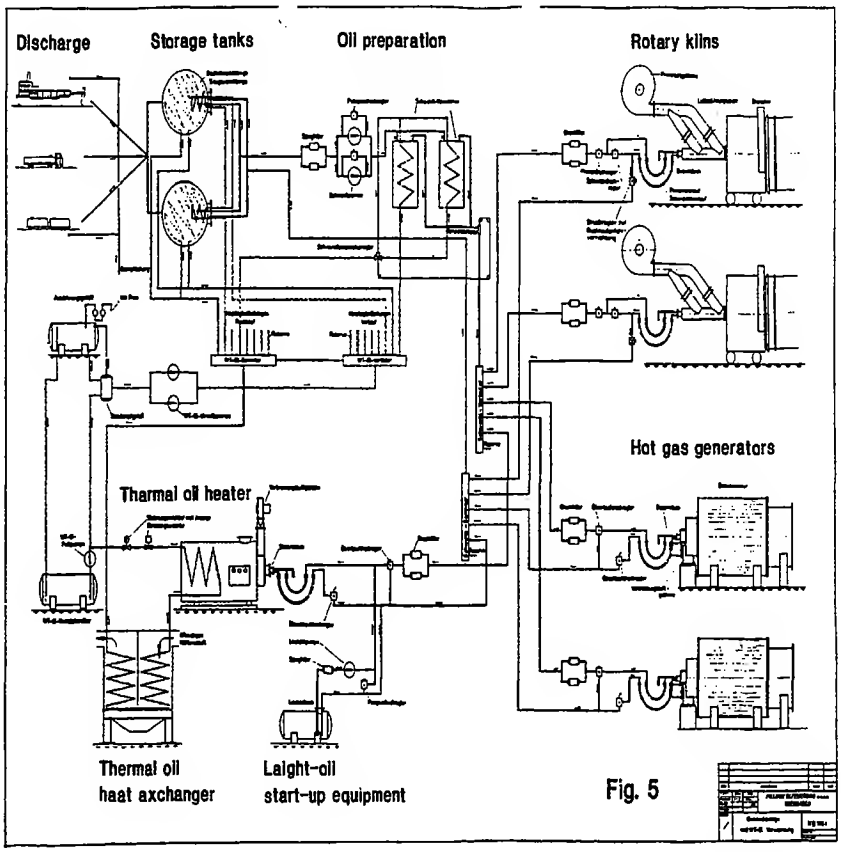
- operation in a constantly liquid phase
- low pressures even at operating temperatures of 250 to 300°C
- no freezing problems

They might be treated up by:

- oil fired thermal oil heaters
- electrical submersion heaters
- waste heat based thermal oil heater (e.g. cooler exhaust air)

Thermal oils are subjected to aging. Their quality has therefore to be checked in regular intervals of about one year. About every five years replacement by a new charge is required (see Fig. 9).

Figure 9: Fuel Oil Preparation System Based on Thermal Oil



3.3.3 Heating with Electricity

Due to high operating costs, direct electrical heating of fuel oils is used for low capacities only. However, it is sometimes used as auxiliary heating for large systems to permit starting when the system is cold.

Electrical power is also used in heating oil lines through "resistance heating". The oil line itself is used as the conductor for high current, low voltage power.

3.3.4 Heating with Flame Radiation

The heating medium in this case is the flame itself. The thermal oil heater is an example of the direct fired heater. Replace the thermal oil with fuel oil and this, then, is the direct fired fuel oil heater.

Since fuel oil cannot be heated to the same high temperature as the thermal oils, burner flame modulation (shape and length) within the heating chamber must be closely controlled to maintain a narrow oil temperature range, e.g. (120°C ± 2+C) over a wide range of oil flow. This close burner flame control must be maintained to prevent overheating and carbonization of the residual oil.

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3.4 Quality of Fuel Oil Preparation

For heavy oil combustion, the kinematic viscosity at the burner nozzle must lie within the range of 12 to 20 cSt - preferably 12 - 15 cSt (upper limit 20 cSt) - this ensures that the droplet size needed for good combustion can be achieved. In today's heavy oil market, particularly in the South American OPEC countries, heavy oil is offered which has a significantly higher viscosity than the limit specified by DIN 51 603. It is therefore essential to keep track of the relationship viscosity - temperature and adjust the oil temperature as necessary.

Fig. 10 shows the kinematic viscosity of different fuel oil types in function of temperature. The upper limits for atomization and pumping are indicated.

Fig. 11 shows a conversion table for the different viscosity units.

Furthermore it is important to keep the oil temperature constant within a very narrow range to have a stable flame.

Figure 10: Kinematic Viscosity of Current Fuel Oils

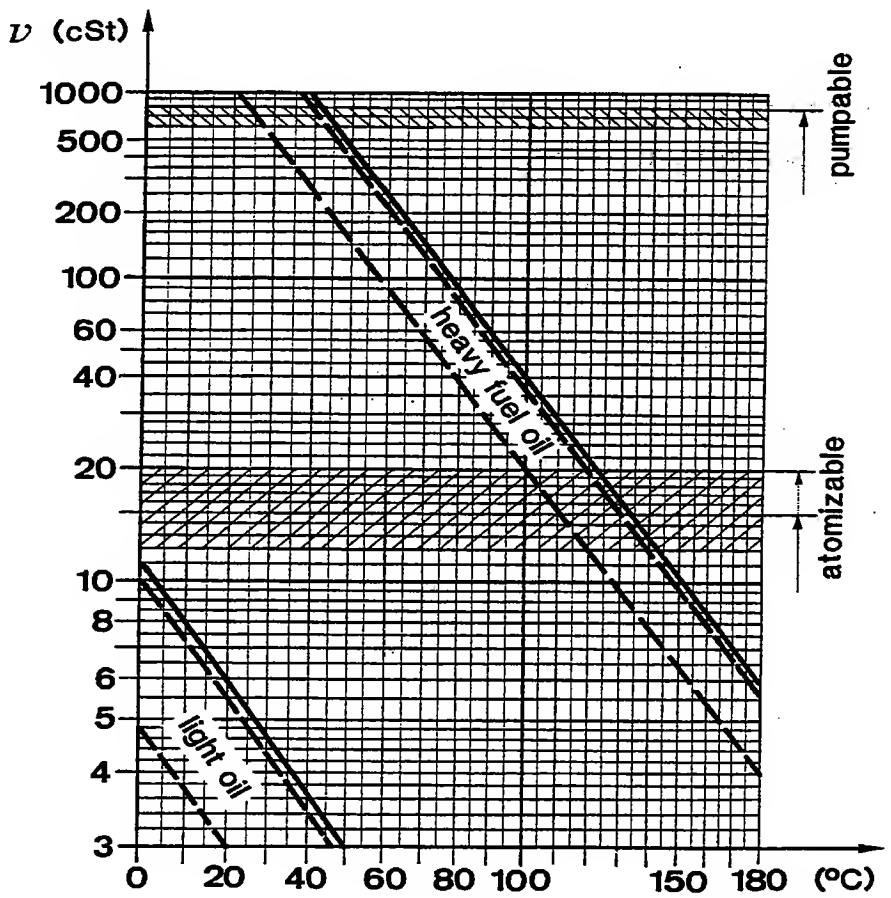
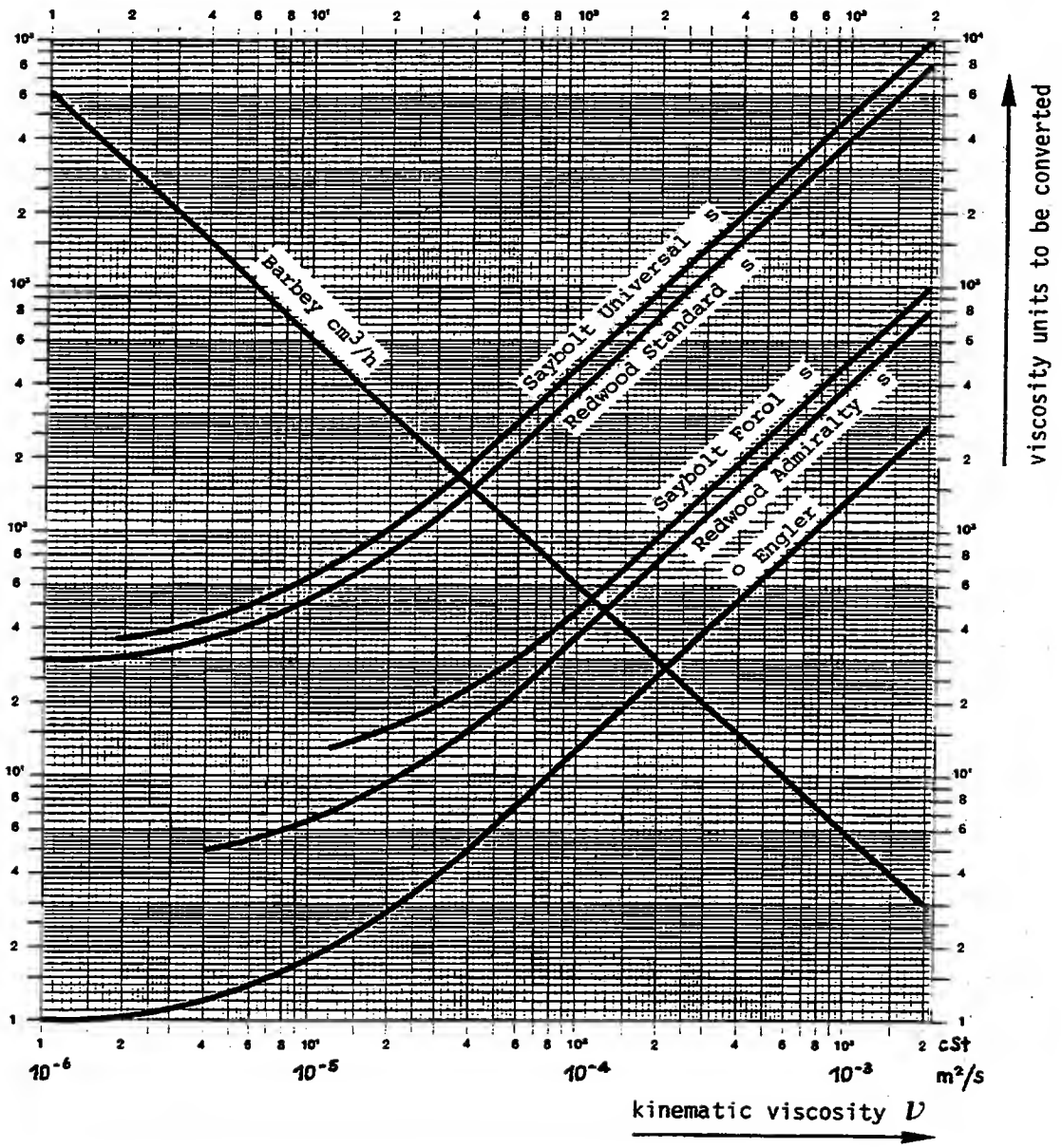


Figure 11: Conversion of Different Viscosity Scales



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3.5 Control Loops in the Fuel Oil Circuit

Between storage tanks and fuel oil burners, there are generally four control loops installed, which have to keep constant the following values:

1) Fuel oil temperature at the storage tank suction point.
2) Pressure in the oil circuit line between storage tanks and preparation station (Bypass of a part of the flow back to the storage tank; see Fig 7).
3) Temperature of the fuel oil to be atomized (Preparation Station).
4) Atomizing pressure: Accomplished by means of a bypass valve which leads part of the flow back to the storage tank (see Fig. 7) or by means of variable speed high pressure pumps, which are directly controlled by the oil flow meter.

For burner nozzles with separate feed for axial and radial oil (Pillard, Unitherm), the oil pressure difference for optimum atomizing is set to 1,0 – 1,5 bar. However, as the accuracy of the reading on the oil manometer at the operating pressure of about 40 bar is unsatisfactory, it is recommended that both channels are equipped with flow meters. The pressure (flow characteristics given by the nozzle suppliers) can be taken into account in optimizing atomization.

Furthermore, whenever a kiln stop occurs, the oil lance and the atomizer head have to be cleaned automatically by steam or compressed air in order to avoid overheating and coking of the oil. Continuation of burner cooling has to be assured by having the primary air fan connected to the auxiliary power generators. In cases of prolonged kiln stops removal of the oil lance is preferable, thus, also providing the opportunity to check the condition of the atomizer plate, which is very important for complete combustion.

4. NATURAL GAS FIRING SYSTEMS

4.1 Natural Gas Preparation

Gas distribution by means of pipelines is accomplished at pressures of 30 to 80 bars. At consumer's site the gas pressure is reduced to the required operational pressure, mostly by means of a two stage expansion process. The first stage takes place in the NG transfer station while the second runs off in the NG pressure reduction station.

As a standard solution the NG transfer station is an independent, self-sustaining installation contained in a separate building (noise suppression). Similarly to the fuel oil preparation plant, all equipment is duplicated and provided with a number of bypass possibilities. The main equipment list is as follows (Fig. 12):

- Remote controlled main shut-off safety valve
- Transfer station inlet filters for protection of equipment from solid particles originating from the pipeline
- Thermal oil heated exchangers aiming to preheat the natural gas to such an extent that the following temperature drop due to expansion will not cause valve internal and external ice formation (Joule - Thompson effect: 0.3 to 0.5°C/bar)
- Safety shut off valves
- Pressure reduction valves (for reduction of the gas pressure to the pressure level of the plant internal distribution network of 3 to 10 bar)

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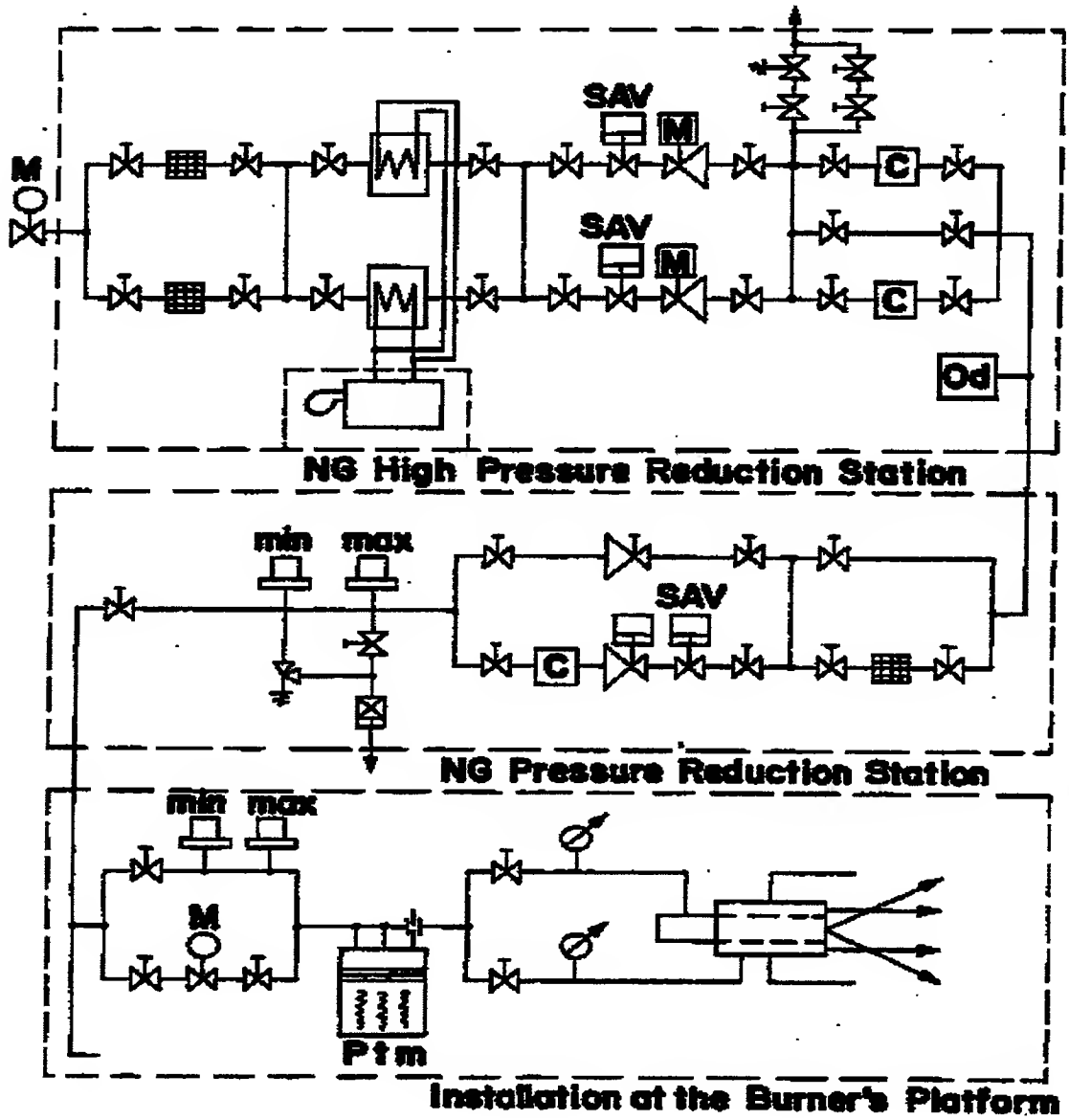
Figure 12: Handling and Preparation of Natural Gas in the Cement Plant



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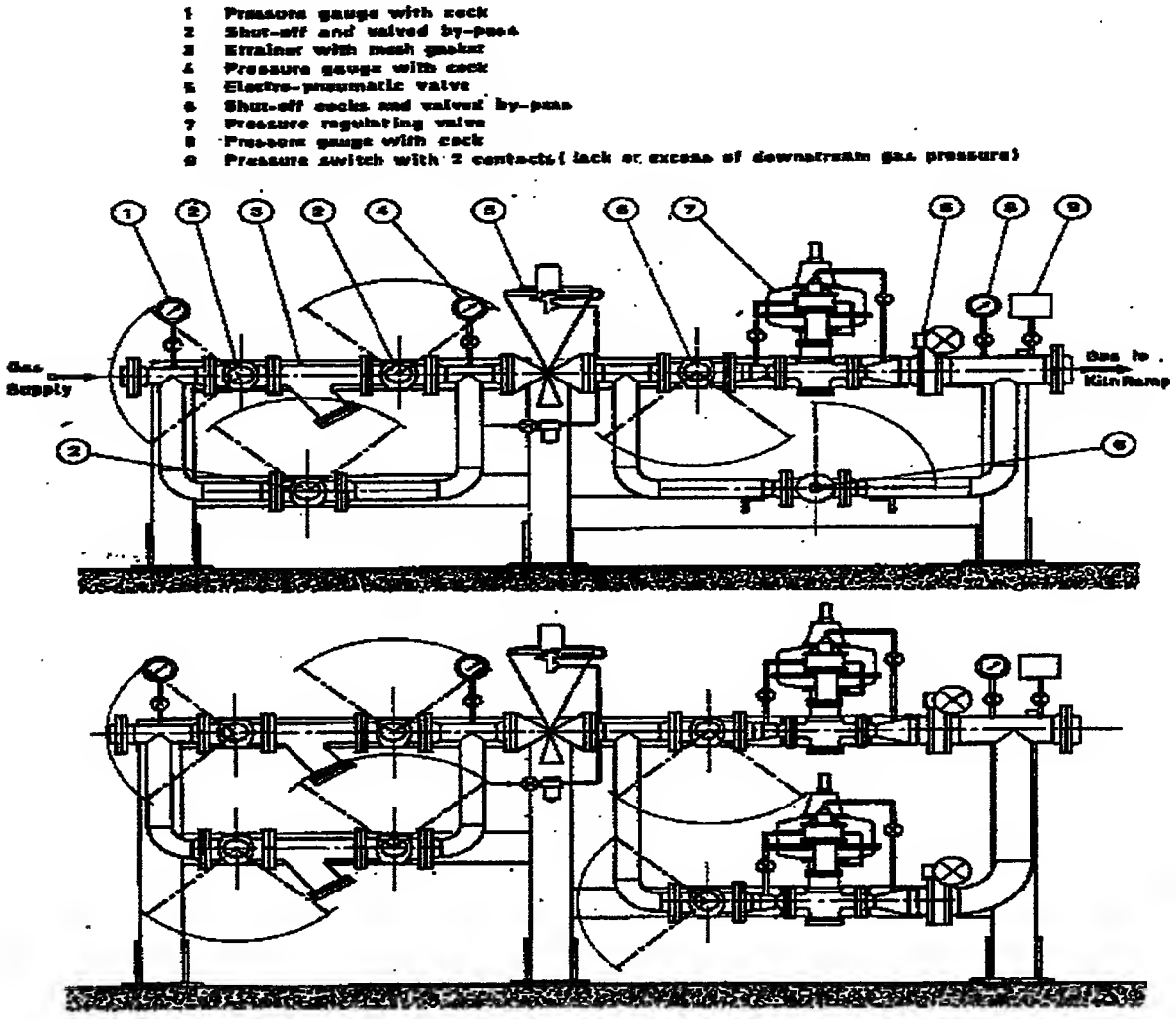
The heat value of the natural gas can be measured and recorded continuously by means of on-line calorimeters. Though this is often not done - plant people tend to rely on the heat values given by the gas suppliers - it would be worthwhile, since in some cases the heat values might vary in range of ±300 kJ/Nm^3 from day to day.

To enable leaks from the gas pipes to be detected surely and quickly, a powerful odorizer (e.g. mercaptan) is added to the gas just after the gas leaves the transfer station.

The second stage of pressure reduction, taking place in the pressure reduction station, is located near the point of consumption (Fig. 13). With the exception of the NG heaters it contains about the same equipment as the transfer station. The aim of this installation is to completely even out supply network pressure fluctuations and to set the final pressure according to the requirements of the consumer i.e. the burner and kiln systems.

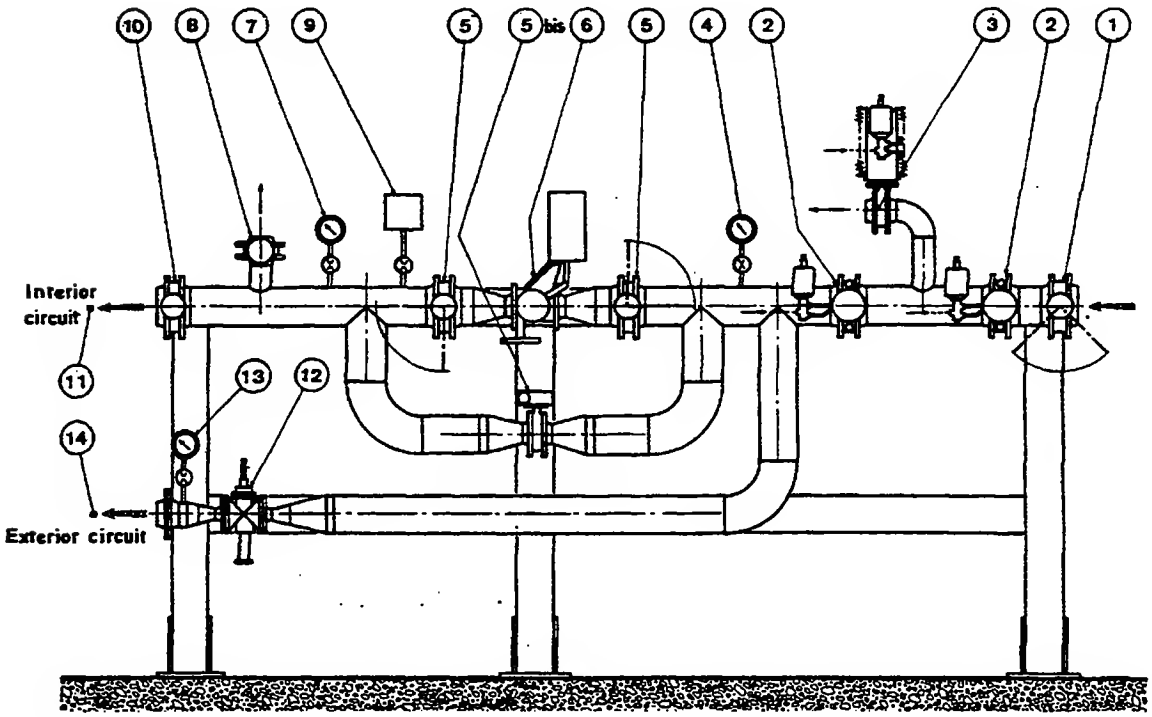
Figure 13: Secondary Pressure Reducing Unit



Immediately before the kiln, the gas stream is split up in order to supply the radial and the axial gas nozzle of the burner (Fig. 14).

Figure: 14: Kiln Ramp Unit

1 Manual shut-off cock
2 Electro-pneumatic shut-off valve
3 Electro-pneumatic vent valve
4 Pressure gauge with cock
5 Regulating valve shut-off cocks
5bis Regulating valve by-pass cock
6 Interior circuit regulating valve
7 Pressure gauge with cock
8 Vent cock
9 Pressure switch with 2 contacts
10 Interior circuit shut-off cock
11 Interior circuit flexible hose
12 Exterior circuit, manual regulating valve
13 Pressure gauge with cock
14 Exterior circuit, flexible hose



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4.2 Safety Precautions

4.2.1 Flexible Hoses Bursting

Since there is some risk of the flexible gas hoses between kiln burner and gas supply line bursting or of the proceeding valves etc. failing, pressure monitors for the maximum and minimum pressure are inserted immediately before the hoses concerned. In the event of an emergency stop, a safety stop valve, or two in series, are actuated to stop any further input of fuel at once.

4.2.2 Leak Tests

To check the gas pipes and fittings inside the plant for leaks the following methods are used:

- Normally leaks can be detected naturally as a result of adding odorizer.
- When machines are switched off, the hissing sound of the escaping gas is easily discernible.
- A somewhat riskier method is to run a naked flame along the gas pipe. This results in a flaming torch being produced at the leak, which cannot be overlooked. There is no risk of this flame striking back into the supply pipe (quenching distance, lack of oxygen), but escaped gas could cause an explosion.

When constructing buildings which contain gas pipes, it is essential to allow for sufficient ventilation. This point does not usually give rise to any difficulty in cement works. But to be quite sure, certain items of equipment can be fitted with guard flames from the start. Their task is to ignite any gas that escapes before a large quantity of explosive mixture has a chance to collect.

A further possibility is to install gas detectors in critical places such as the gas preparation station or the burner tunnel.

4.2.3 Explosions in the Kiln

The most important requirement is that the fuel should not be allowed to enter the kiln unintentionally or at an uncontrolled rate, as this is essential to prevent explosions occurring in the kiln itself or in the systems following it (e.g. preheater tower, EP).

This means that the fuel input has to be stopped immediately in the event of the flame going out. In this respect it must be said that extinction of the flame in a hot kiln has never been observed so far, even during material rushes.

Nevertheless during the start up of the cold kiln, lifting off and extinction of the flame can occur, for example caused by partly blocked burneroutlets which lead to increased injection speed of the gas. If the gas is injected with a too high speed, the flame can be blown out.

Therefore careful observation of the flame during the whole start up period is of utmost importance. In the case of the flame going out, fuel supply has to be cut off immediately to prevent explosions.

Excessive fuel input can also cause explosions because of CO accumulations. Therefore careful monitoring of CO concentrations is important.

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Safe Handling of Coal and other Combustible dusts

F. Schneider
(Original title: Basic safety theory of solid fuel preparation)

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SUMMARY

Dust explosions can only occur when - besides certain marginal conditions - the following factors simultaneously are present:

- Stirred-up, combustible dust
- Oxygen
- Source of ignition

A basic distinction is made between active explosion protection techniques (prevention of the occurrence of explosions) and design related explosion protection (reduction of the effects of explosions).

In practice, the following measures are applied:

1) **Preventive Measures:**
   Their aim is:
   * to exclude possible ignition sources within the installation
   * to prevent the building up of coal dust deposits, wherever possible
   * to detect the source of a fire as soon as possible
   * to extinguish the fire with a minimum of danger

2) **Explosion Protection Measures:**
   * Either active protective measures involving inert gas operation. This is the case when the oxygen concentration within the pulverizing plant is kept below the critical limit - for solid fuels dust, as a rule, less than 12 to 14% - as long as combustible dust is present in the system.
   * Or, alternatively, design related protective measures based on the use of explosion resistant construction in accordance with VDI Guidelines No. 3673.

1. INTRODUCTION

The operational safety of solid fuel plants is an important decision factor when the choice of the preparation system is being considered. For this reason, the three most important basic systems will be briefly reviewed here, differentiated according to their different methods of handling gas and coal dust (Fig. 1).

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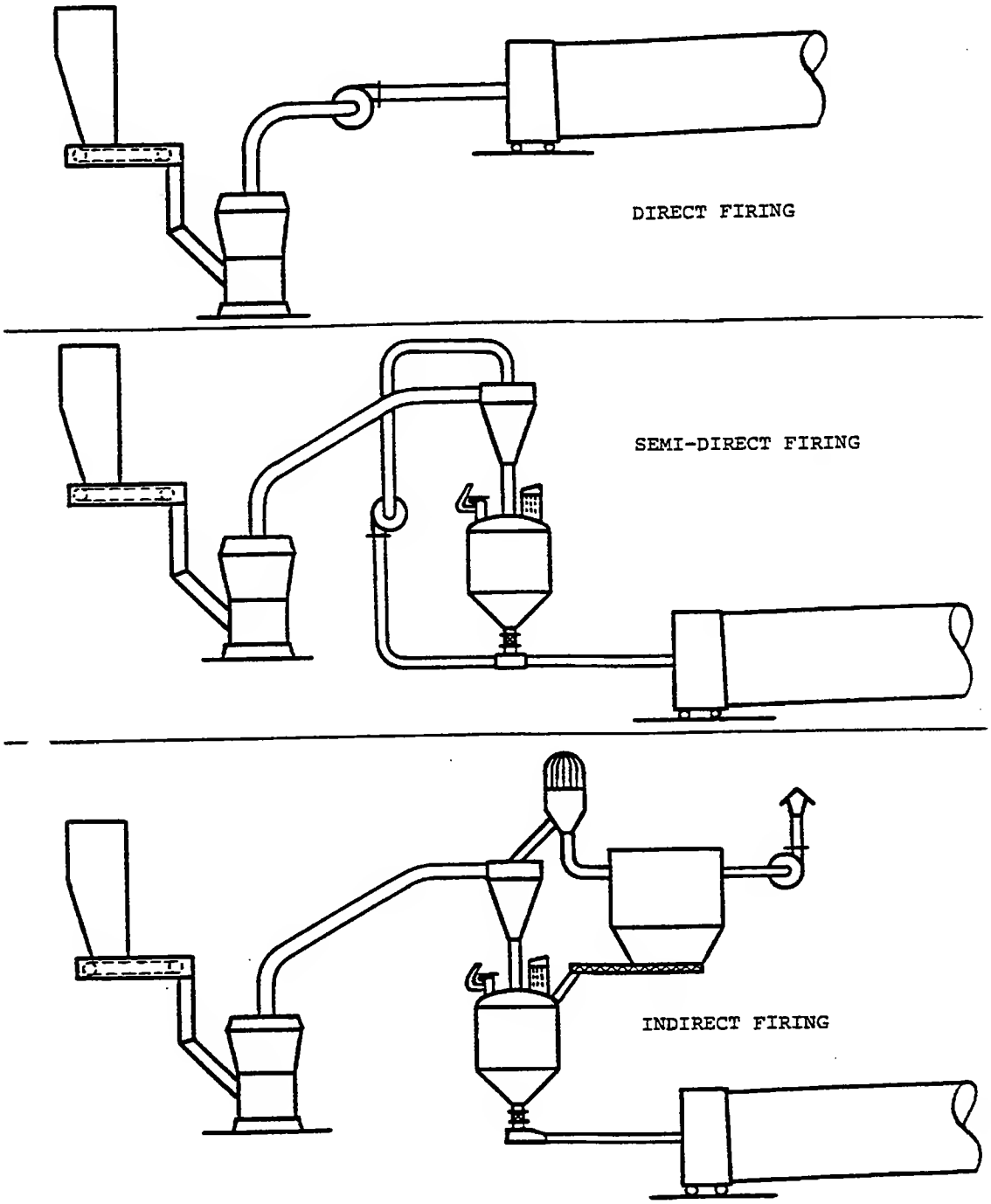
Fig. 1 Firing Systems



a) Direct firing
In this system, the combustible dust is conveyed into the kiln together with the exhaust gases resulting from the drying-cum-grinding operation. This arrangement represents the simplest design and is easily controllable from the safety point of view. However, there are also considerable disadvantages involved with the use of this system in clinker manufacture with increasing ballast content of the fuel.

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b) Semi-direct firing
In this system the combustible dust is separated in an intermediate silo, while the mill exhausts, possibly as recirculated air, are conveyed to the cement kiln as the primary air supply. This results in the technical disadvantages of direct firing being reduced to a certain extent - at the expense of a somewhat more complex installation - but all drying gases are still conveyed to the kiln.

c) Indirect firing
This solution is surely the best possible version when the operation of a rotary kiln is being considered. The pulverized fuel can be conveyed to the firing system from the silo independently from the pulverizing plant operation. The firing system can be operated with a primary air ratio designed for optimal flame generation, as the mill exhaust gases are filtered. Against this we have increased risks with respect to safety due to the operation of the filters and silos and higher control technique requirements.

Further discussion of the decision criteria for the selection of an optimal preparation system is outside the scope of this lecture. However, it is certain that when factors such as

- the growing size of installation
- installations with several firing systems
- the use of fuels rich in ballast
- the use of fuels of widely differing quality characteristics

are considered, the decision will be influenced in favor of the indirect firing system which needs far more advanced and sophisticated safety techniques than the simpler direct firing system does.

For this reason the damage prevention possibilities discussed below refer basically to the indirect firing system and must be adjusted accordingly if they are applied to other systems.

2. BASIC PRINCIPLES FOR HANDLING PULVERIZED FUELS

2.1 Development of Dust Explosions and Fires

In order to effectively ensure the safety of a solid fuel preparation plant, we must first be aware of the sequence of the possible fuel reactions.

Dust explosions can only occur if the following three conditions are simultaneously fulfilled (Fig. 2).

a) Stirred-up, combustible dust present in explosive concentration.

b) Air or oxygen above the critical concentration, for coal dust as a rule, above 14%, for lignite above 12%.

c) An ignition source possessing energy above the minimal ignition energy (depending upon the type of dust).

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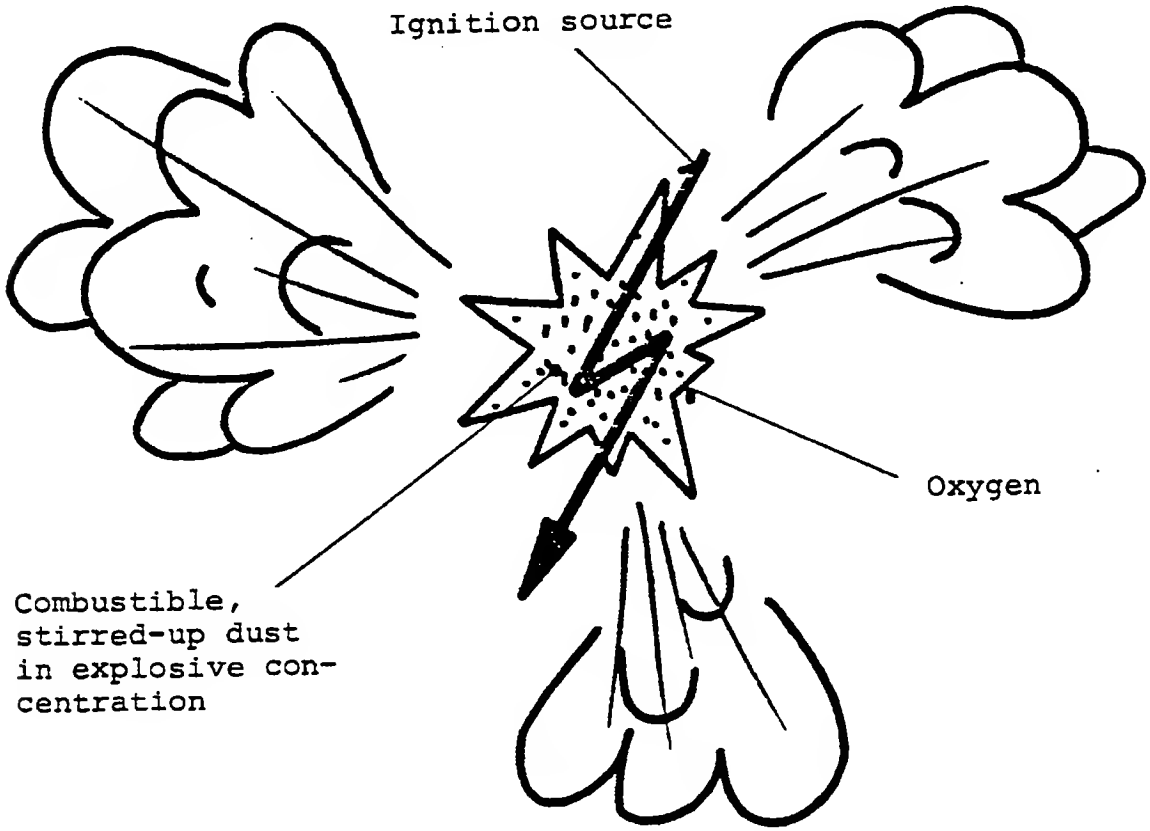
Fig. 2 Preliminary Conditions for an Explosion



After the ignition of an optimally explosive mixture in an enclosed space, the pressure increases more or less rapidly until it reaches the maximal explosion pressure P_{max}, and then decreases more or less slowly to the original pressure, depending on the aerodynamic conditions (Fig. 3). While the maximum explosion pressure is almost independent of the container's form and size, and in case of coal and lignite dusts, amounts to approximately seven to nine times the initial pressure the maximum rate of pressure rise $\left(\frac{dp}{dt}\right)_{max}$ - which is a measure of the explosion violence - is dependent on the container volume in accordance with the cubic law:

$$\left(\frac{dp}{dt}\right)_{max} \times V^{\frac{1}{3}} = constant K_{st}$$

K_{st} is a material coefficient that depends on the type of dust, the degree of turbulence of the dust/air mixture at the moment of ignition, the grain size distribution, and the type of ignition source. The method for determining K_{St} is given in the VDI Guidelines No. 3673.

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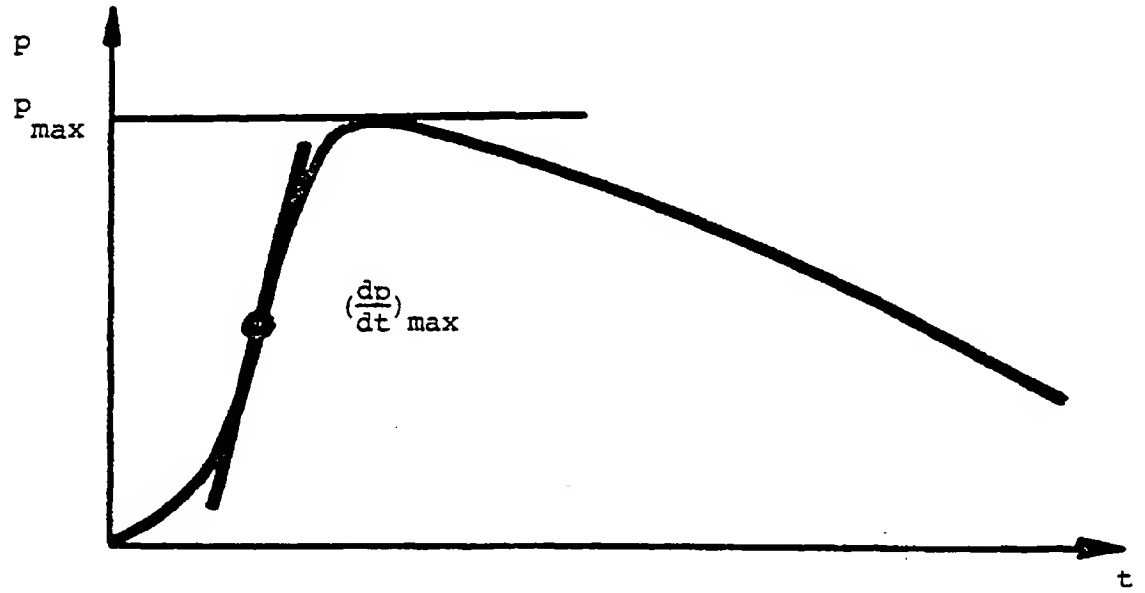
Fig. 3 Pressure Development of an Explosion Over Time



The degree of explosion violence of dusts is subdivided in industrial praxis into explosion classes, whereby the explosion class and K_{St} are related in the following manner:

Table 1

Dust Explosion Class	K_{St} (bar . m . s^{-1})
St0	0
St1	> 0 to 200
St2	> 200 to 300
St3	> 300

All types of mineral coals as well as the majority of lignites belong to explosions class St1.

Table 2 illustrates arbitrarily selected comparative values for K_{St} characterizing different types of dust.

Table 2

Dust Type	K_{St} (bar . m . s^{-1})
Hard coal	85
Lignite	150
Organic pigments	300
Aluminium	550

This comparison shows, that hard coal dust develops a less violent explosion than aluminium dust.

It must be noted, that the value „K_{St}" does not allow any conclusion as regards the risk involved with that particular dust. The main significance of K_{St} is for the dimensioning of design related protective measures.

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Smoldering fires, characterized by slowly smoldering combustion, can occur wherever combustible dust is stored for a longer period of time, whereby the ignition sources can be spontaneous combustion, initiated by external heat sources, mechanical sparks, or electrical sparks and arcs. Combustion propagation in smoldering fires is quite possible in very low oxygen concentrations.

2.2 Possible Protective Measures against Dust Explosions and Fires

In dust explosion protection techniques a distinction is made between active protective measures (prevention of the occurrence of explosions) and design related explosion protection (explosion resistant construction).

2.2.1 Active Explosion Protection

The active explosion protective techniques aim to exclude at least one of the three preliminary conditions necessary for an explosion, i.e.:

- Stirring-up of combustible dust
- Oxygen content above the critical concentration of generally 12% for lignite or 14% for hard coal
- Ignition source

2.2.1.1 Ignition Source

In a pulverizing plant, ignition sources cannot be excluded with absolute certainty. It is always possible that mechanical sparks will be generated by the action of foreign bodies or by friction between moving machine part or that the hot gas or coal feeding system will supply smoldering fuel particles.

2.2.1.2 Combustible Dust

It is of course impossible to replace the combustible dust with a non-combustible material in the preparation of fuel.

Thus, the only remaining possibility is the exclusion of air or oxygen respectively, or the reduction of the oxygen content in the fuel preparation plant.

2.2.1.3 Air and Oxygen

Dust explosions can be effectively prevented through inertization, i.e. the replacement of the oxygen in the air by a non-combustible gas, particularly CO_2 or N_2, if it can be ensured that the inert gas atmosphere will be maintained as long as combustible dust is present in the system.

The maximal O_2 concentration, below which no explosive propagation reactions of mineral coal dust are noted, is approx. 14%, the one for lignite approx. 12%. However, this concentration can vary in accordance with the type of fuel processed. As a safety margin of at least 2% O_2 is required, the maximal permissible limit of 0_2 concentration for mineral coal dust is therefore as a rule 12%, for lignite 10%.

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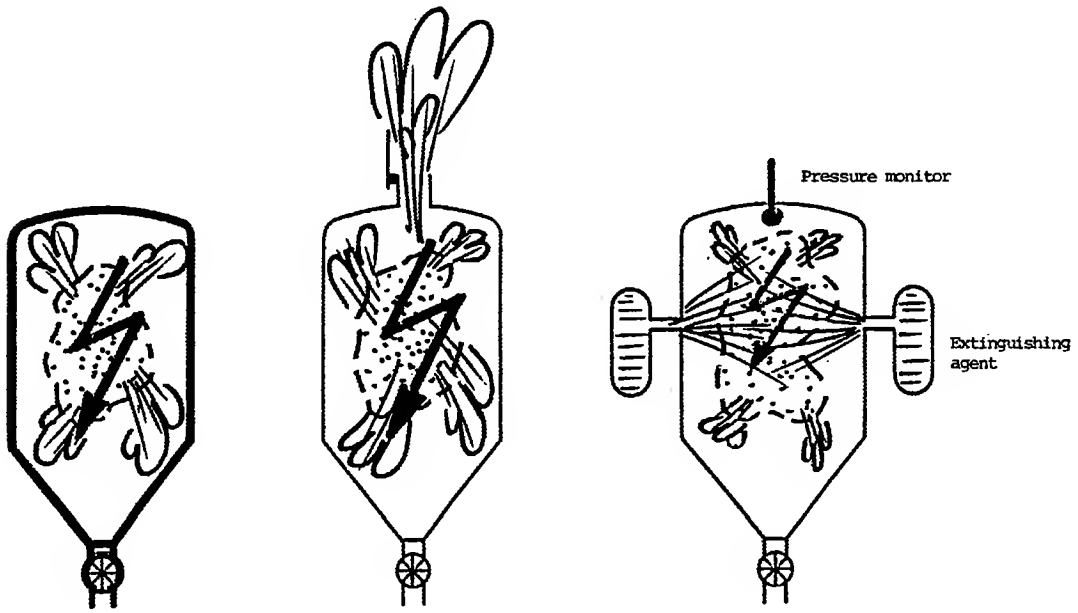
2.2.2 Design Related Explosion Protection

Reduction of the effects of already proceeding explosions, and therewith the protection of people and machines, can be achieved by:

- Explosion pressure resistant construction
- Explosion pressure venting measures
- Explosion suppression techniques (Fig. 4)

Fig. 4 Passive Protection Measures



2.2.2.1 Explosion Pressure Resistant Construction

Explosion pressure resistant construction restrict any possible explosion to the dust conveying installation, whereby a certain amount of minor damage to the installation commensurate with the complexity of the facility is accepted.

All dust conveying installation parts as well as the adjacent equipment and sealing elements must be designed to resist the maximal explosion pressure of 9 bar expected in the case of coal or lignite dust. If deformation of the container is accepted, the maximum permissible explosion pressure may be up to 50% above its design value (pressure shock resistant design). A design for 6 bar static overpressure is required for an expected maximum explosion pressure of 9 bar. Such construction methods are of course quite complex and expensive. However, in the event of an accident the installation is again operational within a short time.

2.2.2.2 Explosion Pressure Venting Measures

In a broader sense explosion venting means all measures that serve to open temporarily or permanently the previously closed installation in a safe direction, at the beginning or after a certain spreading of an explosion. The purpose of this is to prevent any overstressing of the mechanical equipment beyond its pressure shock resistance. The strength of the equipment does not have to be designed for P_{max}, but only for the reduced explosion pressure P_{red} (Fig. 5). A deformation of the container may again be acceptable, but it must not burst.

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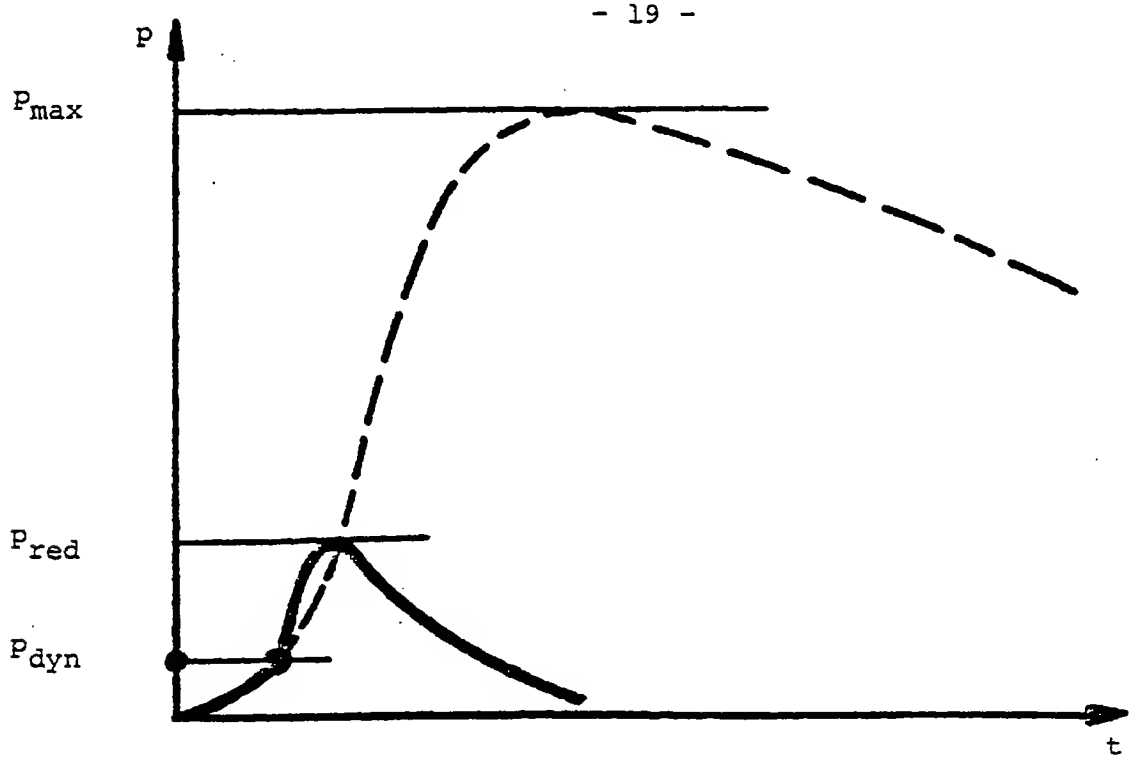
Fig. 5 Pressure Response in Explosion-Pressure-Relief Techniques



The explosion pressure venting technique operates in the following manner: When the dynamic response pressure of the pressure venting installation is reached, predetermined breaking points, rip foils or doors open to vent the shock wave outdoors, mainly by means of amply dimensioned discharge channels. Immediately after the pressure venting system responds an increase in the temporal rate of pressure rise can often be observed which is due to the higher turbulence caused during the venting of the shock wave. The pressure rise then quickly stops at P_{red}. Guidelines concerning the design layout and dimensioning of the explosion pressure venting installations are contained in VDI Guidelines No. 3673. If the method of explosion venting is applied not only the inserts of the containers such as filter cloths etc. must be considered but the expected recoil forces as well. With a pressure venting area of 1 m^2, a reduced explosion pressure of 2 bar, and under the assumption that the shock wave escapes with the velocity of sound, a thrust of approx. 15 t acts upon the housing to be protected. This must be properly supported or else the container may be torn from its foundations.

2.2.2.3 Explosion Suppression

In the explosion suppression techniques, the shock wave preceding the combustion front or the infrared radiation of the combustion area is detected by a device which quickly distributes extinguishing agents under a propellant pressure of 60 to 120 bar by means of detonator-operated valves. With a programmed dynamic response pressure threshold (P_{dyn}) of the detectors, the maximal explosion pressure is again lowered to a reduced level (P_{red}).

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2.2.2.4 Limitation: Explosions from Ducts into Containers

The described constructive protection techniques are effective under the condition that the reaction takes place as described in the paragraph 2.1. The description is applicable to most explosions that occur in pulverizing plants. However, if an explosion strikes from a duct into a container, and in doing so the residual dust deposited there is stirred up with great turbulence and ignited, the reaction within the duct and the adjacent container can develop into a detonation of such dimensions that the resulting pressures can amount to 50 times the original pressure, accompanied by a combustion front traveling at supersonic speed, so that any relief or suppression installation is too sluggish in action. However, such events are, fortunately, relatively rare in coal operations. As a limit for a spontaneous explosion propagation, an explosion characteristic of 100 bar.m.s^{-1} was observed under particular conditions in a 200 m long pipeline of 1800 mm diameter located at the experimental mining research station in Dortmund, while the usual values for coal are generally lower (approx. 85 bar.m.s^{-1}).

However, if the principles of design related explosion protection are to be consequently pursued, every duct conveying combustible dust in an explosive concentration and whose length exceeds five times its diameter must be safeguarded by an explosion vent placed ahead of its inlet into a container (such as a filter) (Fig. 6). Through this any explosion originating in the pipeline will be vented so that the protective measures taken with respect to the adjacent container can be designed in accordance with the criteria of an explosion starting in the container itself.

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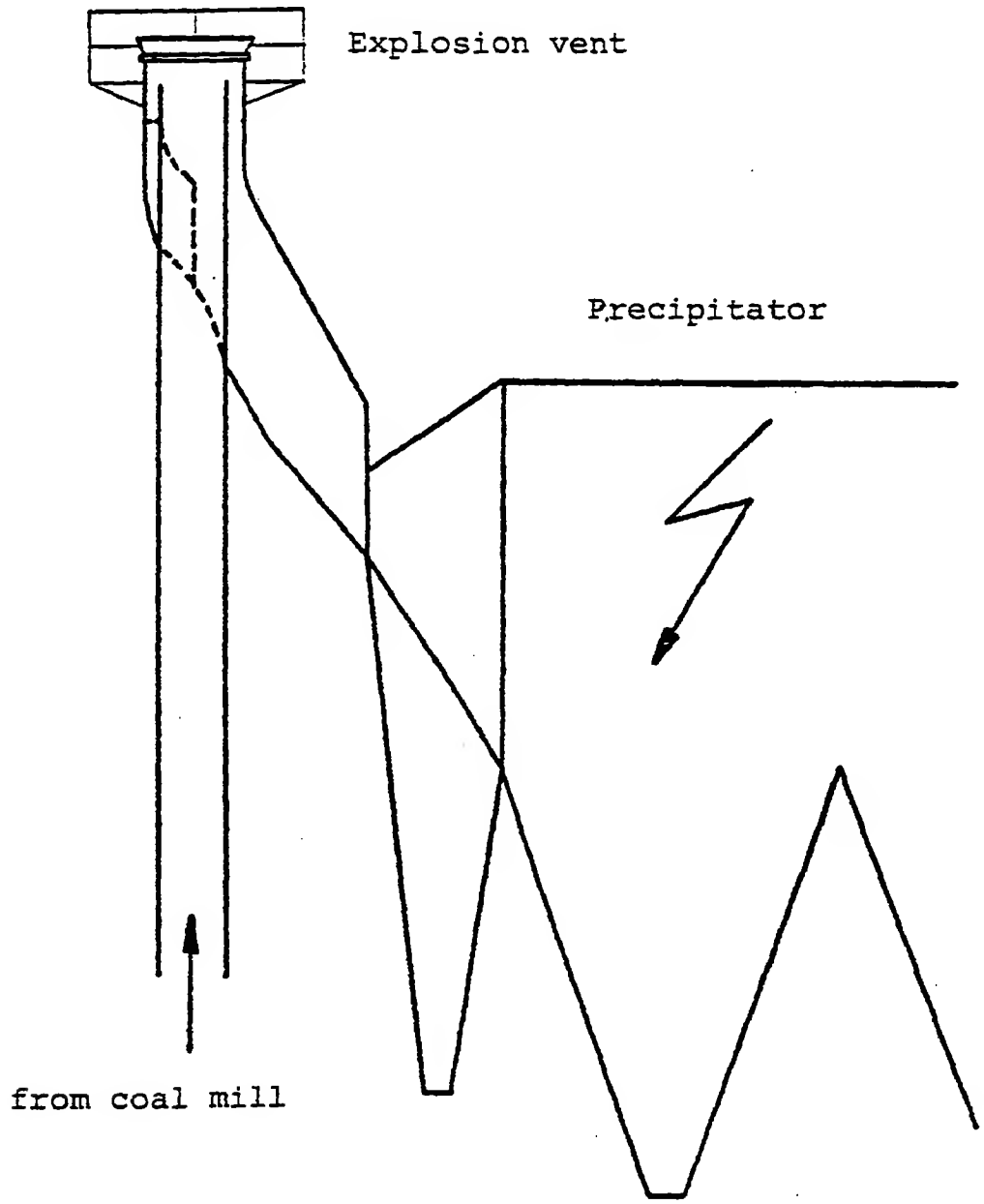
Fig. 6 Venting of a Duct in Front of a Precipitator



2.2.3 Prevention of Smoldering Fires

Smoldering fires in dust deposits are best prevented by preventing the possibility of greater quantities of dust accumulating. This is achieved through the appropriate design and slope of surfaces, pipelines and supports, as well as sufficiently high gas speeds within the conveyor systems.

In silos where great quantities of combustible dust are stored for the plant's own specific purposes, any combustion that may occur must be detected as early as possible by carefully monitoring of the dust temperature and the CO content of the silo atmosphere so that proper countermeasures can be taken.

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3. APPLICATION OF PROTECTIVE MEASURES IN THE INDUSTRIAL ENVIRONMENT

The fire and explosion protection measures described above result for practical applications on the one hand in a network of preventive safety measures that significantly reduce the risk of an accident in the operation of combustible dust installation, and on the other hand in actual explosion protection techniques that can prevent explosions, or at least shall hold the explosions within acceptable limits.

3.1 Preventive Safety Measures

The primary aim of preventive safety measures is to exclude possible ignition sources as causes of conflagration or explosion if at all possible. In addition, they are also intended to prevent secondary damage caused by the expulsion or stirring up of vast quantities of dust and their subsequent ignition.

These essentially preventive safety measures can be listed as follows:

- Temperature measurement of
  - mill exhausts
  - stored dust, preferably in silo entry and exit
- CO analysis of
  - the silo atmosphere in silos
  - mill exhaust after the filter
- Prevention of local overheating caused by friction in conveyor belt systems, high speeds of screw conveyors, bucket elevators, rotary valves, and bearing, and/or the detection of increasing temperatures by measuring techniques. Relative velocities of moving parts < 1 m/s are considered safe, > 10 m/s are considered as potential ignition sources.
- Spark separators in air heaters
- Metal separator prior to the mill
- Prevention of electrostatic discharges by conductive connections and grounding of all installation parts
- Prevention of arcing in electrofilters by appropriate voltage control measures
- Prevention of dust accumulation possibilities:
  - All surfaces to have a slope of at least 70° to the horizontal plane, especially in filter or silo cones
  - Regular disposal of dust deposits
  - Gas speeds in conduits of more than 22 m/s
- Protection of the stored dust from the effects of external heat, for instance by spraying the silo externally with cooling water
- Provision of inert gas supplies (e.g. CO_2) for inertization of the silos in the case of smoldering fires
- Cleanliness of operating rooms
  - Effective removal of the dust generated by means of proper dedusting installations
  - Safe elimination of dust deposits by means of suitable auxiliary material

From the point of view of safety a solid fuel pulverizing plant must be operated as continuously as possible, as critical situations often arise when the plant is not in operation. This fact must be considered when the capacity of the installation is being decided upon.

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3.2 Explosion Protection Techniques

3.2.1 Inert Gas Operation

As described under 2.2.1, active explosion protection in solid fuel pulverizing is practically limited to inert gas operation, i.e. operation with a maximum of 10 to 12% oxygen in the pulverizing plant, depending on type of fuel, as ignition sources and the stirring-up of dust can never be excluded with absolute certainty.

Active explosion protection can be applied if hot inert gases such as the kiln exhaust from cement kilns or hot gases from a combustion chamber, combined with a corresponding design for the mill's recirculation gas are available. In the last case the dew point problem becomes significant, therefore this solution is rarely applicable for very moist fuels, or special measures will have to be taken for drying of the circulation gases.

If the inert gas atmosphere can be maintained with absolute certainty through appropriate design and interlocking of the installation for as long as combustible dust is present in the system, design related protection measures become in principle redundant. In those cases where these conditions cannot be guaranteed, for example, because hot gases with higher oxygen content are being used such as clinker cooler exhausts, or because of dew point problems, design related explosion protection techniques must be rigorously applied.

3.2.2 Explosion Pressure Resistant Construction

Explosion pressure resistant construction, i.e. the dimensioning of the installation section to resist maximal explosion pressure, are mainly applicable where pressure venting methods cannot be used at all or only with difficulty, for geometrical reasons.

This is mostly the case in mills, and definitely in all conduit pipe systems where the length of the system exceeds five times the tube diameter. As a rule such components are designed to withstand a static overpressure of 10 bar.

3.2.3 Explosion Pressure Venting Measures

All combustible dust conveying components that are not in themselves designed to be explosion pressure resistant, such as cyclone, filters, pulverized fuel silos, etc. are to be provided with properly dimensioned devices for explosion pressure venting. Thereby containers and all interconnected aggregates such as bin vent filters, etc. must be dimensioned in pressure shock resistant design to withstand the reduced explosion pressure.

Explosion venting openings within a particular building must be connected to properly dimensioned exhaust channels leading into the open. In order to prevent an explosion originating in the mill spreading into the filter via the conduit pipe, the conduit pipe must be equipped with an explosion vent in front of its connection to the filter. This measure is not required for pneumatic conveying systems as in this design the dust concentration is normally above the explosion limit. In addition, the minimal ignition energy is significantly higher under the operating conditions of pneumatic conveying than it is in the case of stirring-up combustible dust in containers.

The area containing the vent opening for explosion pressure venting must not be accessible to anyone when the installation in operation.

VDI Guidelines No. 3673 can serve as a basis for the design of such an explosion pressure venting system. Naturally, the system must be inspected regularly.

3.2.3.1 Underpressure Protection

After venting an explosion in very large enclosures such as pulverized fuel silos through explosion flaps considerable underpressure can develop inside the silo due to dynamic

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effects and due to cooling down of the hot gases remaining in the silo after the explosion. Typical examples for the size of underpressure valves are given in Table 3. Guidelines for the individual design of underpressure valves can be taken from the relevant literature (10).

Table 3

Volume	m^3	100	1000
Diameter	m	3.4	.5
Cylindrical length	m	9.5	22.0
Plate thickness	mm	6	8
Max. negative pressure	mbar	100	25
Required aspiration area	m2	0.1	1.0

3.2.4 Explosion Suppression

Techniques of explosion suppression can basically replace all the previously mentioned methods. However, in practical experience it has been seen that in pulverizing plants, the costs involved in the consequent application of explosion suppression techniques are significantly higher than they are for explosion pressure venting techniques and explosion pressure resistant construction methods, both with respect to procurement and maintenance of the sensitive equipment.

Thus applicability of explosion suppression may be primarily limited to existing, insufficiently protected pulverizing plants whose retrofitting in accordance with alternative protection techniques would be entirely uneconomical.

3.2.5 Fire Extinguishing Measures

If an accumulation of considerable quantities of combustible dust can be prevented inside the actual pulverizing plant (except in pulverized fuel silos), any fires that may arise following an explosion will not be able to grow to any significant size. The installation of a fire extinguishing system can nevertheless still be recommended for cloth filters and electofilters.

In the case of smoldering fires in pulverized fuel silos, all further fuel supply must be stopped immediately. Following this, the silo exit must be made airtight and the silo atmosphere flooded with CO_2. Sufficient time must now pass until the temperature conditions have normalized. An underpressure valve is required in order to avoid collapsing of the silo due to the vacuum produced during cooling down. The above procedures can take several days, depending on the size of the smolder location. An alternative technique is to deliver the fuel as quickly as possible to the burner system via the dosing and conveyor systems. Of course this method is possible only when the dosing and conveying systems are heat-resistant, dustproof and explosion resistant. In addition, under no circumstances is glowing fuel to be returned to the silo, as for instance via overflow feeders.

Fig. 7 shows the practical preventive safety measures for pulverized fuel silos. In Fig. 8 the application of design related protective measures for solid fuel preparation is illustrated.

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Fig. 7 Preventive and Safety Measures for Coal Dust Silos

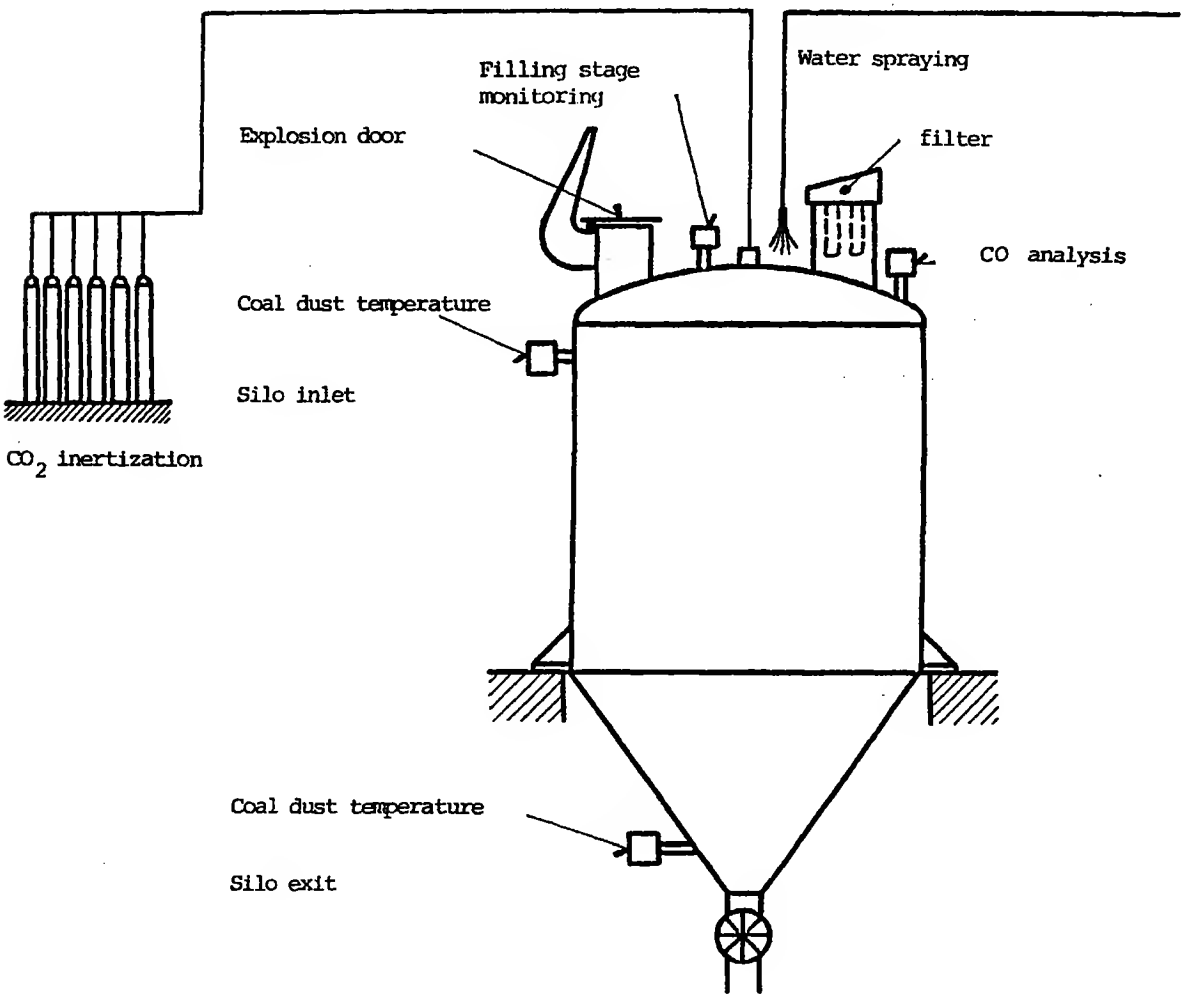

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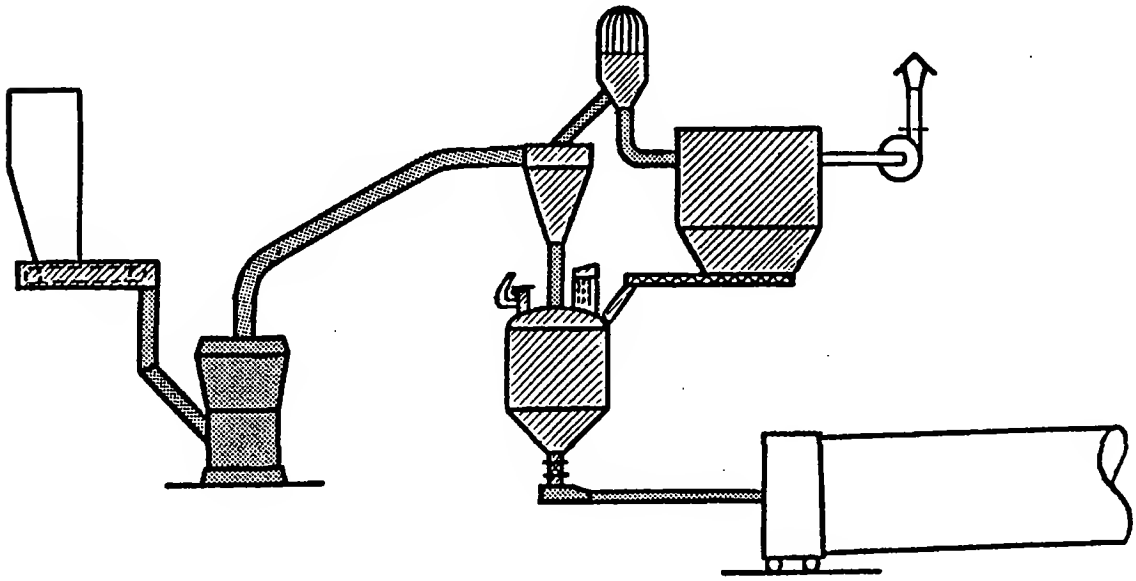
Fig. 8 **Example to Show the Application of Design Related Explosion Protection**



Explosion pressure vented, pressure-shock-resistant

Explosion pressure resistant (ND 6)

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4.1 Approximate Values for Explosion Limits and Ignition Temperatures

The numerical values of the following data are depending on the test procedure applied and can vary within certain limits according to the origin and geological age of the coals. The following values refer to the Literature (9).

- Explosion Limits

1) Dust concentration:
   * lower explosion limits 40 to 130 g/m3
   * upper explosion limits 2000 to 6000 g/m3
2) Oxygen concentration:
   * hard coal 14%
   * lignite 12%
3) Concentration of non-combustible parts (ash):
   * hard coal (-medium volatile bituminous) 65%

- Ignition Temperature

	Cloud °C	Layer °C
Lignite	380 to 450	225 to 300
Hard coal	590 to 710	245 to 380
Petrol coke	690	280

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Chapter 5

Burners and Flames

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Burners and Flames

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PT 98/14354/E

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Introduction

The function of the burner is to introduce the fuel into the burning zone.

The propagation of the combustion process depends on how fast the combustible comes into contact with oxygen. It is therefore the essential function of the burner to regulate this mixing process adequately in order to achieve a correct flame shape.

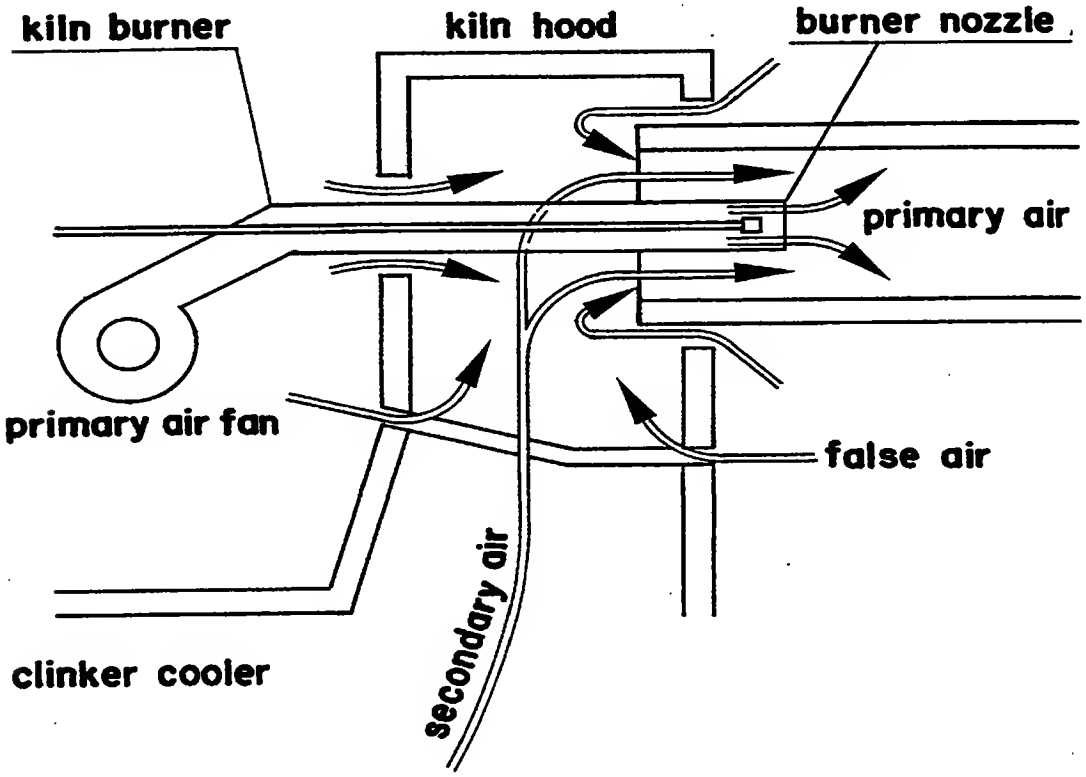
This process must take place in such a fashion that the heat is released at exactly the right place without producing any damaging effects and without producing excessive pollutant elements such as NOx, SOx and CO. Consequently, any optimization of the burning process must start with the correct adjustment of the flame.

This paper describes how the flame can be adjusted, what burner types are available and under what conditions they work best.

1. TERMINOLOGY

- Primary air + secondary air + false air = combustion air
- Stoichiometric combustion air + excess air = combustion air

Figure 1: Terminology of Combustion Air



2. BURNERS

2.1 Mono Channel - / Straight Burners

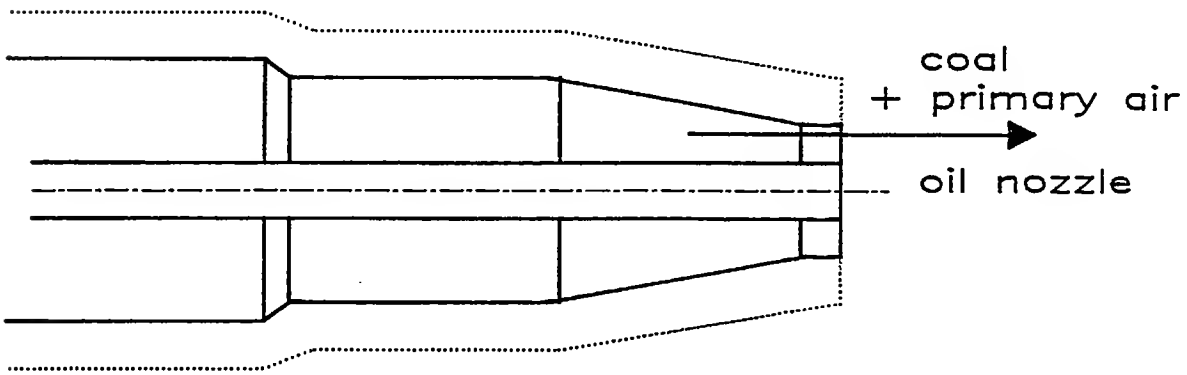
2.1.1 Burner Design

The mono channel burner is the most simple burner design. With this burner type, coal dust and all the primary air is injected together through a single tube. Usually this type is used for long kilns, equipped with direct firing.

Mono channel / straight burners can also be used for fuel oil firing or for a combination of coal and oil firing (additional channel for the oil nozzle in the center).

Conical burner tips can be used to increase the injection velocity (Fig. 2).

Figure 2: Straight Burner



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2.1.2 Burner Characterization

A high axial impulsion (massflow of fuel and primary air multiplied with the injection velocity) leads to an intense mixing of the combustion air with the fuel. This intensive mixing has two effects:

1) strong and stable flame; good (complete) combustion
2) high NOx formation

Recommended range of specific axial impulsion (Gax) for mono channel burners:

$$Gax = \frac{M_{(\text{transport air+fuel})} \cdot V_{\text{transport air}}}{Q_{\text{fuel}}} = 6 - 7\frac{N}{MW}$$

M: Massflow Transport Air + Fuel (kg/s)

V: Injection Velocity (m/s)

Q: Fuel Input (calorific value · fuel massflow) $\left[\frac{MJ}{kg} \cdot \frac{kg}{s}\right] = [MW]$

This corresponds with the old rule of thumb, which states that the kinetic energy of the primary air jet of a mono channel burner should be kept constant within certain limits:

$$(\text{Velocity of Primary Air})^2 \quad (\%\text{Primary Air}) = 65'000 - 75'000$$

Even if this formula will not give optimal values in each case, it enables a rough estimate of the dimension of the burner if presupposed as a second condition that the primary air jet velocity should lie between 50 and 100 m/sec (valid for straight burners without swirl).

2.2 Multi Channel Burners

2.2.1 Burner Types

The most simple design of a burner is the mono channel burner (see Chap. 2.1). However, for optimum flame shaping when considering changing coal quality and different requirements from the point of view of raw mix burnability, burners with adjustable flame are to be preferred. In such burners, the primary air is usually divided into an axial and a radial component with the coal also introduced via a concentric ring tube.

These burners are called multi channel burners, and are usually suitable for alternate or combined firing of coal, oil or gas. The axial air is injected in the direction of the kiln axis (similar to a mono channel burner where all the air is injected in axial direction).

The radial air or swirl air is injected with a direction towards the kiln wall.

The swirl component of the radial air creates a rotating air flow along the kiln axis (similar to the threat of a screw). This airflow is also pushing towards the outside, in direction of the kiln wall.

Since the radial air channel is located inside the axial air channel (both are concentric ring channels), the radial air is opening up / widening the flow of the axial air.

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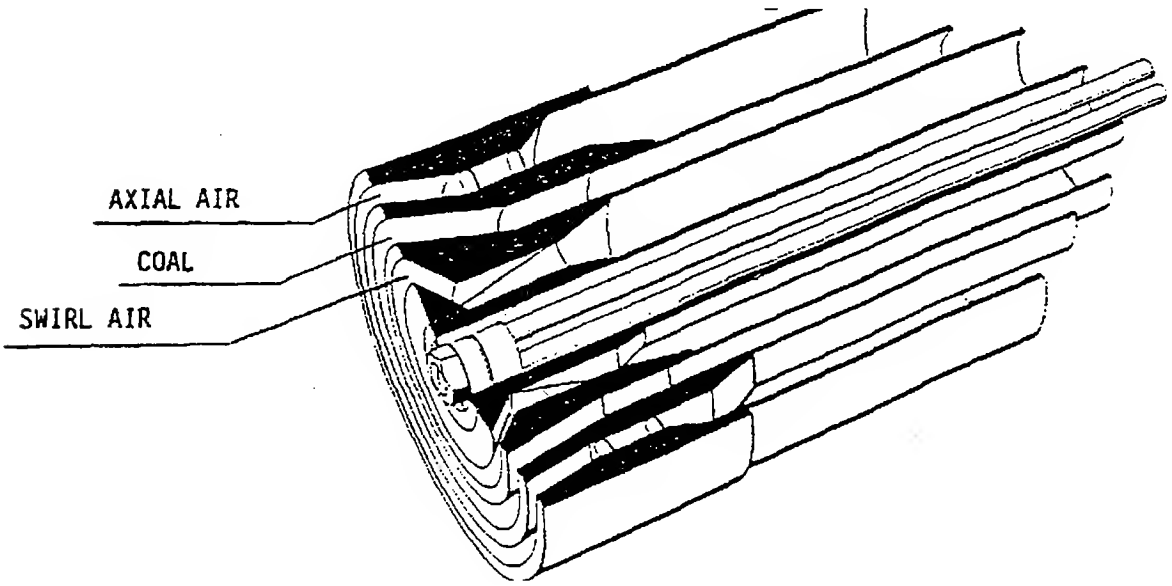
An increase of radial air versus axial air therefore creates a wider and shorter flame. An increase of axial air versus radial air create a longer flame.

Besides flame shaping, the primary air also has to keep the burner pipe cool.

A typical example of the first generation of multi channel burners is the Pillard 3-Channel Burner (Fig. 3). This burner has the coal channel in between the axial- and the radial air channel. A problem recognized with these burner types is that a shortening of the flame tends to produce a too wide flame (flame impeigements on the kiln wall).

Furthermore coarse cool particles (residue on 200 µm sieve) can be thrown out of the primary air jet by the radial air. These particles can cause reducing condition on the clinker bed and high NOx formation.

Figure 3: Conventional 3-Channel Burner (Pillard)



A new generation of multi channel burners has therefore been designed. With special arrangements and constructions of the primary air channels the above mentioned negative effects can be avoided through the creation of a longer and more homogenious internal recirculation zone in the flame (see chapter 3.2.2).

The leading burners of this generation are the Pillard Rotaflam and the KHD Pyrojet.

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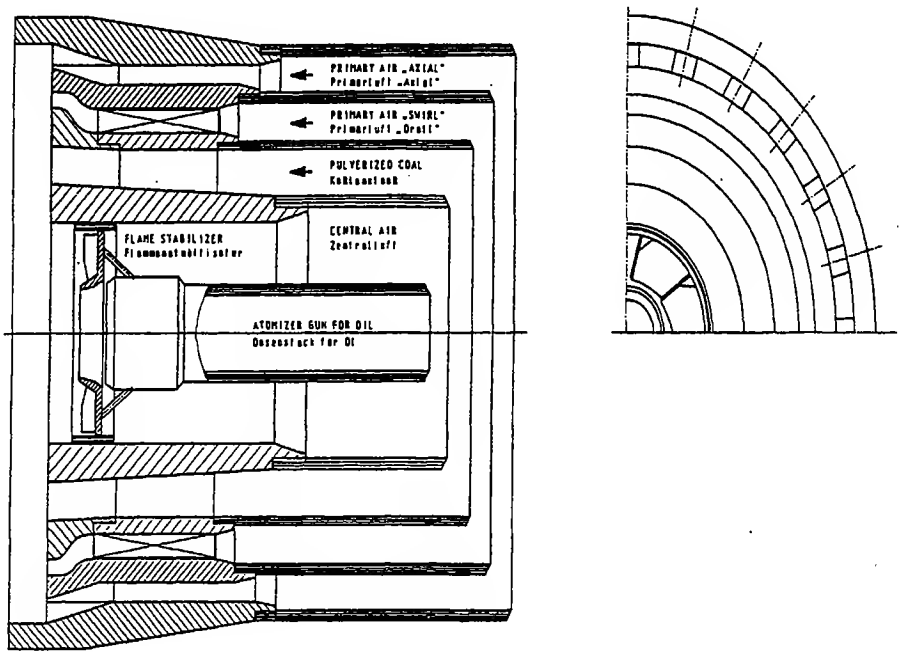
The particular features of the Pillard Rotaflam Burner (Fig. 4) are the location of the coal channel inside the axial and radial air channels, as well as the flame holder / flame stabilizer (bluff-body-effect) in the enlarged center cross section.

Figure 4: Pillard Rotaflam Burner



The particular feature of the KHD Pyrojet Burner (Fig. 5) is the jet air. The effect of this burner can be explained by the better and more uniform mixing of fuel and secondary air due to the jet air being introduced at nearly sonic velocity. For this reason the Pyrojet requires a compressor for the jet air.

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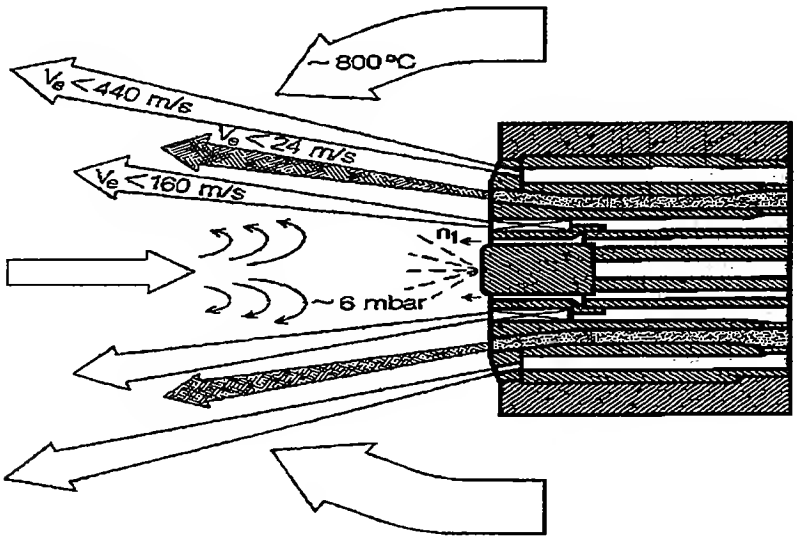
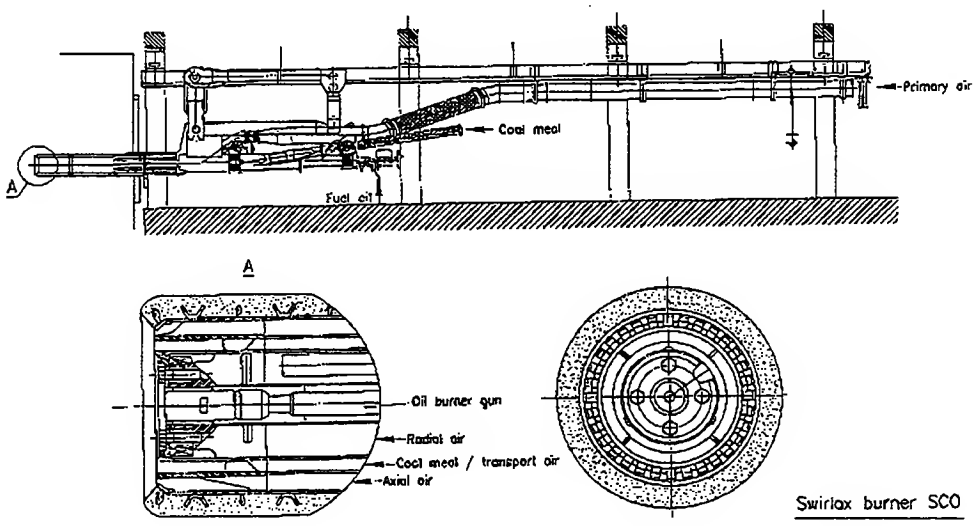
Figure 5: KHD Pyrojet Burner



The FLS Swirlax Burner (Fig. 6) applies Pyrojet technology with a license from KHD. The experience in the Holderbank Group is limited.

Figure. 6: FLS Swirlax Burner



Unitherm offers an interesting solution with their M.A.S. Burner (Fig. 7), featuring only one primary air channel with adjustable swirl. However, so far with no application within the Holderbank Group.

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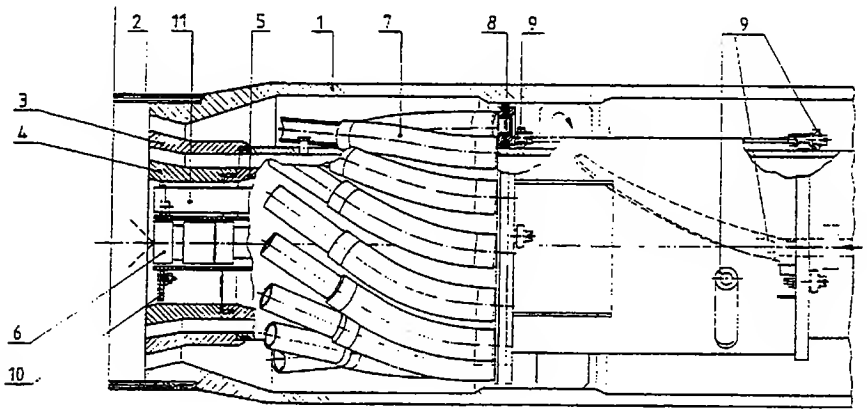
Figure 7: Unitherm M.A.S. Burner



2.2.2 Burner Design Recommendations

(Calculation of Gax and S_b : see Annex)

- Specific Axial Impulsion: Gax = 3 - 7

  low volatile coal (10% volatiles): Gax = 7

  high volatile coal (35% volatiles): Gax = 3

  The axial impulsion affects the overall entrainment into the flame. In general a higher axial impulsion results in enhanced mixing and higher NOx emission levels.

- **Swirl Number: Normal Range: S_b = 0.1 - 0.25**

  Maximum Range: S_b = 0.4

In general higher tangential momentum (expressed through the swirl number) results in a more rapid heat release in the near burner zone and higher NOx emission levels.

- **Primary Air Ratio: 10 - 12%**

Experience with these new generation (low primary air) burners has shown, that primary air ratios of 6 - 8% are on the technical limit below which it is no longer possible to guarantee stable combustion conditions.

With primary air ratios of 6 - 8%, any disturbance of the burning process tends to shift combustion to the back kiln zone, producing high kiln inlet temperatures and poor clinker quality (underburning). Therefore in designing the primary air content for modern burners, a minimum of 10% is recommended (including transport air in coal fired systems).

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Seen from a heat saving point of view the primary air ratio should be as low as possible in order to recuperate as much hot secondary air as possible. On the other hand, the kinetic energy of the fuel air mixture must be sufficiently high to provide a good mixture with the secondary air to ensure rapid burning.

- Axial air velocity (injection): 100 - 190 m/s (Pyrojet: 300 m/s)
- Radial air velocity (injection): 100 - 190 m/s
- Pressure of radial and axial air: 150 - 200 mbar (Pyrojet axial air: 0.5 - 1 bar)
- Transport air coal (injection): 20 - 30 m/s

2.3 Fuel Oil Atomizers

Once properly prepared in terms of filtering, heating up and delivering to the burner with constant pressure and viscosity, the fuel oil must be atomized for effective mixing with the combustion air. Therefore fuel oil atomizing nozzles are used. These nozzles are located in the center of the burner, surrounded by the injection of the primary air. The oil nozzle is held in place by a jacked tube which is a fixed part of the burner. Thus the atomizing nozzle is retractable, which is necessary to change the orifice plate when increasing the throughput (only mechanical atomizers with fixed orifice - see below) or to take out the oil nozzle whenever it is not needed (e.g. switching to coal firing) to prevent overheating or coking of the unused atomizer.

For fuel oil atomization different principles are employed:

- Mechanical atomization with fixed orifice and variable pressure
- Mechanical atomization with variable orifice and constant pressure
- Assisted atomization with steam or compressed air

2.3.1 Mechanical Atomizers with Fixed Orifice and Variable Pressure

This type of atomization is the most common. Hereby the oil throughput is governed by the pressure of the fuel oil (within the range given by the selected discharge opening/orifice plate).

With these atomizers the fuel oil flow in the atomizer head is often subdivided into an axial and a radial component. By adjusting the pressure and thus the ratio of these components, it is possible to alter the spray angle of the fuel jet. In general, an increase of the radial/tangential oil pressure leads to intensified swirling of radial and axial oil which has the tendency to shorten the flame. Typically the differential pressure is in the range of 1.5 bar (tangential minus axial oil pressure) with an overall pressure of approx. 40 bar. Since the reading accuracy of such small values, compared to the operating range of 40 bars, is unsatisfactory, it is suggested to equip both, radial and axial oil flow with oil flow measuring devices and optimize on flow basis using the flow-pressure curve of the nozzle supplier or to install a separate measurement of the pressure difference between radial and axial oil pressure. Flame shape control is, however, not only a result of atomizer adjustments, but also a function of primary air control.

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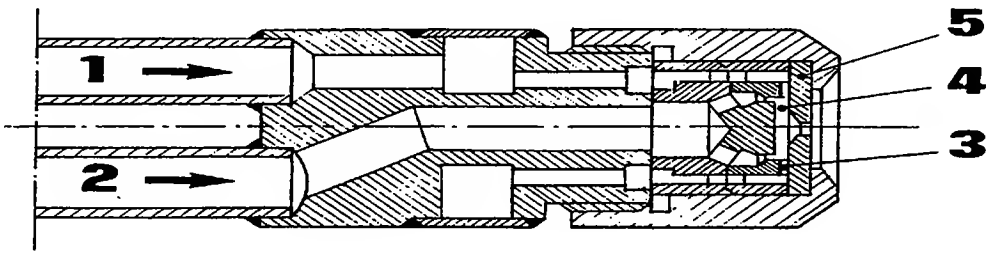
Fig. 8 and 9 show two current atomizers (Pillard and Unitherm) with radial-axial flow or alternatively return-flow for start-up operation. For return-flow, the axial oil flow is used to return a portion of the radial oil flow to the storage tank, in order to have a high flow velocity and oil pressure in the nozzle head (swirl chamber) despite the small amount of oil injected in the kiln (start up phase). Thus the turndown ratio can be increased, still with a good atomization.

Atomizer turndown ratios of 10 to 1 are often given by the suppliers. Practical turn down ratios (without changing the orifice plate) however, are limited to values below 5 to 1 (even for return flow operation during start up).

As an additional feature, the length of the swirl chamber in the Unitherm atomizer is adjustable.

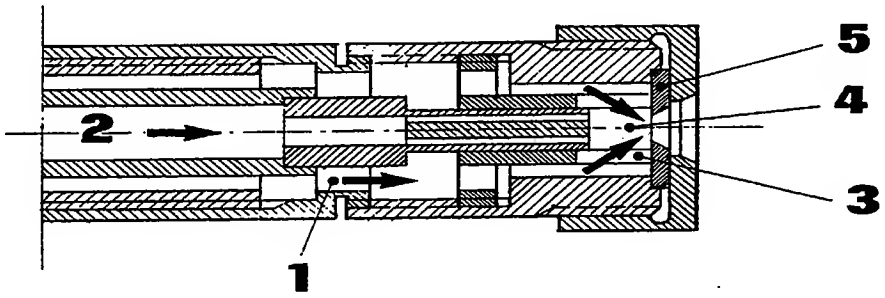
Fig. 10 (Coen Tri-Tip Nozzle) shows a mechanical atomizer with fixed orifice without radial-axial oil flow division.

Figure 8: Pillard MY Atomizer



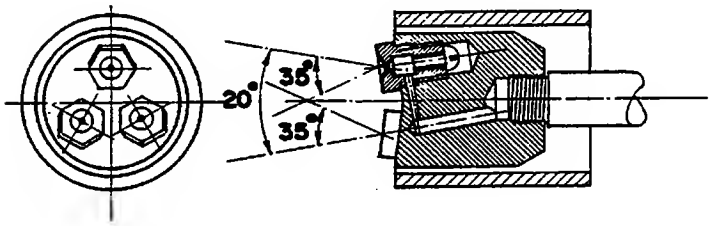
1 tangential oil flow
2 axial oil flow
3 tangential slots
4 swirl chamber
5 orifice plate

Figure 9: Unitherm Atomizer



1 tangential oil flow
2 axial oil flow
3 tangential slots
4 swirl chamber
5 orifice plate

Figure 10: Coen Tri-Tip Nozzle



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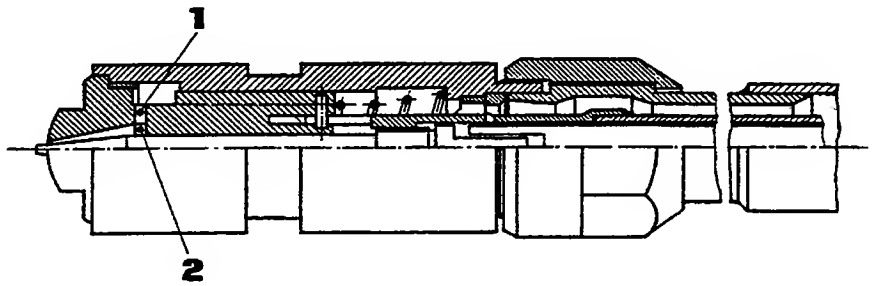
2.3.2 Mechanical Atomizers with Variable Orifice and Constant Pressure

This type of atomizer employs the adjustable needle valve principle for throughput control. By moving the needle position, contrary to the above described types, the orifice can be adjusted. Atomizing pressures are in the range of 20 bar. The turndown ratio are also limited. Needle value atomizers are mainly used by FLS for long wet kilns (see Fig. 11). Flame shaping is accomplished by adjusting the needle position, oil pressure and primary air.

Figure 11: FLS Atomizer (Needle Valve Principe)



1) tangential slots
2) swirl chamber

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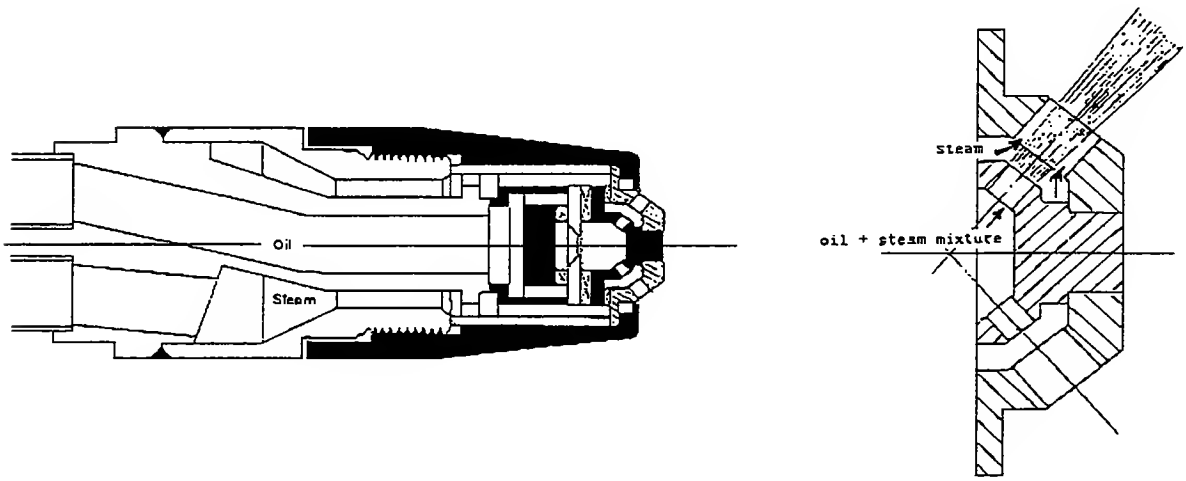
2.3.3 Nozzles with Assisted Atomization through Steam or Compressed Air

This type of atomizer (Fig. 12) uses steam or compressed air instead of radial oil to create an intense swirl in front of the orifice plate. The advantage of these atomizers is the higher turndown ratio because even a small amount of oil can be atomized effectively with steam or compressed air. The disadvantage of these atomizers is the need for a significant amount of steam or compressed air, which cost money to produce.

Figure 12: Pillard Atomizer with Assisted Atomization



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2.4 Natural Gas Burners

In most modern gas burners the gas flow divided is into radial and axial gas. Primary air is not necessarily needed. However most burners use primary air for flame shaping and burner cooling.

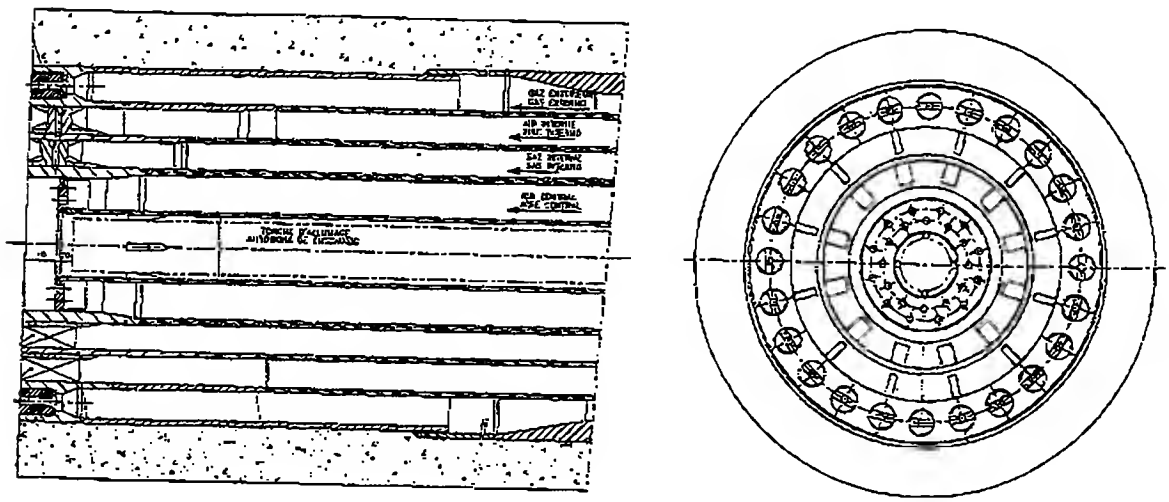
Pillard Rotagas Burner (Fig. 13)

The Rotagas burner is the most recent development from Pillard. The burner is designed for 100% gas firing. Compared with the conventional Pillard Gas Rotaflam, the possibilities to adjust the flameshape have been ameliorated.

Arrangement of the channels (from outside to the center):

- exterior, high pressure gas channel
- radial swirl air channel
- interior, low pressure gas channel
- central air channel
- center: jacket tube for ignition burner

Figure 13: Pillard Rotagas Burner



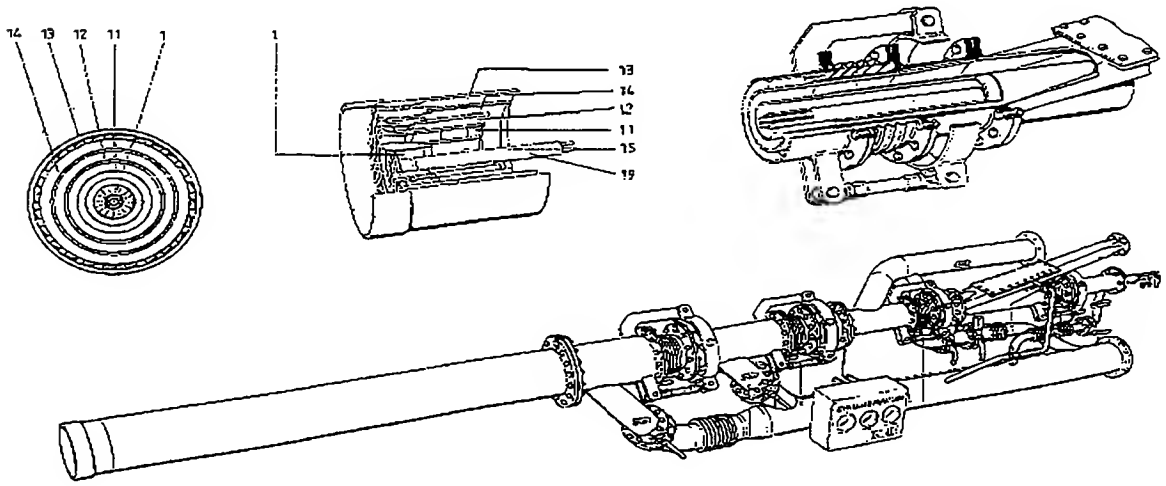
Pillard Rotaflam KGD Gas/Coal/Oil Burner (Fig. 14)

The Rotaflam multipurpose burner is equipped for combined or separate firing of gas, coal and oil. Flame shaping is achieved with separate axial and radial primary air.

Arrangement of the channels (from outside to the center):

- axial air
- radial swirl-air
- single gas channel
- pulverized coal channel
- central air / flame stabilizer
- center: jacket tube with oil atomizing nozzle

Figure 14: Pillard KGD Gas / Coal / Oil Burner



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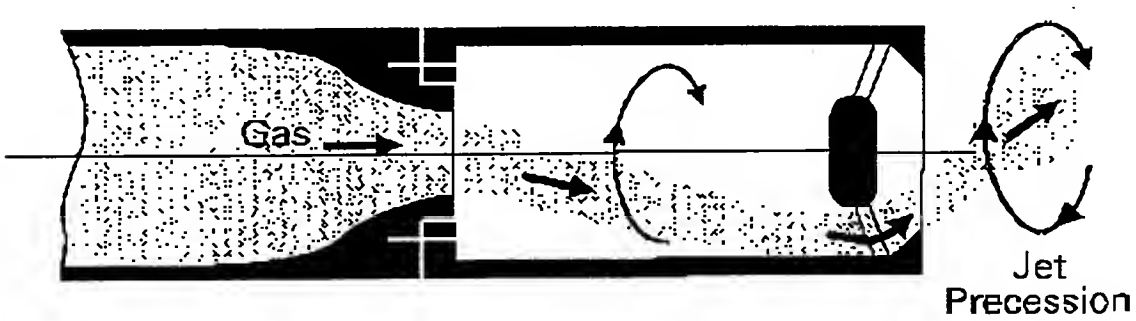
Gyro-Therm Gas Burner (Fig. 15)

The Gyro-Therm burner applies a special flow phenomena to achieve the air/gas mixing. A "processing jet" is generated in a specifically designed nozzle. Experiences with this burner are limited.

Figure 15: Gyro-Therm Gas Burner



KHD Gas Burner (Fig. 16)

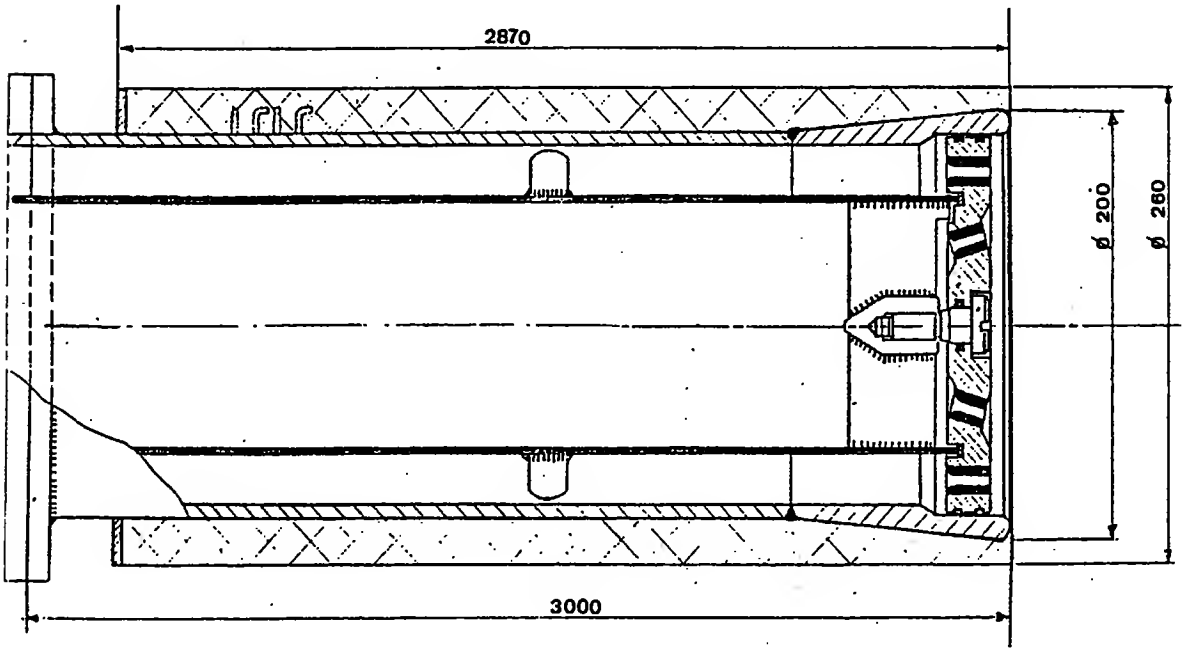
This burner has been used in various kilns since a long time. Owing to the principle on which it operates, it requires a rather high supply pressure (3 - 5 bar) to allow the fuel throughput and the shape of the flame to be varied.

Primary air is not needed for this burner.

Arrangement of the channels (from outside to center):

- axial gas channel
- center: radial gas

Figure 16: KHD Gas Burner



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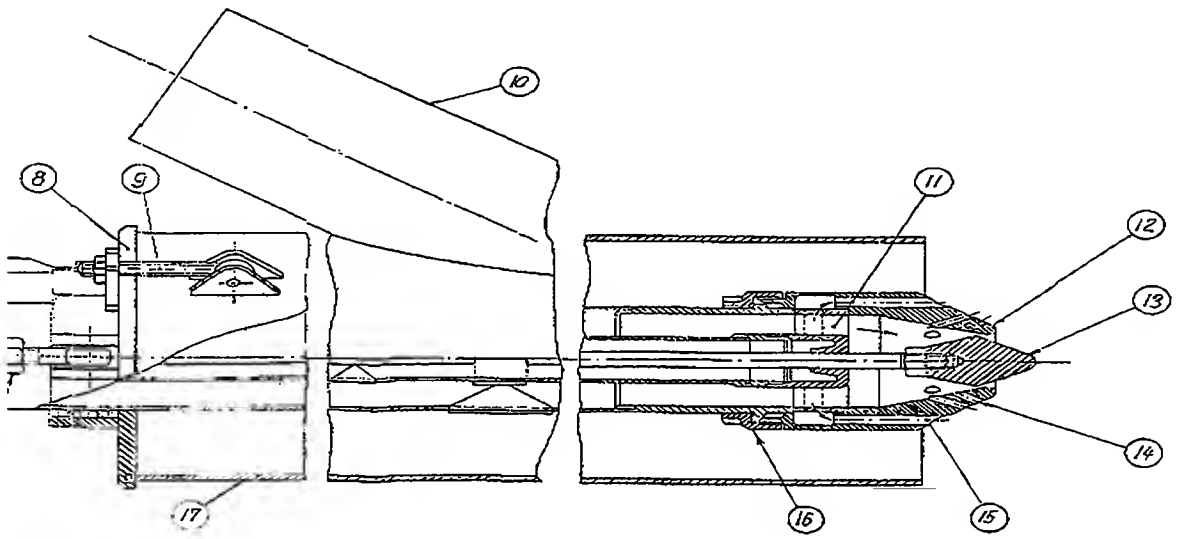
FLS Gas Burner (Fig. 17)

Flame adjustment is done with this burner using the "needle valve method".

Arrangement of the channels (from outside to center):

- primary air (10)
- secondary gas (15)
- primary gas (14)
- center: primary gas with regulating cone (13)

Figure 17: FLS Gas Burner



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3. FLAMES

3.1 Prerequisites for the Ideal Flame

The optimization of the burning has to start with the correct adjustment of the flame.

A look at the effects of poor burning makes this immediately clear:

- Unstable coating behavior, particularly in the transition zone, reduces the lining life.
- Incomplete burning and a local reducing atmosphere dramatically increase sulfur volatilization and build-up of coating in the preheater and in the kiln inlet area. Thus a significantly higher dust cycle is created which shifts the entire temperature profile toward the kiln inlet.
- With high CO-formation, secondary combustion forms at the back of the kiln which leads to ring formation.
- As a result, the kiln cannot operate at maximum output, the specific heat consumption increases and the efficiency of the unit drops.

The "ideal" flame can prevent, or at least keep within limits, the operating problems described above. The flame is stable over the entire burn-out distance:

- By continually mixing hot secondary air into the burning zone. Therefore combustion can take place in a controlled manner over the entire flame length.
- No local temperature peaks are formed.
- No local reducing conditions develop over the clinker bed.
- Burn-out is complete at the end of the sinter zone.

In addition this "ideal flame" has to be achieved with the lowest possible formation of NOx in the exhaust gas.

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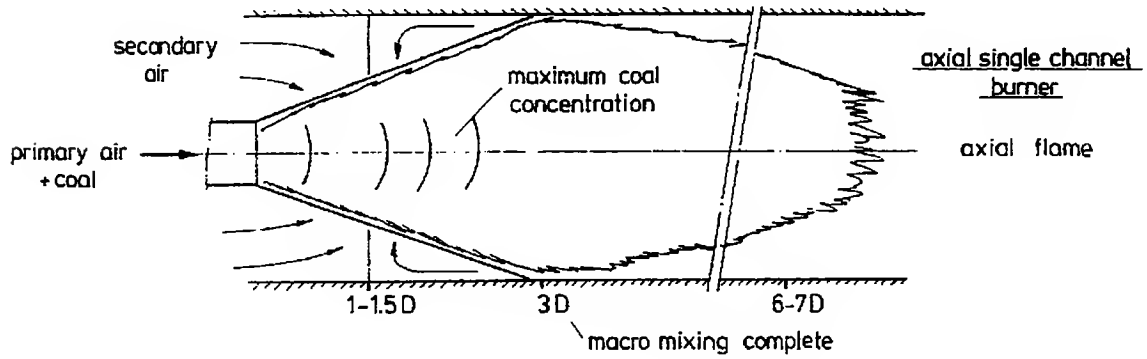
3.2 Flame Characteristics of the Different Burner Systems

3.2.1 Single Channel Burner

Straight burner pipes tend to produce an axial flame without internal recirculation. The heating up of the fuel jet to ignition temperatures is predominantly by external recirculation of the hot combustion gases (Fig. 18).

Figure 18: Flame Shape of Single Channel Burner



Effects:

- Long sinter zone
- Long retention time of the kiln charge in the hot zone and thus high volatilization of alkalis and sulfur (very suitable for the production of low-alkali clinker)
- High NOx formation

With a constant primary air ratio, the length of the flame reaches a minimum for a given primary air velocity. If the velocity is further increased, the primary air jet develops an excessive suction effect which results in a reverse flow of flue gases. The recirculating flue gas thins the secondary air so much that the flame becomes longer again.

For the recommended range of the specific axial impulsion see chapter 2.1.

However, for optimum flame shaping in response to changing production requirements, burners with adjustable flame (multi channel burners) are to be preferred.

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3.2.2 Multi Channel Burner

Multi channel burners can produce a divergent flame with internal and external recirculation zones. The ability to change the relationship between axial and radial air provides an important control mechanism for influencing the flame shape.

A hollow cone flame shape is produced, which can be modified by adjusting the pressure and/or injection-orifice of the radial and axial air (Fig. 19).

The first generation of three channel burners (e.g. Pillard 3-Channel) has some negative effects on the flame shape, if there is a high content of radial air used. Two different flame zones can appear:

- In the first zone with internal recirculation there is intense combustion. Depending on the arrangement of the swirling flow, in this zone coarse fuel particles are spun out of the flame and then burn quickly in the oxygen-rich atmosphere of the hot secondary air.
- In the second, long, instable zone, dominated by external recirculation, burn-out is completed.

Effects:

- Peak temperatures in the internal recirculation zone.
- With very divergent flames, there are problems with the lining.
- CO formation above the clinker bed caused by incomplete burn-out of the extended fuel jet.
- Unstable coating formation in a long transition zone, caused by an enlarged unsteady burn-out zone.
- Increased NOx formation because of the long retention time of the gasses at high temperatures.
- High sulfur volatilization because of the reducing zone above the clinker bed and the long retention time at relatively high temperatures.

The new generation of three channel burners (e.g. Pillard Rotaflam and KHD Pyrojet - see also chapter 2.2) has been optimized so that these effects are largely avoided. The special arrangement and construction of the primary air channels make the internal recirculation zone (IRZ, Fig. 19) longer and more homogenous. This reduces the length of the burn-out zone with external recirculation. To reduce NOx formation, these burners have been optimized for very low primary air quantities.

For a faster mixing of the primary air with the fuel, these burners have an enlarged flame stabilizer in form of a bluff body in the center.

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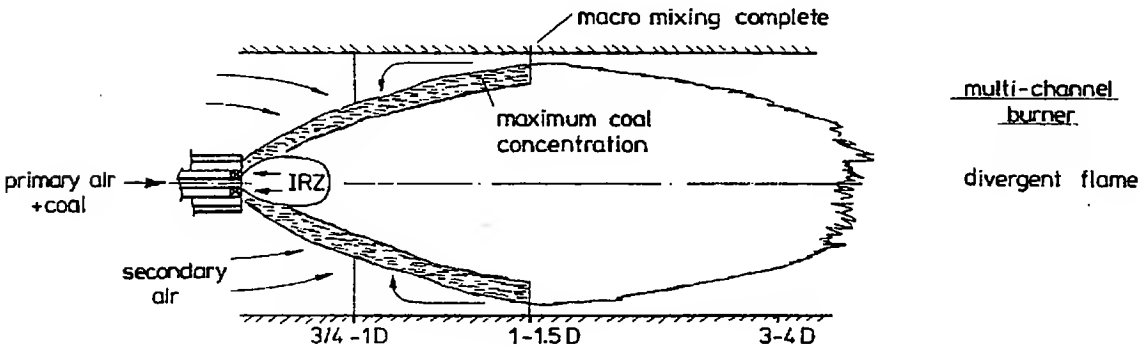
Figure 19: Flame Shape of New Generation Multi Channel Burner



Effects:

- Homogeneous temperature distribution, no excessive temperature peaks.
- Low volatilization rate of alkalis and sulfur.
- Homogeneous recirculation zone, and therefore low NOx formation.
- In some cases the flame is too long. Therefore a rearrangement of the coal channel in between the axial and radial air (Pillard Rotaflam) is under discussion.

3.3 Factors Influencing the Flame

In most cases the most favorable operation is achieved with a rather short and powerful flame, giving a high heat transfer rate to the material bed and a short and stable burning zone.

The flame shape may be optimized during operation by adjusting the following parameters:

3.3.1 Primary Air Momentum

A shortening of the flame is normally achieved by increasing the injection momentum of the primary air. With existing burners this can be achieved to a limited extent by increasing the radial air and decreasing axial air correspondingly. With jet burners (KHD) the flame can be optimized by varying number and diameter of jet nozzles and adjusting the jet air pressure.

For burner design recommendations: see chapter 2.2.2.

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3.3.2 Position of the Burner in the Kiln

One of the most pronounced influence on flame length is the position of the burner tip: Shifting the burner further in the kiln increases the flame length significantly and vice versa.

This is because the turbulence field of the in-flowing secondary air significantly improves the mixing of the fuel jet with the air. In planetary cooler kilns this effect is less noticeable as the position of the burner tip is defined by the kiln's internal cooling zone.

Recommendations for burner tip position (except planetary cooler):

- Dry kiln: Distance from rotary kiln end to burner tip ≤ 1 m.
- Long wet kiln: Distance rotary kiln end - burner tip approx. 1 m or a little more.

If the burner tip is too close to the rotary kiln end, overheating of the nose ring can occur.

3.3.3 Alignment of the Burner in the Kiln

Basically the burner should be aligned parallel to the kiln axis. In the cold kiln the burner should even be pointed slightly upwards, (specially long burners in kilns with planetary coolers), to compensate for the bending downwards in the hot kiln. If the burner is aligned horizontally (the kiln axis has an angle of approx. 3° to the horizontal) as is often seen, the flame tends to reach the material bed. A local reducing atmosphere is created resulting in high sulfur volatilization.

3.3.4 Secondary Air Temperature

The secondary air temperature defines, firstly, the ignition behavior of the flame (black plume) and, secondly, the possible flame temperature. Insufficient secondary air temperature has to be compensated by fuel, and this means an increase in the combustion gas quantity and a lengthening of the temperature profile. In point of fact, the clinker cooler operation is one of the main factors influencing the flame.

3.3.5 Excess Air

Some excess air is required for complete combustion. The optimum value for excess air to maintain the shortest possible sinter zone is about 10% (equal to 2% O_2 at kiln inlet). Burning at a too low excess air factor increases the burning time and hence the flame length. This creates a reducing atmosphere which increases sulfur volatility thus leading eventually to clogging problems in the preheating zone. If the excess air is significantly higher than the optimum value, the temperature profile is extended again because of a too low flame. This results in an insufficient temperature gradient towards the material bed and a longer sinter zone. For this reason, for example, the secondary firing rate for Air-Through systems is restricted to about 25 ± 5%.

3.3.6 Interaction Flame - Material Bed

As the heat transfer from the flame to the material bed in the sinter zone is almost entirely through radiation, the key factors affecting heat transfer are the temperature and the emissivity of the flame. If radiation is reduced by a dusty kiln atmosphere, a long drawn-out temperature profile with long sinter zone is produced. In this situation, the clinker dust is overheated in the flame and often deposited in the transition zone or even further upstream the kiln in the form of a clinker ring. Ways to counter this effect include all those measures which serve to improve clinker granulation (short and hot flame, different raw mix design).

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3.3.7 Burner Dimensions

Basically the burner must be of the right dimensions for nominal operation. This is observed particularly for burners in kilns with precalcination.

Oversized burner nozzles have to be operated with unfavorable primary air settings (either too high primary air content or too low primary air speed) and should be adjusted for nominal operation.

3.3.8 Pulverized Coal Characteristics

- Volatile content:

The combustion time of pulverized coal increases as the volatile content decreases, therefore low volatile coal has a longer burning time and ignition distance than bituminous coal.

- Grinding fineness:

The burning time of a coal dust grain increases approx. with the square of its diameter. The combustion time of a grain of coal increases as its volatile content decreases. Thus, low volatile coal must be ground more finely in order to burn within the desired time, e.g. in order to produce the desired flame length.

Recommendations for optimum grinding fineness:
see paper "Firing Systems - Handling and Preparation of Noble Fuels".

- Ash content:

A high content of ballast material (ash) has a retarding effect on the burning time caused by the reduced coal dust concentration and the lower flame temperatures as a result of the heat absorption of the ballast material.

- Rate of swelling:

The higher the expansion of the coal grain during heating in the flame, the shorter the burning time. Coal types with high density expand / swell less. Therefore petrol coks has to be ground finer to reach the same combustion time as regular coal.

3.3.9 Fuel Oil Flame Adjustments

A faster burn out of the fuel oil can be achieved by lowering the oil viscosity / increasing the oil temperature (recommendations for optimum oil temperature: see paper "Firing Systems - Handling and Preparation of Noble Fuels") or by better atomization (see chapter 2.3).

3.3.10 Natural Gas Flame Adjustments

The main requirement with natural gas burners is the possibility of producing a reverse flow zone in the center of the flame in order to achieve locally a reducing atmosphere where the hydrocarbon molecules agglomerate to larger chains. This is necessary to increase the emissivity of the gas flame, a prerequisite for heat transfer in the sintering zone.

Adjusting the shape of the flame visually is almost impossible, because clearly defined flame contours are hardly recognizable. Optimization of the flame shape should be done following the combustion indicators (see chapter 3.5).

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3.3.11 Combined Firing of Different Fuels

When firing two different fuels at the same time, the higher volatile fuel tends to burn more rapidly. this reduces the oxygen content so that the remaining fuel burns further to the back of the kiln.

However, a small amount of high volatile fuel can also have a positive effect on the flame, because it accelerates the ignition and burning of the other fuel.

In extreme cases, two separate burning zones are created. It is therefore important to improve the burning time of the less volatile fuel (e.g. by adjusting the fineness of grinding for coal).

3.3.12 Oxygen Enrichment

By adding oxygen to the combustion air, the flame temperature can be increased significantly. At the same time the specific exhaust gas quantity is lowered. This decreases the energy losses of the exhaust gas and allows to increase the production capacity of the kiln.

Practically feasible is the increase of O_2 in the combustion air by 2 - 3% (from 21% to 24%).

Disadvantages are the higher NOx formation, the lower cooler efficiency for planetary coolers and the oxygen costs.

3.4 Combustion Indicators

One of the main problems in the evaluation of the flame is that, in the rotary kiln, it is only possible to observe the flame visually to a limited extent. On closer consideration, however, there are a number of indicators which can provide much more information about the quality of the flame than can be obtained from simple visual observation.

In the following, the most important operation indicators (combustion indicators) with direct relation to firing parameters are discussed:

- Clinker quality (free lime, liter weight)
- Burning zone temperature (pyrometer, NOx, amps)
- Coating formation (indicated by kiln shell temperature profile)
- Exhaust gas composition (CO, O_2)
- Kiln inlet temperature
- Volatilization of circulating elements (hot meal analysis, encrustations in the preheater)

3.4.1 Clinker Quality

There exists a close interdependence between sintering zone temperature, granulometry, free lime and literweight of the clinker. The correlation of these parameters is to a high degree influenced by the flame shape. Fig. 20 shows an example, where by flame optimization, the literweight for the required free lime could be lowered. In other words, for the required clinker quality (free lime), burning could be done less hard (liter weight).

Burning less hard leads to substantial savings of energy and refractories.

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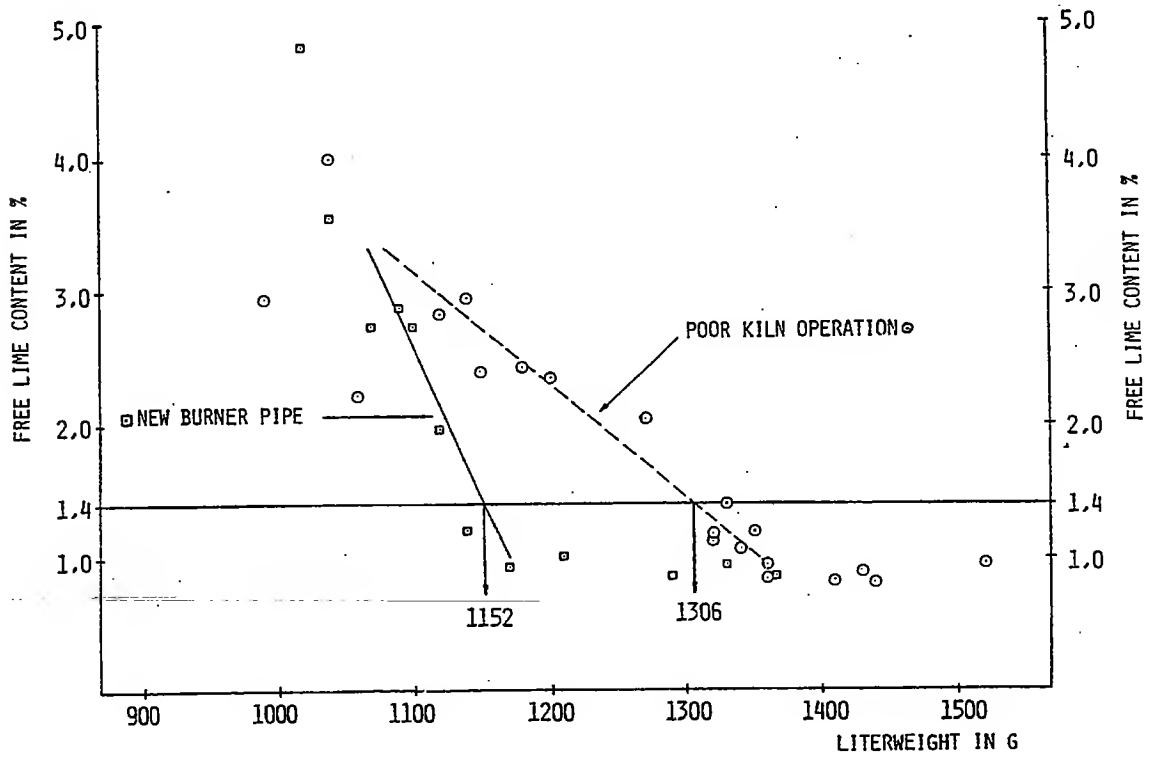
Figure. 20: Correlation between Free Lime Content and Literweight with Two Different Operating Conditions



When modifying the burner settings, the correlation of the parameters shown above has to be closely recorded before and after any change to the burner in order to draw the relevant information for optimum burner settings.

3.4.2 Sintering Zone Temperature

Information about the sintering zone temperature can be obtained by:

- Measuring the clinker bed temperature under the flame using a radiation pyrometer.
- Measuring the NOx concentration in the exhaust gas.
- Measuring the inclination of the kiln charge using a tallumeter.
- Measuring the kiln drive power consumption (Amps or kW) - (only reliable in some cases).

It has to be noted, that all the above mentioned measurements do not supply absolute but rather relative temperature indications and that the NOx-level is also highly depending on the flame characteristics (see chapter 3.6).

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3.4.3 Coating Formation

The length of the sinter zone and transition zone gives a guide to the length and temperature profile of the flame. Ring formation can indicate poor combustion, incorrect burner setting, or insufficient fuel preparation (coal not fine enough or poor oil atomizing).

Coating formation can be determined indirectly, by measuring the temperature profile of the kiln shell.

The influence of burner adjustments on coating formation can be checked by recording the kiln shell temperature profile before and after any change to burner settings.

3.4.4 Exhaust Gas Analysis

Exhaust gas analysis at kiln inlet supplies valuable information on the completeness of the combustion. Due to factors such as fluctuations in fuel supply and quality, generally a too high O_2 level would be required for 0% CO. Usually the kiln is set to an O_2 level at kiln inlet, at which < 500 ppm CO is produced. A thus required O_2 level in excess of 2.5% would indicate combustion problems.

Too high CO levels do not only cause energy losses but do also increase Sulfur volatilization and may cause Sulfur rings and cloggings in the cyclones.

3.4.5 Kiln Inlet Temperature

With preheater kilns, the kiln inlet temperature (= back end temperature) supplies information on flame length and retarded combustion. The target is to have the kiln inlet temperature as low as possible. Kiln inlet temperatures in excess of 1100°C need improvement of the firing system.

3.4.6 Volatilization of Circulating Elements

The flame has an important influence on the volatilization of the circulation elements, especifically on Sulfur. This is governed by factors such as retention time of the material in the hot zone (flame length) and local or general reducing atmosphere including the presence of oversize fuel particles in the material bed (Fig. 21).

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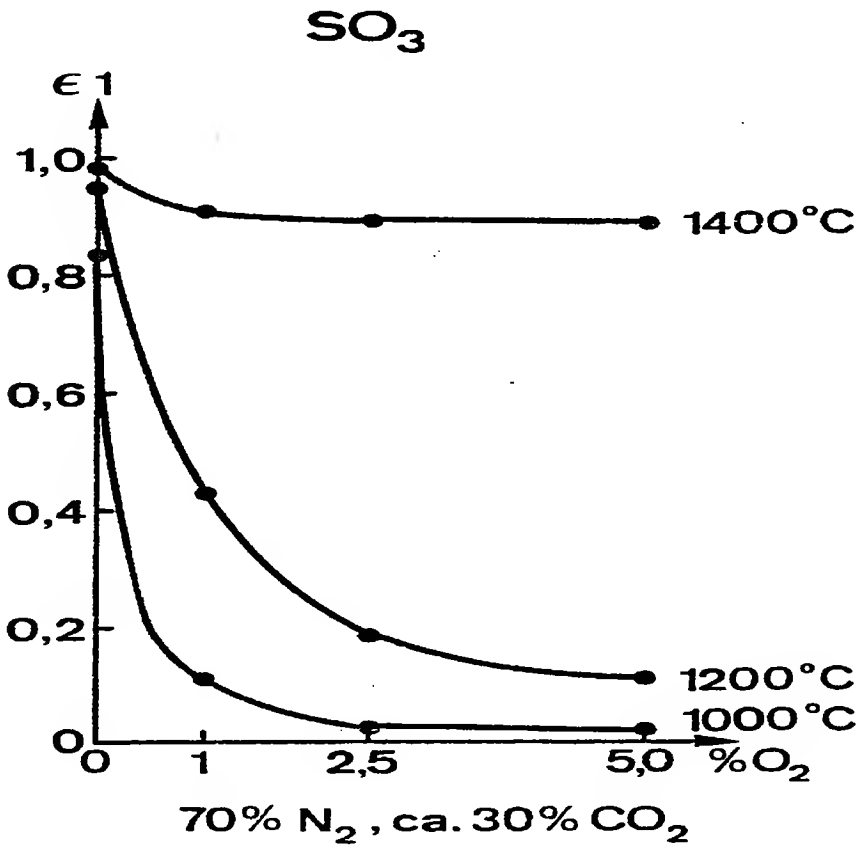
Figure 21: Influence of Temperature and O_2 Concentration on Sulfur Volatility



To assess the degree of volatilization of the circulating elements, the enrichment of SO_3, Cl and K_2O in the hot meal has to be measured before and after any change to the firing system.

3.5 NOx Formation

NOx formation is dominated as well by peak temperatures as by the amount of air entrained into the primary fuel jet at ignition. NOx reduction measures are deduced essentially from these facts as follows:

- Low primary air ratio
- Flame front near the burner (short ignition distance)
- Flame shaping with the aim to avoid high peak temperatures with at the same time shorter flame
- Lower burning temperatures (free lime, raw mix)

The minimum technically achievable NOx emission with measures related to the rotary kiln burner are in the order of magnitude of 800 to 1'000 mg/Nm^3 (dry basis). Further reduction of NOx requires additional secondary measures such as staged combustion (air / fuel staging, reburning) at the precalciner or NH_3 injection.

For more details on NOx formations see paper "State of Technology of Rotary Kiln Burners".

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3.6 Flame Adjustment Procedure

1) Follow the operating instruction of the supplier for a medium flame setting.
2) Wait until kiln is stable before undertaking any adjustment.
3) Progressively adjust parameters (axial/radial air, oil pressure, gas pressure) to get required flame.

- Cautions:
  - axial air outside channel also serves to cool the burner pipe; keep always min.8% of total primary air
  - watch continuously the corresponding combustion indicators
- The kiln reacts slowly to any change. It may take up to a few days to reach stable running conditions again. It is therefore useless to try to adjust a flame within one shift!

4) It is not recommended to operate the kiln with the shortest possible flame. A safety margin for adjustment in both directions should be maintained for control of burning zone disturbance.

NOx Emission: In some countries with severe regulations, the NOx emission might be in a near future the most important parameter for flame adjustment.

4. SECONDARY FIRING / PRECALCINER

The burning conditions for a secondary firing or precalciner burner are quite different from the kiln firing:

- In most cases the combustion takes place in a exhaust gas + air mixture instead of pure air.
- Combustion takes place in a very dusty atmosphere.
- Temperature range of 1000°C instead of 2000°C.

Due to the poor burning conditions, incomplete combustion is quite common in precalciners. Beside CO, coal firing produces carbon skeletons and also CH_4, which both cannot be traced by CO measuring equipment. Further signs for incomplete combustion are:

- Higher gas temperature at bottom cyclone outlet than at precalciner outlet.
- Only moderate drop of the gas temperature over the two lowermost cyclone stages.

Both indication an after-burning within the preheater. This results in increased exhaust gas temperature and heat consumption.

Improvement Measures:

- Avoiding fluctuations of the fuel feed.
- Grinding the coal to the required fineness.
- Providing enough gas retention time in the precalciner. As a rule of thumb for coal firing:
  - gas retention time = 2 to 3 sec.
  - (kiln capacity [t/d]) / (precalciner volume inside lining [m^3]) $= 7 \pm 2$ t/m^3 d

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2) "Flames and Burners"
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3) "State of Technology of Rotary Kiln Burners"
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6. ANNEX

6.1 Formulas and Definitions for the Calculation of Burner Momentum and Swirl Number

6.1.1 Primary Air / Combustion Air:

Kiln heat consumption	Q	[MJ/kgcli]	
Min. combustion air Amin.	0.26 x q	[Nm^3/kgcli]	Good approximation for conventional fuels
Total combustion air A	n x Amin.	[Nm^3/kgcli]	n = excess air factor, n>1
Excess combustion air	Amin. x (n-1)	[Nm^3/kgcli]	
Primary air ratio	Expressed in % Amin.	[%A min.]	

Note:

In order to get lower primary air ratio figures, burner suppliers usually relate primary air ratio to total combustion air.

6.1.2 Burner Geometry:

Following burner geometry calculations are based on the list of symbols and units stated below:

r_i	Burner channel radius of channel I	[m]
$r_{eg.i}$	Equivalent channel radius of channel I	[m]
G_x	Axial momentum	[N]
$G_{x.i}$	Axial momentum of channel I	[N]
G_{ax}	Specific axial momentum	[N/MW]
G_t	Tangential momentum	[N]
M_i	Mass flow through channel I	[kg/s]
Q_{fuel}	Fuel heat input	[MW]
S_b	Burner swirl number	[-]
$v_{i.ax}$	Axial velocity in channel I	[m/s]
$v_{sw.tan}$	Tangential velocity on swirling channel	[m/s]

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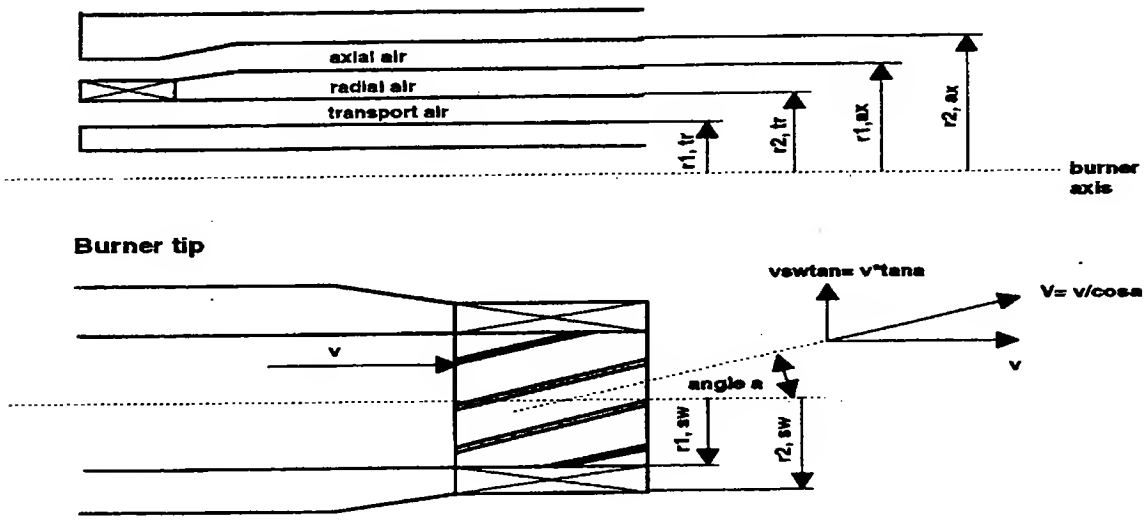
Figure A: Typical Burner Geometry



6.1.3 Mono Channel Burner:

Total specific axial momentum through burner [N/MW]:

$$G_{ax} = \frac{M_{(tr+c)} \times v_{tr}}{Q_{fuel}} \left[\frac{N}{MW}\right]$$

6.1.4 Multi Channel Burner:

Total specific axial momentum through burner [N/MW]:

$$G_{ax} = \frac{\left(M_{sw} \times v_{sw,ax} + M_{(tr+c)} \times v_{tr} + M_{ax} \times v_{ax,ax}\right)}{Q_{fuel}} \left[\frac{N}{MW}\right]$$

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Burner Swirl Number:

$$Swirl\,number = \frac{Tangential\,Momentum[N] \times Characteristical\,Swirl\,Radius[m]}{\Sigma(Axial\,Momentum[N] \times Characteristical\,Channel\,Radius[m])}[-]$$

$$S_b = \frac{Gt[N] \times r_{eq.sw}[m]}{\Sigma G_{xi}[N] \times r_{eq.i}[m]}[-]$$

A common method for the calculation of the characteristical or equivalent radius is to determine the radius of gyration for each individual channel cross-section as follows (Mathur and Maccallun - 1967):

$$r_{eq.i} = \frac{2 \times \left(r_2^{\,3} - r_1^{\,3}\right)}{3 \times \left(r_2^{\,2} - r_1^{\,2}\right)}[m]$$

For a typical multi channel coal burner with axial-, transport- and swirl air, the burner swirl number can be calculated according to the following formula:

$$S_b = \frac{Msw \times v_{sw,\tan} \times r_{eq.sw}}{M_{ax} \times v_{ax} \times r_{eq.ax} + M_{(tr+c)} \times v_{tr} \times r_{eq.tr} + M_{sw} \times v_{sw} \times r_{eq.sw}}[-]$$

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Chapter 6

Combustion Engineering

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Combustion, Gasflows and Gas Composition

A. Obrist
VA 89/5584/E

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SUMMARY

In context with gas measurements on a cement kiln there are important numerical relations which must be understood.

Such numerical relations involve the following subjects:

- Fuel properties, calorific value
- Raw meal properties
- Combustion calculations
- Gas composition
- Gas properties
- Gas quantities
- False air

Calculations can be done with complete, exact formulas but sometimes also by using approximations. Approximations are never 100% precise but often sufficient for practical purposes. Important, basic approximations given in this chapter are e.g.:

- Min. combustion air = $A_{min} \approx 0.26\ Nm^3/MJ$
- Min. combustion gas = $V_{min} \approx 0.28\ Nm^3/MJ$

The exhaust gas of a cement kiln consists of:

- Gas from raw meal
- Gas from combustion
- Excess air and false air
- Water from slurry or raw meal

By considering the above contributions the exhaust gas quantity of a kiln can be calculated.

This method of calculation and many other relations are given in this chapter.

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1. INTRODUCTION

Once, in the Greenfield cement factory, they wanted to do a few measurements on their kiln system, which seemed to run at its capacity limit. The specialist who was in charge of these measurements wanted to know the actual gas flows in the kiln system. Unfortunately, a few years ago, the designers of the Greenfield kiln had to fight with some difficult space problems when they had to plan the layout of the gas ducts in the narrow space. They never had considered that somebody would have to take flow measurements in this ductwork. The shape of the gas ducts was far away from the „ideal, long straight tube" for a precise flow measurement.

After a hard job within the hot areas of the gas ducts the specialist realized that his results from the pitot tube measurement was still not as precise as he had imagined. He therefore started to think about his problem. Maybe, there was another method to come to a result? Obviously gas flow has something to do with the fuel combustion and also raw meal produces some gas. So, why not calculate the gas from other parameters such as fuel quantity and raw meal? And wouldn't it be possible to calculate also the gas composition? But where do I find the necessary relations and formulas?

With this kind of thoughts in mind the specialist in the Greenfield plant was about to use the paper on hand, and finally to do a more interesting and effective job.

2. COMBUSTION

2.1 Basic Relations and Definitions

2.1.1 Combustion

Combustion of fuels is a chemical reaction of fuel with oxygen (O_2) according to the general scheme:

FUEL	+	OXYGEN	→	COMBUSTION PRODUCTS (CO_2, H_2O, SO_2, Ash)

Combustion reactions usually go through intermediate steps where some intermediate products may occur. However after complete combustion (total oxidation) the resulting combustion products are of extremely simple nature because complete combustion always ends at only 3(!) simple gas molecules, namely CO_2, H_2O and SO_2.

The overall combustion reactions can therefore be characterized by three very simple combustion equations:

$C + O_2$	→	CO_2
$2H + ½ O_2$	→	H_2O
$S + O_2$	→	SO_2

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2.1.2 Air

Oxygen for combustion of fuels is taken from the air. For the considerations within this chapter the composition of the dry air can be simplified (neglecting trace gases) as follows:

OXYGEN (O_2)	21.0 Vol%
NITROGEN (N_2)	79.0 Vol%
AIR	100.0 Vol%

According to the local climate air contains some water vapor, e.g. 2 Vol% at 20°C and 80% relative humidity.

2.1.3 Normal Conditions

„Normal conditions" for gases are defined as:

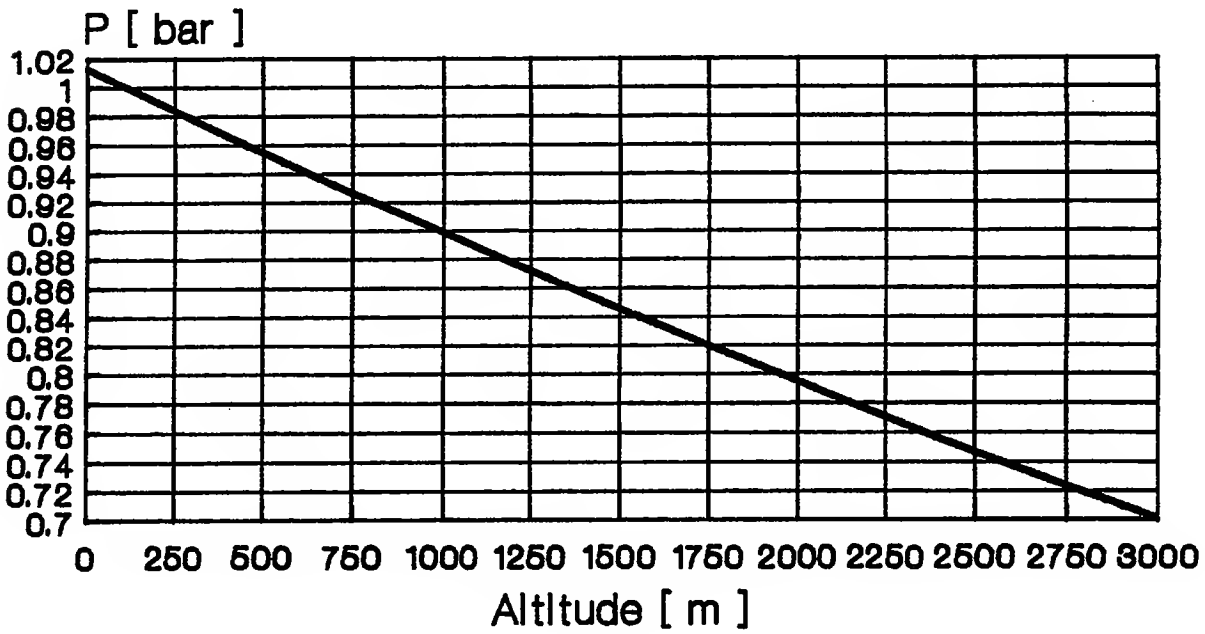
- Pressure = 1.0133 bar = 760 Torr
- Temperature = 0°C = 273.16°K

By the above conditions, the Nm^3 (Normal-cubicmeter) is defined which is used as unit for gas quantities.

Note:

1) The „normal conditions" refer to the average atmospheric pressure at sea level (Fig. 1 at altitude = 0 m) and at the zero point of the Celsius temperature scale (0°C).
2) There is a similar definition of „standard conditions" (USA) which refers to the 60°F point of the Fahrenheit temperature scale (30 inch Hg = 1.016 bar, 60°F = 15.6°C). Unfortunately these conditions are not equal to the „normal conditions" in the metric system.

Fig. 1 Average Barometric Pressure in Function of Altitude



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2.1.4 Kmol and Ideal Gas

To characterize the quantity of substances in context with chemical reactions the unit kmol is used.

One kmol means a certain number of molecules (Avogadro's constant = 6.02×10^{26}/kmol).

If the molecular weight of a compound is taken in kg it equals to 1 kmol of this substance. For example: The molecular weight of CO_2 is 44. Therefore 44 kg of CO_2 are equal to 1 kmol (which contains 6.02×10^{26} molecules) of CO_2.

One of the basic relations of chemistry and thermodynamics says that a certain number of 1 kmol of any gas molecules takes always the same volume (at constant pressure and temperature). The formula of the gas molecules does not have any influence on their volume in gaseous state.

To say this more precisely it must be added that this applies only for ideal gases. Gases at low partial pressures and at temperatures above the critical point can be considered as ideal gases. All gases occurring in context with this chapter can be considered as ideal gases with sufficient accuracy (approx. 0.1%).

This fact can be used by the following quantitative relationship:

- 1 kmol of an ideal gas takes a volume of 22.4 m^3 at normal conditions (1.0133 bar, 0°C)

Or in short form:

- 1 kmol of gas = 22.4 Nm^3

Note that even H_2O and CO_2 behave nearly like ideal gases as long as they occur in gas mixtures at low partial pressures.

2.1.5 Conversion of the Volume of Ideal Gases

Ideal gases or mixtures of ideal gases behave according to the fundamental equation:

- $pv = RT$
  (p = absolute pressure, V = volume, R = gas-constant, T = absolute temperature)

Therefore, volume conversions from condition 1 to condition 2 can be done easily by means of the ratios of absolute temperature [K] and absolute pressure as follows:

$$V_2 = V_1 \times \left(\frac{P_1}{P_2}\right) \times \left(\frac{T_2}{T_1}\right)$$

Example: Convert V_1 = 1 m^3 at 350°C (623.16 K) and 0.9 bar to normal conditions:

$$V_2 = 1m^3 \left(\frac{0.9\,\text{bar}}{1.0133\,\text{bar}}\right) \times \left(\frac{273.16\,\text{K}}{623.16\,\text{K}}\right) = 0.389 Nm^3$$

This type of calculation is frequently used for practical gas flow calculations. However, for the considerations within this paper it will not be required in the following.

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2.1.6 Minimum Air and Air Factor

To perform a complete combustion a theoretical minimum amount of oxygen O_2 min) is required, depending on the type of fuel. The corresponding minimum quantity of air is called A_{min}:

$$A_{min} = \frac{O_2 min}{0.21} [Nm^3]$$

Practically a combustion requires always slightly more air than the theoretical minimum A_{min} in order to avoid local lack of O_2 and unburnt products. The air factor „n" is defined as the following ratio:

$$n = \frac{A}{A_{min}} [-]$$

A is the effective air quantity, n must be always higher than 1 in order to maintain complete combustion.

2.2 Calorific Value

2.2.1 Gross and Net Calorific Value (CV)

The calorific value of a fuel sample is measured in a bomb calorimeter. The fuel sample and a surplus of oxygen are ignited in the bomb and after combustion the amount of heat is measured. The bomb is thereby cooled to room temperature level.

By cooling the calorimeter the water vapor is condensed and therefore the heat of water condensation (2450 kJ/kg H_2O at 20°C) is included in the resulting heat.

The direct result (amount of heat) obtained from the calorimeter is therefore always the so called gross calorific value CV_{gross} (in German: oberer Heizwert, Brennwert, H_o) of the fuel.

The gross value, however, is not very significant for common technical applications because the effect of water condensation does usually not occur. Therefore the net calorific value (in German: unterer Heizwert, H_u) value is calculated by subtracting the heat of condensation, as follows:

$CV_{net} = CV_{gross}$ - (water in combustion products) x 2450 [kJ/kg fuel]

The water in combustion products is calculated as follows:

$H_2O = H_2O_{fuel} + 9 \times H_{fuel}$	[kg/kg fuel]

↑hydrogen in fuel [kg/kg fuel]

Important Note: Within this chapter and also within the cement course chapter on Heat Balances the net calorific value is used as reference. Unless otherwise noted, fuel energy or fuel heat always refers to net calorific value.

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2.2.2 Calculation of CV

2.2.2.1 Gaseous Fuels

If the gas components are known by analysis the calorific value can be calculated exactly by adding the contributions of the pure gas components.

The gas analysis is given as volume % (which is the same as mole %) and the calorific value is usually referred to one Nm^3 (not kg of gas).

Gas Component	**Formula**	**CV net [MJ/Nm^3]**
Methane	CH_4	35.8
Ethane	C_2H_6	63.8
Propane	C_3H_8	91.3
Butane (gas)	C_4H_{10}	118.8
Pentane (gas)	C_5H_{12}	146.1
Ethylene	C_2H_4	59.1
Propylene	C_3H_6	86.1
Acetylene	C_2H_2	56.1
Carbon monoxide	CO	12.6
Hydrogen	H_2	10.8
Hydrogen sulfide	H_2S	23.2

Example:

Natural gas:	CH_4	=	90.5%
	C_2H_6	=	2.0%
	C_3H_8	=	0.5%
	CO_2	=	0.4%
	N_2	=	6.6%
			100.0%

$CV = 0.905 \times 35.8 + 0.02 \times 63.8 + 0.005 \times 91.3 = 34.1$ MJ/Nm^3

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2.2.2.2 Liquid and Solid Fuels

Usual liquid and solid fuels (fossil fuel oil and coal) consist of complex organic compounds. Usually the ultimate analysis or the elemental analysis (content of C, H, S, N, O) is available.

Strictly speaking it is impossible to calculate the exact calorific value of a fuel only from its elemental composition without knowing the kind of its organic compounds. However, as long as fossil fuel oils and coals are considered, the following approximation produces fairly accurate results:

$CV_{net} \approx$	34.8 x C + 93.9 x H + 10.5 x S 6.2 x N - 10.8 x 0 - 2.5 x W	[MJ/kg]

C, H, S, N, O, W are the weight fractions [kg/kg fuel] of carbon, hydrogen, sulfur, nitrogen, oxygen and water.

The above approximation produces also good results for other organic materials such as wood, paper and peat. But is should not be used for extreme cases such as e.g. pure carbon or pure sulfur.

Note that for exact determination of the CV of fossil fuel oils and coals only the calorimeter method - and not the above approximate calculation - can give the correct result.

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2.2.3 Examples of Calorific Values

(Including also alternative fuels)

	CV [MJ/kg] (net)	
Pure polyethylene	46	
Light oil	42	
Heavy oil	40	
Pure polystyrene	40	
Pure rubber (without inert material)	36	
Anthracite	34	High grade
Waste oils, various refinery wastes	30 to 40	
Petcoke	33	
Waste tires (with steel and inert material)	28 to 32	
Bituminous coal (low ash)	29	
Bituminous (high ash)	24	
Acid sludge, acid tar (from oil refining)	16 to 22	
Lignite (10% moisture)	16 to 21	
Pot liners (from aluminium smelter)	20	
PVC	19	
Palm nut shells (10% moisture)	19	Medium grade
Pressed olive cake	18	
Dried peat (10% moisture)	18	
Fuller earth (from oil refining)	13 to 18	
Dried wood, bark, saw dust (10% moisture)	16	
Rice husks (10% moisture)	16	
Shredder wastes	15	
RDF (from domestic refuse, 10% moisture)	15	
Cardboard, paper (air dry)	15	Low grade
Dried sewage sludge (10% moisture)	10	
Domestic refuse (30% moisture)	8.5	
Pure iron (heat of oxidation!, occurs e.g. in waste tires)	7.5	

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2.3 Combustion Calculations

2.3.1 Exact Calculations

The following calculations are based on the combustion equations (see 2.1.1) and some basic relations.

To demonstrate the calculation method, the simple example of combustion of pure carbon is given as example.

2.3.1.1 Combustion of Pure Carbon (C)

The combustion of 1 kg of pure carbon (C) without any excess air (n = 1) is considered.

The combustion equation and the corresponding weights and volumes are as follows:

C	+	**02**	→	**C02**
= 12 kg/kmol		= 32 kg/kmol = 22.4 Nm3/kmol		= 44 kg/kmol = 22.4 Nm3/kmol

The above equation refers to 1 kmol or 12 kg of C. The final results are wanted per 1 kg of C i.e. per 1 kg of fuel.

- The minimum oxygen is:

  O_{2min} = 22.4 Nm^3/12 kg = 1.87 Nm^3/kg C

- Air contains 21% O_2, therefore:

  A_{min} = 1.87 Nm^3/kg / 0.21 = 8.89 Nm^3/kg C

The combustion products are only CO_2 and N_2 (coming from A_{min}):

CO_2	=	22.4 Nm^3/12 kg	=	1.87 Nm^3/kg C
N_2	=	A_{min} x 0.79	=	7.02 Nm^3/kg C
Total combustion gas	=	V_{min}	=	8.89 Nm^3/kg C
Calorific value of pure C	=	CV (from handbook)	=	32.8 MJ/kg C

If A_{min} and V_{min} are referred to the CV the following results are obtained:

A_{min}/CV	=	0.271 Nm^3/MJ
V_{min}/CV	=	0.271 Nm^3/MJ

From this simple example it would appear that $A_{min} = V_{min}$, but in general the V_{min} will be a few percent higher than A_{min}. Nevertheless the example of pure C gives already a fairly representative impression of a typical combustion calculation.

2.3.1.2 General Case

In general a fuel is given either by its elemental composition or by its volume composition in case of gases. Therefore two types of formulas are required (2.3.1.2.1 and 2.3.1.2.2) which are based either on weight composition or on volume composition of the fuel.

A_{min} = Minimum air

V_{min} = Minimum combustion air

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2.3.1.2.1 Calculation based on Elemental Analysis / Weight Composition

C, H, S, N, O, H_2O are the weight fractions (kg/kg fuel) of carbon, hydrogen, sulfur, nitrogen, oxygen and water in the fuel.

$$A_{min} = 8.89 \times C + 26.5 \times H + 3.3 \times S - 3.3 \times 0 \quad [Nm^3/kg\ fuel]$$

$$V_{min} = 0.79 \times A_{min} + 0.8 \times N + 1.87 \times C + 0.7 \times S + 11.2 \times H + 1.24 \times H_2O + A_{min} \times H_2O\ air \quad [Nm^3/kg\ fuel]$$

The terms in the V_{min}-formula mean:

- $0.79 \times A_{min} + 0.8 \times N \rightarrow N_2$ }
- $1.87 \times C \rightarrow CO_2$ } dry gas
- $0.7 \times S \rightarrow SO_2$ }
- $11.2 \times H + 1.24 \times H_2O + $ + Amin x H_20 air $\rightarrow H_2O$ wet

Example:

Coal:

C	=	71.0%
H	=	4.0%
O	=	6.0%
N	=	1.5%
S	=	1.0%
H_2O	=	0.5%
Ash	=	16.0%
		100.0%

Moisture of air = O

$A_{min} = 8.89 \times 0.71 + 26.5 \times 0.04 + 3.3 \times 0.01 - 3.3 \times 0.06 = 7.207\ Nm^3/kg\ coal$

$V_{min} = 0.79 \times A_{min} + 0.8 \times 0.015 + 1.87 \times 0.71 + 0.7 \times 0.01 + 11.2 \times 0.04 + 1.24 \times 0.005 = 7.494\ Nm^3/kg\ coal$

2.3.1.2.2 Calculation based on Volume (or Mole) Composition
(only for gaseous fuels)

The general formula of any compound (also inert gases) in the fuel gas is defined as:

$C_c\ H_h\ S_s\ O_o\ N_n$

Examples:	CH_4	→	c = 1	h = 4
	C_2H_6	→	c = 2	h = 6
	CO	→	c = 1	o = 1
	H_2S	→	h = 2	s = 1
	N_2	→	n = 2	

A fuel gas is a mixture of various compounds, each having a certain volume fraction of vf (Nm^3/Nm^3). The combustion calculations need a summation of all compounds in the mixture, therefore the Σ (sigma) sign appears in the equations.

$$A_{min}=\frac{1}{0.21}\sum vf\times\left(c+\frac{h}{4}+s-\frac{o}{2}\right) \qquad [Nm^3/Nm^3 fuel]$$

$$V_{min}=0.79\times A_{min}+\sum vf\times\left(\frac{n}{2}+c+s+\frac{h}{2}\right)+A_{min}\times H_2O_{air} \qquad [Nm^3/Nm^3 fuel]$$

The above two formulas apply for any type of gaseous compounds (combustible or even inert) and their mixtures.

The terms in the V_{min}-formula mean:

0.79 x A_{min} + Σ vf x (n/2)	→	N_2	⎫
Σ vf x (c)	→	CO_2	⎬ dry gas
Σ vf x (s)	→	SO_2	⎭
Σ vf x (h/2) + A_{min} x H_2O_{air}	→	H_2O	} wet

Example:	90%	CH_4	→	vf = 0.9	c = 1	h = 4
	4%	C_2H_6	→	vf = 0.04	c = 2	h =6
	3%	CO	→	vf = 0.03	c = 1	o = 1
	2%	N_2	→	vf = 0.02	n = 2	
	1%	O_2	→	vf = 0.01	o = 2	

$$A_{min}=\frac{1}{0.21}\left[0.9\times\left(1+\frac{4}{4}\right)+0.04\times\left(2+\frac{6}{4}\right)+0.03\times\left(1-\frac{1}{2}\right)+0.01\times\left(-\frac{2}{2}\right)\right]=9.262 Nm^3/Nm^3$$

$$V_{min}=0.79\times A_{min}+\left[0.9\times\left(1+\frac{4}{2}\right)+0.04\times\left(2+\frac{6}{2}\right)+0.03\times(1)+0.02\times\frac{2}{2}\right]=10.267 Nm^3/Nm^3$$

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Composition of V_{min}

The composition of V_{min} (N_2, CO_2, SO_2, H_2O) can be easily calculated by identification of the terms of the formula of V_{min}.

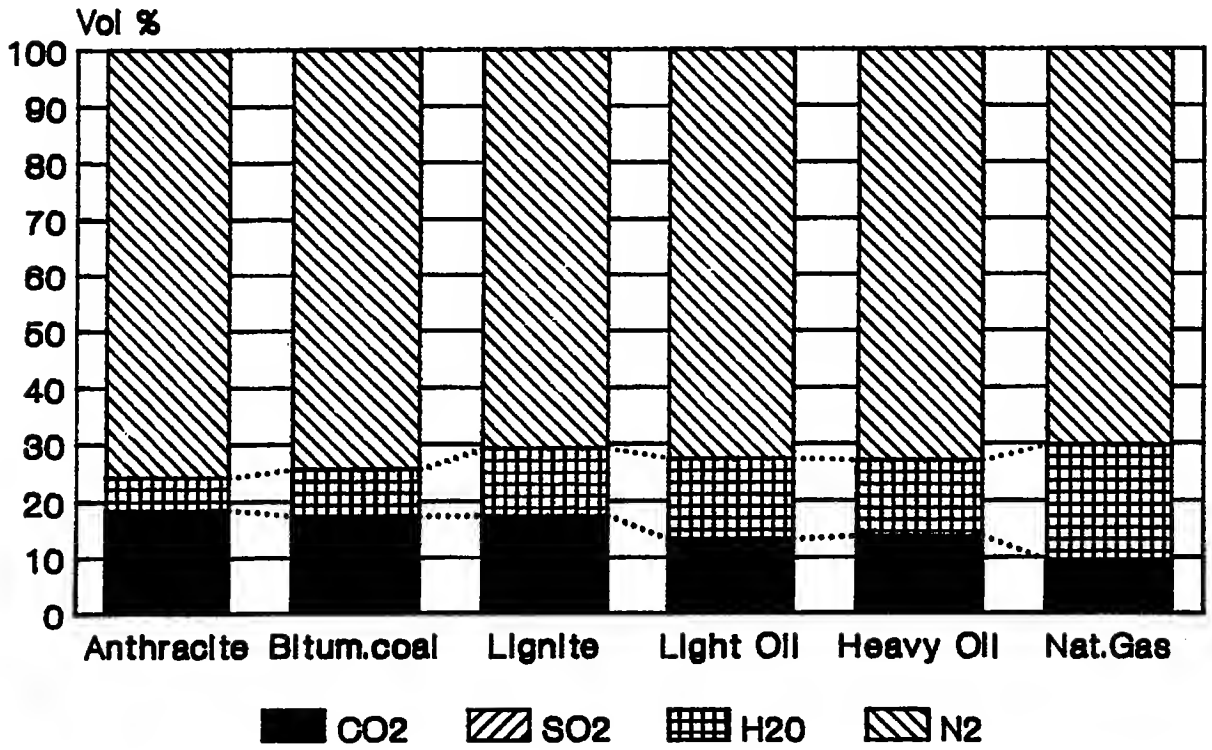
The best way to show this is on the previous example of coal where a V_{min} of 7.494 Nm^3/kg coal was obtained.

Comp.	Calculation Term	Nm^3/kg coal	% in V_{min}
N_2	0.79 x 7.207 + 0.8 x 0.015 (A_{min})	5.706	76.1
CO_2	1.87 x 0.71	1.327	17.7
SO_2	0.7 x 0.01	0.007	0.1
H_2O	11.2 x 0.04 + 1.24 x 0.005	0.454	6.1
Total		7.494	100%

Note: Due to the absorption of SO_2 with cement raw meal the above calculation of SO_2 will not produce the true amount of SO_2 in the gas of a cement kiln.

Fig. 2 shows typical compositions of V_{min} for 6 common types of fuels.

Fig. 2 Typical Compositions of V_{min} (for 6 common fuels)



2.3.2 Approximate Calculations

In the previous paragraph 2.3.1 the exact calculation procedures for A_{min} and V_{min} are shown. In many practical cases it is, however, not necessary to do exact calculations if approximations of sufficient accuracy can be given.

Such approximates have been found by investigating a large number of different types of fuels (see HMC report VA 81/4849/D).

For all fuels of practical use in the cement industry such as coal, fuel oil, natural gas and most of the alternative fuels the following approximations for A_{min} and V_{min} can be used:

Minimum air:	Amin ≈ 0.26 x CV
Minimum combustion gas (wet):	Vmin ≈ 0.28 x CV

where A_{min} resp. V_{min} = [Nm^3] and CV = [MJ]

Or expressed in words:

Each MJ (Megajoule) of combustion energy requires 0.26 Nm^3 minimum air and produces 0.28 Nm^3 minimum combustion gas.

By using the above approximations, a very quick determination of gas quantities is possible.

Note that no analysis of the fuel is required.

Furthermore, the above approximations can also be used if the CV of the fuel is not given expressively. For example, if the specific heat consumption of a cement kiln is 3.5 MJ/kg clinker the specific A_{min} and V_{min} per kg clinker can be calculated directly:

- A_{min} = 3.5 x 0.26 = 0.91 Nm^3/kg cli
- V_{min} = 3.5 x 0.28 = 0.98 Nm^3/kg cli

If the type of fuel is known the actual factors can be taken from the following table in order to obtain somewhat more accurate results.

Type of Fuel		**A_{min}-Factor (Nm^3/MJ)**	**V_{min}-Factor (Nm^3/MJ)**
Anthracite, coke	at 1% moisture	0.26	0.27
Bituminous coal (low to medium volatile)	at 1...2% moisture	0.26	0.28
Lignite (high volatile)	at 10% moisture	0.26	0.29
Wood / Peat	at 10...15% moisture	0.26	0.30
Light oil		0.26	0.29
Heavy oil (bunker oil)		0.26	0.28
Tar		0.26	0.28
Waste tires (rubber)		0.26	0.27
Natural gas (high CH_4 content)		0.27	0.30
Natural gas (approx. 24% C_2H_6)		0.26	0.30

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3. GAS FROM RAW MATERIAL

3.1 Gases from Dry Raw Meal

If cement raw meal is heated up, hydrate water and CO_2 are released.

The true quantity of dry raw meal (not kiln feed) required to produce 1 kg of clinker is called R. Normally R equals to approx. 1.55 kg/kg cli.

R must be calculated from the loss on ignition:

$R = \frac{1}{1 - L.O.I}$	[kg/kg cli]

↑
loss on ignition of raw meal in [kg/kg meal]

The above formula applies if the kiln dust is completely returned into the kiln. If some dust is discarded (without return to kiln) the factor R increases accordingly.

The quantity of hydrate water released from dry raw meal is calculated as follows:

H_2O_{hyd} = R x hydrate content x 1.24	[Nm^3/kg cli]

↑ (kg hydrate water/kg raw meal)

The quantity of CO_2 released from dry raw meal is calculated as follows:

CO_2 = [R x (1 - hydrate content) - 1] x 0.509	[Nm^3/kg cli]

↑ (kg hydrate water /kg raw meal)

Normal dry cement raw meals release approximately the following gas quantities per one kg of clinker:

H_2O_{hyd}	≈	0.01 Nm^3/kg cli
CO_2	≈	0.27 Nm^3/kg cli
Total	≈	0.28 Nm^3/kg cli

In most cases it is sufficient to use the above approximation. If dust is discarded the above value must be increased accordingly.

3.2 Water from Wet Raw Meal or from Raw Slurry

If w (kg/kg) means the moisture or slurry water content of the kiln feed, the amount of water vapor is calculated as:

$H_2O = \frac{w}{1 - w} \times R \times 1.24$	[Nm^3/kg cli]

In case of a dry kiln the above quantity is usually negligible, but it is essential on a wet kiln. At a slurry water content of e.g. 35% it amounts to 1.03 Nm^3/kg clinker.

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4. KILN GAS

4.1 Measurement of Exhaust Gas Composition

Within this section O_2, CO_2, CO and N_2 are considered. Trace gases such as NO_x and SO_2 and hydrocarbons belong to the scope of emission measurements and are usually in the order of less than 0.1% and are therefore not significant with regard to the main components (O_2, CO_2, CO, N_2).

4.1.1 Gas Sampling

Although gas sampling may appear as one of the easiest things, it is in fact the source of more than 50% of all measuring errors! Sampling can be done either continuously or by extracting a gas sample into a rubber bladder.

The aim of sampling is to obtain a representative gas sample from a gas duct into the analyzer. The possible problems and solutions in general are as follows:

Problem	Solution
Gas is not homogeneous within the cross section of the gas duct, because gases do not mix well (sample not representative)	Sample the average of the total cross section (instead of one sampling point). Avoid sampling points where poor upstream mixing seems obvious. Prefer sampling points after fans (mixing effect of fan blades).
False air is entering into gas sampling system(too much O_2 in gas sample)	Check tightness of sampling system, especially when sample is extracted from high underpressure. Avoid small sampling tubing which may become clogged by dust (and produce high suction pressure at low sample gas flow). In case of spot sampling with rubber bladder: rinse bladder at least once with sample gas.
O_2 may react with steel tube at > 400°C (too little O_2 in sample)	Use hot extraction tubes made of sicromal, ceramics or quartz.
CO_2 may be absorbed by dust and condensate (too little CO_2 in sample)	Use a filter directly at the hot extraction point. Keep sampling system free of dust and condensate(e.g. rubber bladder must be clean inside). To a certain extent reliable sampling for CO_2 will always be a problem as long as gas cooling is applied.
Long storage time in rubber bladder (too little CO_2 and too much O_2 due to diffusion)	Analyze rubber bladder after 30 min at the latest.

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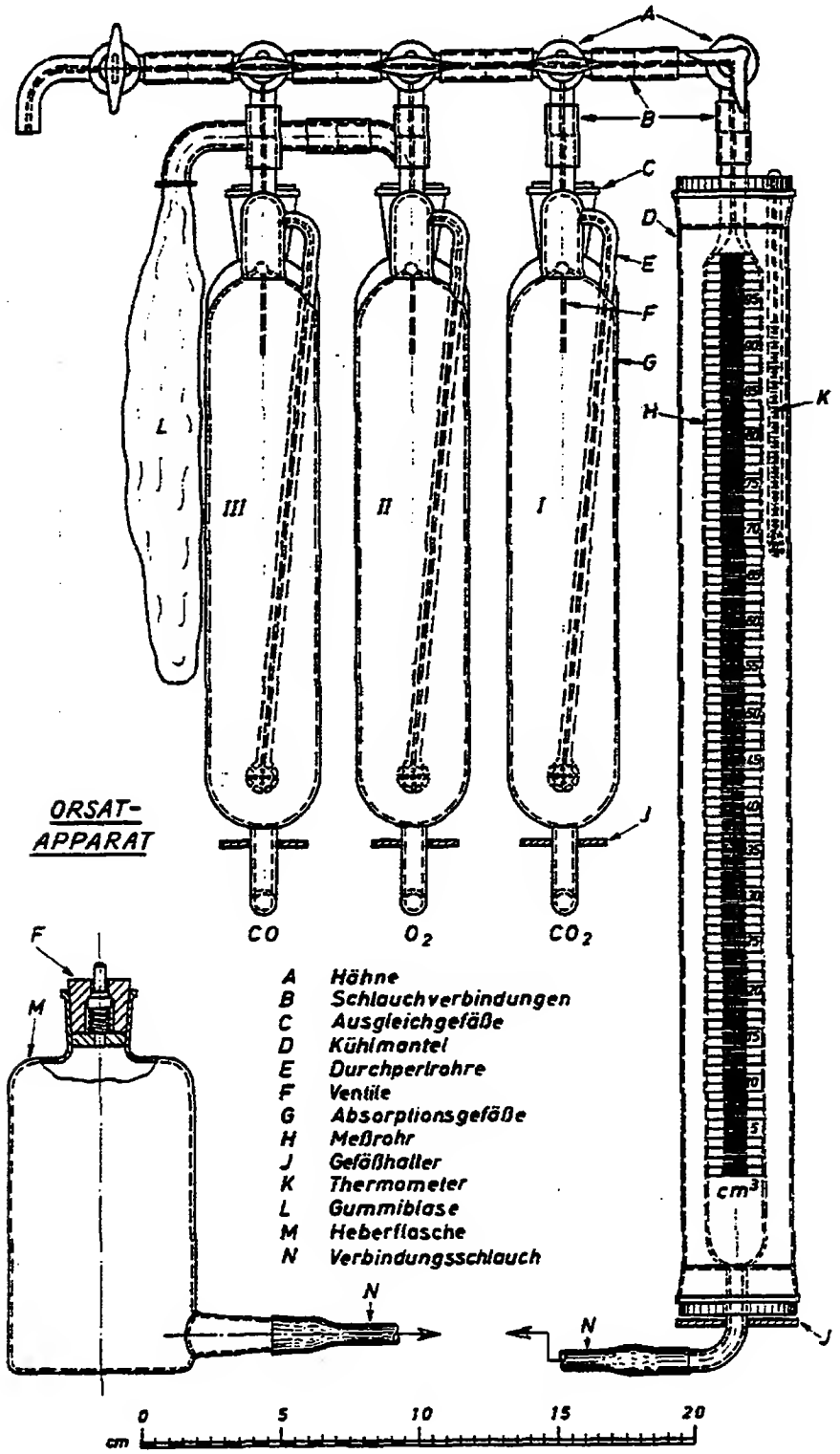
4.1.2 The Orsat Apparatus

An Orsat apparatus is shown in Fig. 3. A gas sample is filled into a burette volume of 100%. Step by step, CO_2, O_2 and CO are absorbed by contacting the gas sample several times with the absorbing agent. The absorbed gas volume is measured after each step in the burette.

Fig. 3 The Orsat Apparatus



It is important to maintain the following, correct sequence of absorption:

1) CO_2 → KOH solution
2) O_2 → pyrogallic acid / KOH
3) CO → $Cu_2 Cl_2$ solution

In order to avoid measuring errors the following hints are useful:

1) Check tightness of Orsat apparatus first.
2) Use fresh absorption liquids.
3) While taking a reading always hold liquid level in burette and expansion tank at same height (constant pressure).
4) Check O_2 absorption by measuring ambient air (21% O_2).

- Advantages of the Orsat:
  - Measuring principle is very clear and simple
  - The Orsat is available (or at least known) in every cement factory
- Disadvantages of the Orsat:
  - Due to is principle a continuous measurement is impossible
  - Analyzing is time consuming
  - Although the Orsat can measure CO it is impossible to detect small traces of CO ≤ 0.1%. Therefore the CO result is only useful in cases of „bad" combustion producing about 0.3 to 1% CO.

4.1.3 Other Gas Analyzers

In most cases continuous gas analyzers which are permanently installed are used for process control in a cement kiln (see chapter Exhaust Gas Analysis). Such permanent analyzers measure only at one location and are not flexible enough for an investigation of the complete kiln system (involving a large number of measuring points).

More and more portable gas analyzers are available on the market. They can usually measure O_2, CO and combustibles continuously. At fairly low costs (less than $4'000) a quite handy and useful equipment can be purchased. For an investigation on a cement kiln this is virtually the ideal tool to perform extensive and quick measurements at various locations in the plant.

In the following a few important measuring principles are compared.

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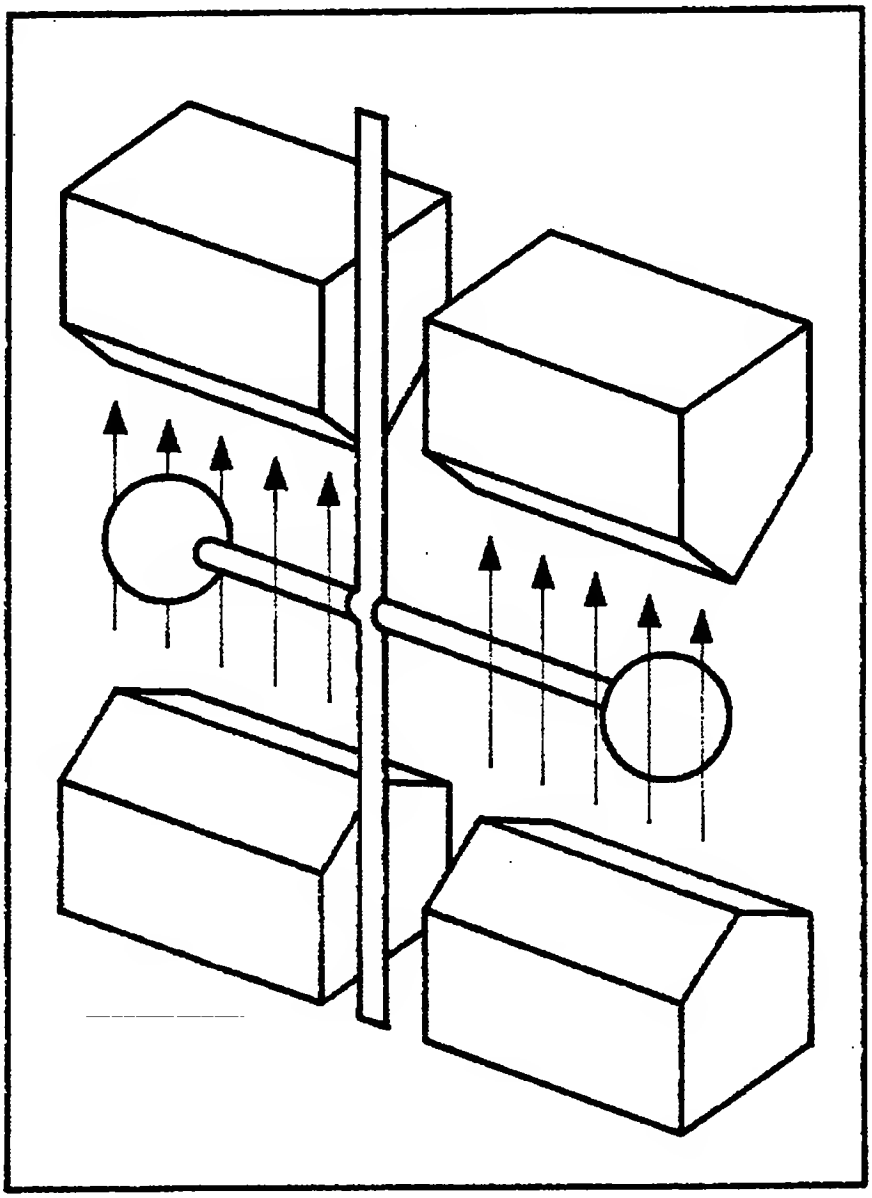
Frequent Measuring Principles for O_2

- Paramagnetic effect of O_2
  - Either thermomagnetical or magneto-mechanical principle is used.
  - Thermomagnetical principle is cross-sensitive to gases having other heat conductivities than O_2 e.g. CO_2.
  - Magneto-mechanical principle with cell containing movable „dumb-bell" is not cross-sensitive to CO_2 (see Fig. 4A).
  - Analyzers based on the paramagnetic effect are the most common type of permanent installation in the cement industry.

Fig. 4A Frequent Measuring Principles for Gases - Paramagnetic O_2 Sensor



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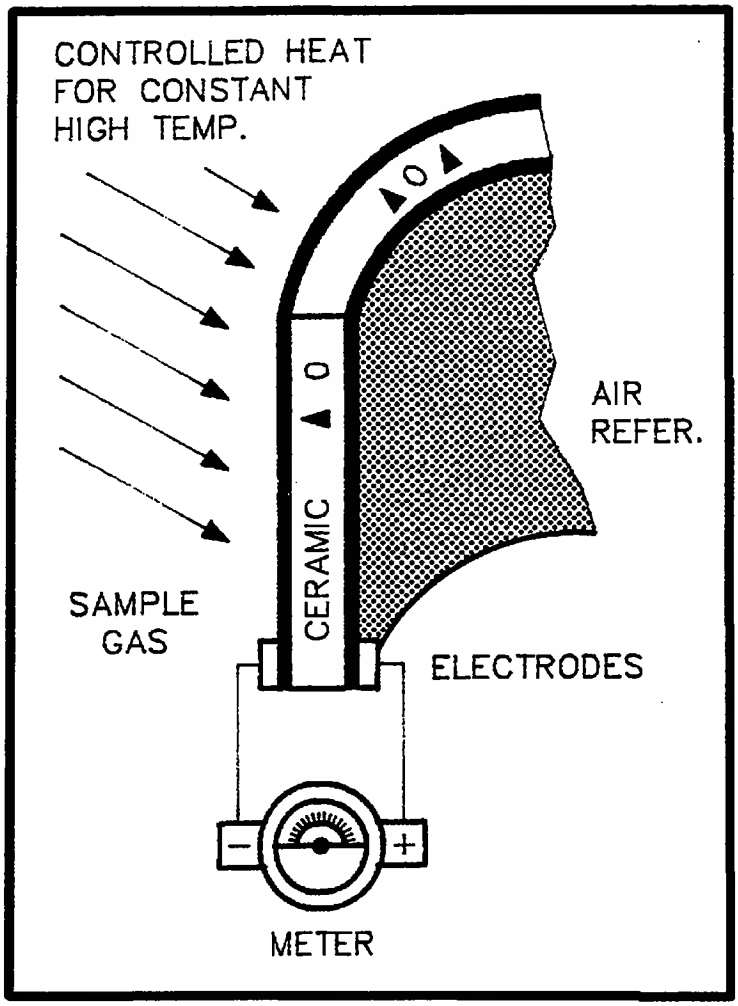
- Electrochemical voltage effect of hot zirconium oxide (Fig. 4B)
  - Working temperatures above 600°C.
  - Measuring the voltage of an electrochemical cell, according to the equation:

$$E = \frac{RT}{nF} \times \ln\left(\frac{PO_2}{PO_2 \mathrm{Ref}}\right) \quad [V]$$

(R = Gas constant, T = [K], n = 4, F = Faraday constant, PO_2 = partial pressure of oxygen)

  - Logarithmic output signal, depending on PO_2.
  - Principle is seldom used.

Fig. 4B **Frequent Measuring Principles of Gases - Zirconiumoxide Cell**



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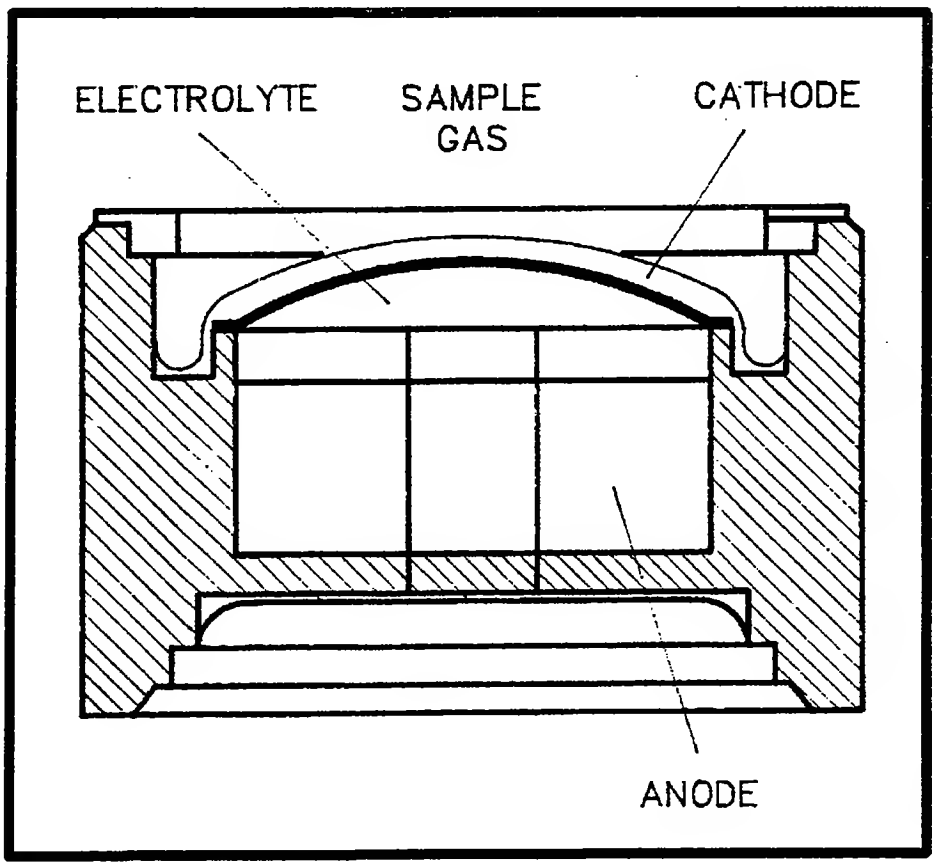
- „Self consuming" electromechanical sensors (Fig. 4C)
  - Basic design of such a measuring cell is similar to a normal battery. Cell consists of anode, electrolyte and air cathode (see Fig. 4C).
  - O_2 enters via diffusion barrier to cathode.
  - At the air-cathode O_2 is reduced to OH. Thereby an electrical current is produced which is proportional to the O_2 content and is used as output signal.
  - Above reaction causes an irreversible consumption of the anode (lead). Therefore lifetime of the cell is limited.
  - Practical lifetime of such a cell is about 1 year (even if cell is not in use a certain aging occurs).
  - Reliable measurements without major cross-sensitivities.
  - Calibration only with air (zero is self adjusting).
  - Application: Portable low cost analyzers.
  - Similar applications: Other electrochemical sensors have been developed for CO, SO_2 and NO_x working on similar principles. By special design and compensation the effect of „self-consumption" could be avoided. These analyzers are widely used for portable applications but not for permanent emission control.

Fig. 4C **Frequent Measuring Principles for Gases - Electrochemical Cell**



4.1.4 Reference to Dry or Wet

The most frequent type of gas sampling includes a cooling of the gas to approx. 0°C (or at least to room temperature). By this cooling the water vapor is condensed and the measurement in the gas analyzer refers to dry gas.

The reference to dry gas will be considered as „normal case" in this paper (unless otherwise noted).

The „classic analyzer" - the Orsat apparatus - refers always to dry gas. For this particular analyzer even some residual water content in the gas sample will not affect the final result, the reading means exactly the dry composition (in this case it would even be wrong to make any correction for residual water vapor in the gas sample!).

Note that there are exceptional cases e.g. where gas samples are not condensed but introduced directly in a hot zirconium oxide cell. In such a case the result will refer to wet gas. Furthermore sampling systems without gas coolers but with dilution of the hot gas instead (thus avoiding any water condensation) exist, but they are hardly used for O_2, CO, CO_2.

Summarizing, sampling with gas cooling and reference to dry gas is considered as normal, i.e. is used for about 99% of all practical measurements.

4.1.5 Determination of Water Content

Since all measuring systems described above are not able to measure water, other methods have to be used if water really has to be measured. Important methods are:

- Two temperatures method:
  - Measuring of wet bulb and dry bulb temperature.
  - If the gas temperature is < 150°C wet and dry bulb temperature can be measured directly in the gas duct. If the gas temperature is > 150°C an extraction of the gas and cooling down to < 150°C is required.
  - Evaluation of gas moisture according to calculation sheet 32599e (see measuring techniques part of cement course).
  - Method is preferred due to its simplicity, however accuracy at high moisture contents may not be always sufficient.
- Condensation method:
  - A gas sample is extracted for a certain time and cooled by ice water, thus water is condensed.
  - The remaining dry gas extracted is measured by a volume counter.
  - By measuring the weight of the condensate the moisture content of the gas can be calculated.

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4.2 Immediate Calculations from Gas Composition

4.2.1 Dew Point

From the water content the dew point can be calculated.

- Vf [Nm^3/Nm^3_w] is the volume fraction of water vapor in the wet gas
- P_{tot} [bar] is the total pressure (usually assumed as 1.0133 bar)

By expressing the vapor pressure in form of a numerical equation the following dew point formula was developed for precise calculations:

$$\tau = \frac{336.48}{5.3362 - \sqrt{17.045 + \ln(Vf \times P_{tot})}} - 179 \quad [C]$$

Example:

Vf = 0.10 Nm^3/Nm^3_w P_{tot} = 1.0133 bar

$$\tau = \frac{336.48}{5.3362 - \sqrt{17.045 + \ln(Vf \times P_{tot})}} - 179$$

$$\tau = 46.09^{\circ}C$$

The corresponding inverse function giving the saturation pressure of water (PW) is as follows:

$$PW = EXP\left(11.4297 - \frac{3591}{t+179} + \frac{113216}{(t+179)^2}\right) \quad [bar]$$

t = 0...120°C, max. deviation ± 0.03%.

A section of the above function is shown in Fig. 5 which can be used for graphical evaluation of the dew point.

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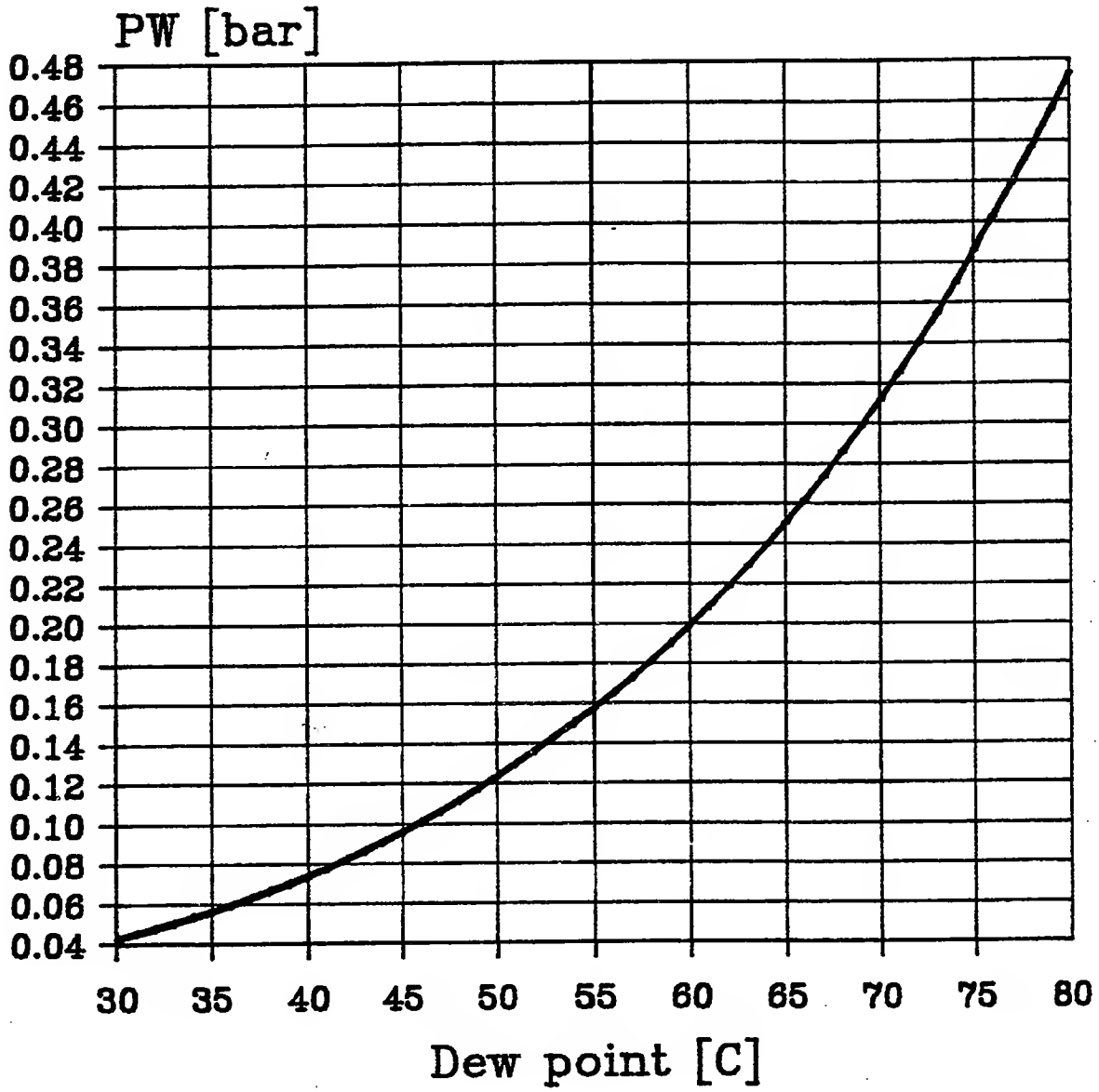
Fig. 5 Determination of Dew Point



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4.2.2 Air Factor n

The air factor n can be calculated precisely from the gas composition by considering the N_2, according to:

$$n = \frac{N_2 eff}{N_2 \min} = \frac{N_2 eff}{N_2 eff - N_2 excess}$$

By using the O_2 and N_2 content of the gas the following important formula can be derived:

$$n = \frac{N_2}{N_2 - \frac{0.79}{0.21} \times O_2} = \frac{1}{1 - 3.76 \frac{O_2}{N_2}}$$

If CO is present the following, general formula applies:

$$n = \frac{1}{1 - 3.76 \frac{O_2 - 0.5CO}{N_2}}$$

Note that e.g. the CO_2 from raw meal will not affect the n although it does affect the gas composition (the ratio O_2/N_2 is constant when CO_2 is added to the gas).

Therefore this formula is very useful and applicable for any case. On the other hand O_2, CO and N_2 are required as inputs which needs a complete Orsat analysis (N_2 is the rest composition after absorption of O_2, CO and CO_2).

Example:

O_2 = 4% N_2 = 68% (CO = O)

$$n = \frac{1}{1 - 3.76 \times \frac{4}{68}} = \underline{\underline{1.28}}$$

4.2.3 Combustion Efficiency

Unburned gases are a sign of incomplete combustion. The most important unburned gas is CO, whereas hydrocarbons are usually < 0.1% on a cement kiln and can be neglected with regard to energy losses.

If only CO is known the loss due to incomplete combustion is:

Q = (dry gas quantity) x CO x 12.6	[MJ]

↑[Nm^3] ↑[Nm^3/Nm^3]

Example:

On a cement kiln the fuel combustion is 3.3 MJ/kg clinker, the dry exhaust gas quantity is 1.3 Nm^3/kg clinker and the CO content = 0.2%.

q = 1.3 x 0.002 x 12.6 = 0.033 MJ/kg cli

$$\text{Relative loss} = \frac{0.033 MJ}{3.3 MJ} = 0.01 = \underline{\underline{1\%}}$$

Some gas analyzers have also a combustible sensor based on a catalytic combustion of the sample gas. Such a sensor responses to all types of combustible gases (CO, H_2, CH_4, C_2H_6 etc.). Although the main combustible component is not CH_4 (methane) but rather CO it is common to express the result as „CH_4 equivalent".

If the total of combustibles is given as their „CH_4 equivalent" the loss is:

Q =	(dry gas quantity) x CH4 x 35.8	[MJ]
	↑[Nm^3] ↑[Nm^3/Nm^3]	

4.2.4 Gas Density

The gas density at normal conditions can be calculated by a simple mix calculation by using the following densities:

Gas	M [kg/kmol]	Density* [kg/Nm^3]
O_2	32.0	1.429
CO_2	44.0	1.964
CO	28.0	1.250
„N_2"**	28.15	1.257
H_2O	18.0	0.804

* Density in a gas mixture (considered) as ideal gas

** Not as pure N_2 but as „rest" in the air (including also Ar and trace gases)

Note: For the calculation of the true gas density all concentrations (O_2, CO_2, CO, N_2, H_2O) are to be referred to wet gas here.

Example:

Exhaust gas density of a suspension preheater kiln with coal firing:

Gas component	Composition of dry gas [Vol%]	Composition of wet gas [Vol%]	
O_2	4%	0.9 x 4 = 3.6%	}
CO_2	31%	0.9 x 31 = 27.9%	} 100 - 10 = 90%
CO	0%		}
N_2	65%	0.9 x 65 = 58.5%	}
H_2O	--	10 %	
Total	100%	100 %	

Density = 0.036 x 1.429 + 0.279 x 1.964
+ 0.585 x 1.257 + 0.10 x 0.804 = 1.415 kg/Nm^3
(at normal conditions)

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4.3 Calculation of Exhaust Gas Quantities

4.3.1 Firing / Hot Gas Generator

The following calculation applies for any type of firing where gases are coming only from fuel combustion and excess air (no gases from raw meal).

Problem: The fuel consumption Q = [MJ/h] of the firing is known and the complete exhaust gas analysis (CO_2, O_2, CO, N_2) is given. By means of this information the gas flow [Nm^3/h] shall be calculated.

Solution: Calculate first the n according to 4.2.2:

$$n = \frac{1}{1 - 3.76 \frac{O_2 - 0.5CO}{N_2}}$$

The gas flow V [Nm^3/h] is then:

V = Q [V_{min} + (n - 1) A_{min}] [Nm^3/h]

V_{min} and A_{min} are in [Nm^3/MJ] and can either be calculated exactly according to 2.3.1 or just simply by introducing the approximations according to 2.3.2, namely:

- $V_{min} \approx 0.28$ Nm^3/MJ
- $A_{min} \approx 0.26$ Nm^3/MJ

If the complete gas analysis is not known but only the O_2 concentration (dry) a further approximation for the dry amount of Vmin, namely 0.25 Nm^3/MJ, has to be introduced which results in the following approximation formula:

$$V = Q \times \left[0.28 + 0.25 \frac{\%O_2}{21 - \%O_2} \right]$$

[Nm^3/h]

↑ [MJ/h] O_2 ref. to dry

Note: If the O_2 was based on wet gas the above formula would become even more simple, namely:

$$V \approx Q \times 0.28 \times \frac{21}{21 - \%O_2}$$

4.3.2 Cement Kiln

On a cement kiln it is common practice to work with specific quantities referred to 1 kg of clinker, therefore the fuel consumption is expressed in q [MJ/kg cli] and V means here [Nm^3/kg cli].

The general calculation formula is:

V = q (Vmin + (n -1) x Amin) + (gas from RM)	[Nm^3/kg cli]

↑ [MJ/kg cli] ↑ [Nm^3/kg cli]

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n is calculated from the complete gas composition according to 4.2.2.

If only O_2 concentrations are available the following approximations can be used for dry kilns:

- from combustion	0.28 * q	[Nm^3/kg cli]
- from raw meal	0.28	[Nm^3/kg cli]
- excess air plus false air	$(0.28+0.25\times q)\times\frac{\%O_2}{21-\%O_2}$	[Nm^3/kg cli]

Total = V = 0.28 (q+1) + (0.28+0.25xq) x $\frac{\%O_2}{21-\%O_2}$	[Nm^3/kg cli]

↑

[MJ/kg cli]

In case of wet raw meal or for wet kilns additional water has to be added (see 3.2) to the above result.

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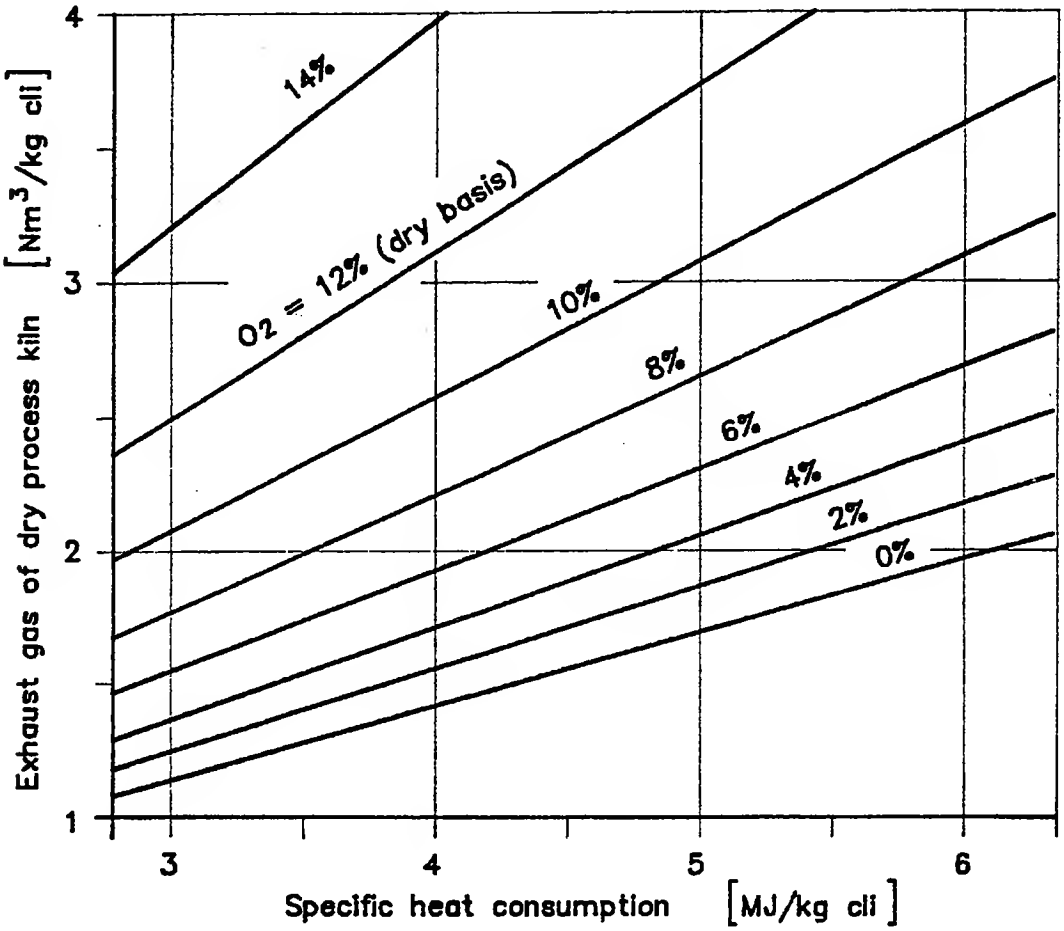
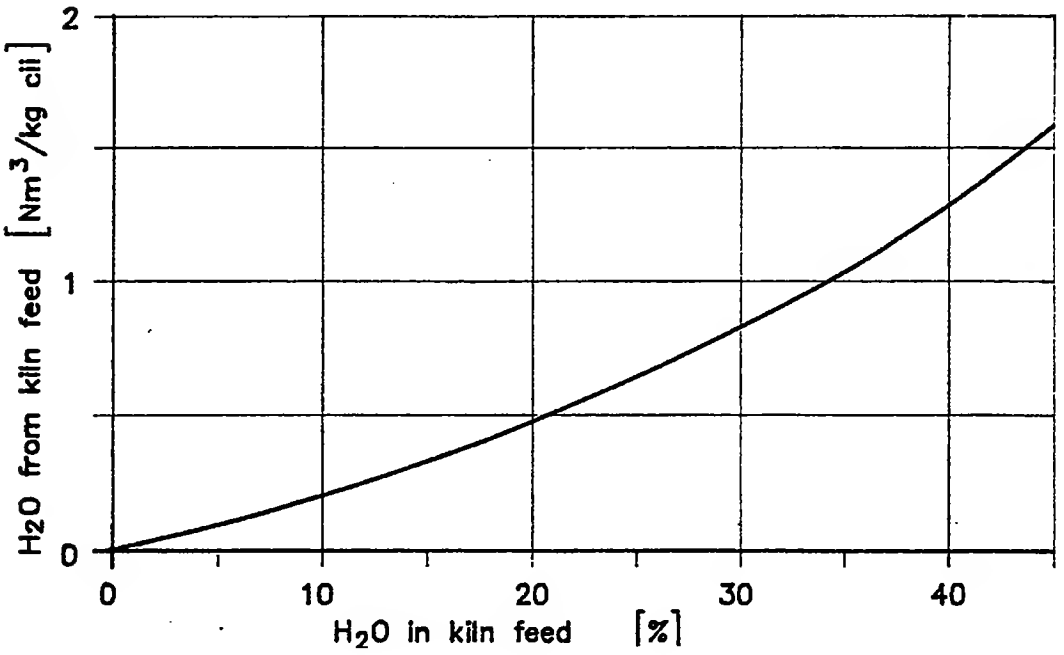
The above approximation can be used in form of diagram (Fig. 6)

Fig. 6 Quick Determination of Kiln Exhaust Gas Quantity





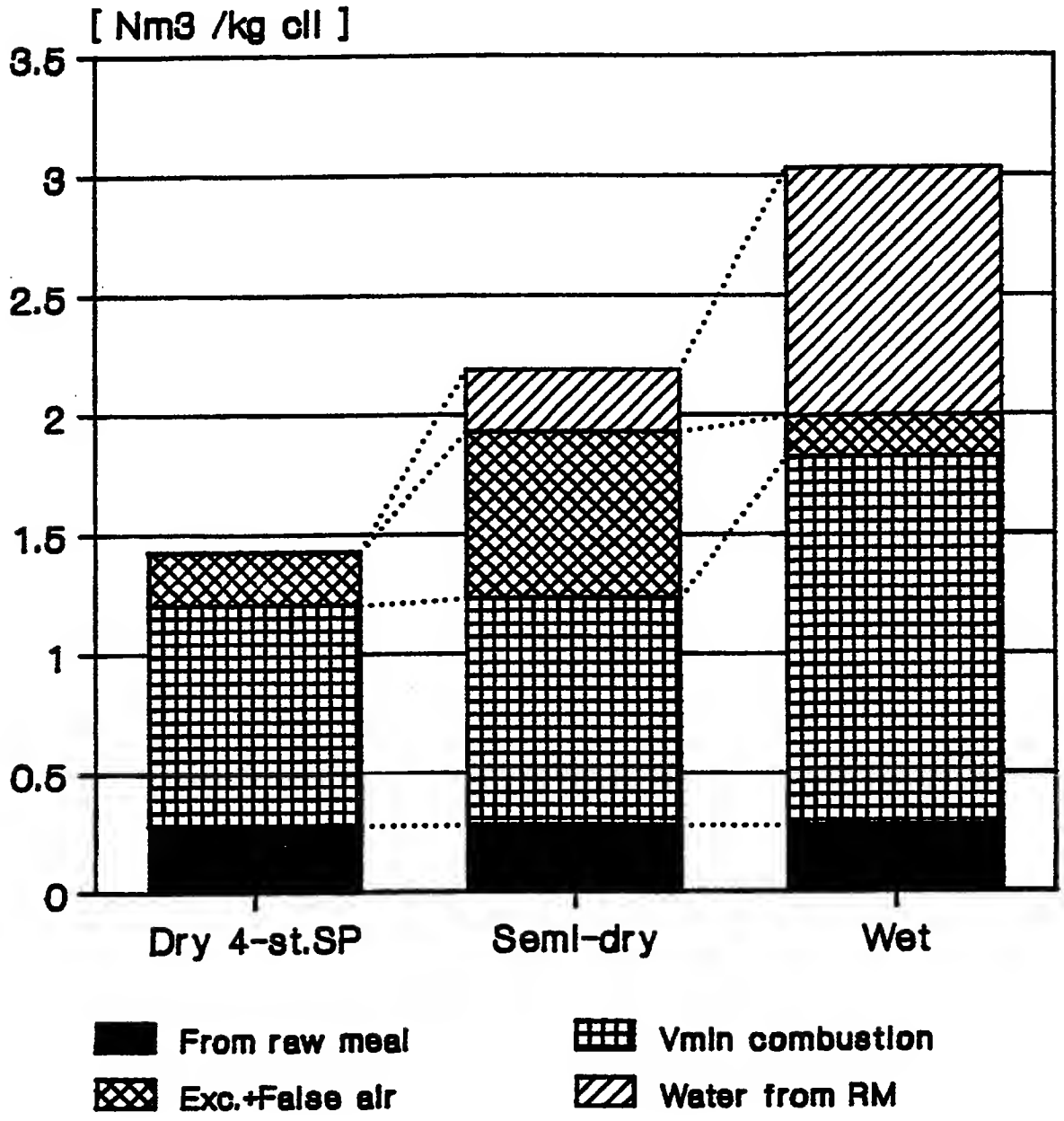
Example:

Dry kiln, q = 3.3 MJ/kg cli, O_2 = 3.5%

$$\text{Total gas} = 0.28\ (3.3+1) + (0.28+0.25\text{x}3.3) \times \frac{3.5}{21-3.5} = \underline{1.425\ Nm^3/kg\ cli}$$

Three typical results for cement kilns are shown in Fig. 7.

Fig. 7 Typical Exhaust Gas Quantities for Three Cement Kiln Systems



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4.4 Calculation of the CO_2 Content of Cement Kiln Exhaust Gas

4.4.1 Introduction

If no measured CO_2 concentration is available it is possible to complete the gas analysis with a calculated CO_2 concentration. This can be useful e.g. for calculation of density or specific heat.

It is possible to calculate the CO_2 content at any point in the exhaust gas system (suspension preheater, cooling tower, raw mill, filter) of a cement kiln if the corresponding O_2 concentration is known.

The calculation is based on two main facts:

1) The CO_2 balance applies (CO_2 comes from combustion and from raw meal)
2) Starting from the O_2 concentration the corresponding dilution with air can be quantified.

In addition a few other preconditions are to be observed here:

- Reference is made to dry gas composition.
- The assumption is made that all CO_2 from raw meal is in the kiln gas (i.e. after complete calcination).
- No gas is lost or extracted from the gas stream under consideration (e.g.: an upstream extraction of a bypass gas is excluded here).
- For numerical calculations the following approximations are used:
  - CO_2 from raw meal $\approx$ 0.27 Nm^3/kg cli
  - N_2 from combustion at n = 1
    N_2 = $0.79 \times A_{min}$ + N_2 from fuel
    = $0.79 \times 0.26 + 0.0015 \approx$ 0.207 Nm^3/MJ
    (this approximation is very accurate for all types of fuels)

4.4.2 Maximum CO_2 Content

In a first step the influence of dilution (or excess air) is assumed as zero (0% O_2). Therefore a theoretical, maximum CO_2 content can be calculated from the CO_2 balance.

The dry gas contains only CO_2 and N_2 which can be calculated as follows:

CO_2	$= 0.27 + c_f \times q$	[Nm^3/kg cli]
N_2	$= 0.207 \times q$	[Nm^3/kg cli]

q [MJ/kg cli] = specific heat consumption

c_f [Nm^3/MJ] = CO_2 from fuel, referred to CV

c_f depends on the type of fuel and can be calculated according to paragraph 2.3.1.2 (by considering only the CO_2 term in the V_{min} formula).

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Typical values for c_f are:

Coal	=	0.050 Nm^3 CO_2/MJ
Fuel oil	=	0.039 Nm^3 CO_2/MJ
Natural gas	=	0.028 Nm^3 CO_2/MJ

The concentration CO_{2max} is calculated from the above quantities of CO_2 and N_2:

$$CO_{2max} = \frac{CO_2}{CO_2 + N_2} \quad [Nm^3/Nm^3]$$

4.4.3 True CO_2 Content

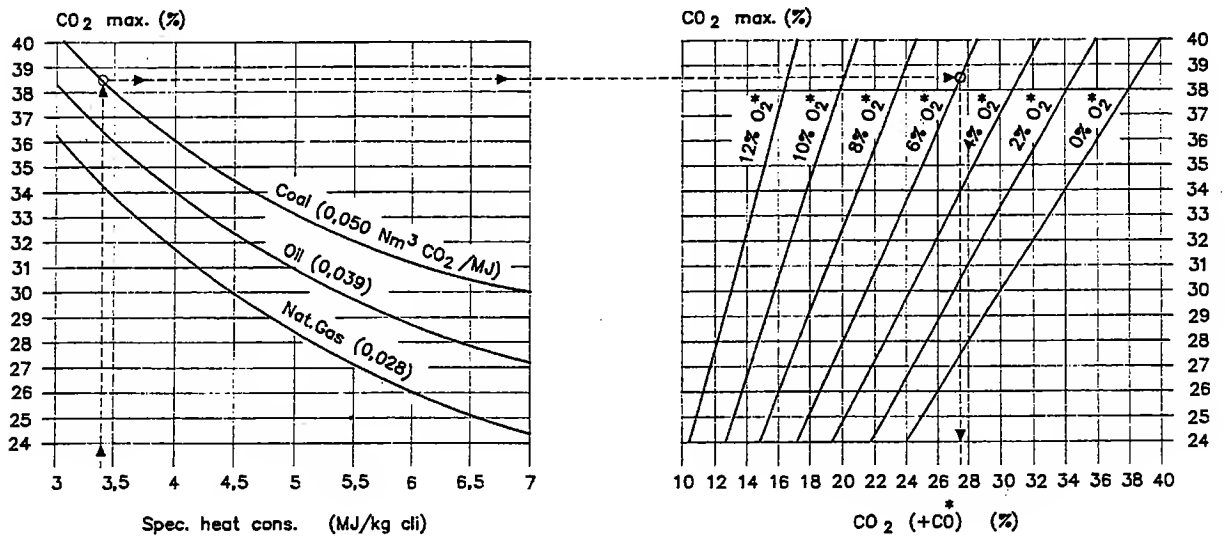
Due to dilution of the gas with air and due to possible formation of some CO the true CO_2 content is:

$$CO_2 = CO_{2max}\left(1 - \frac{O_2}{0.21} + \frac{CO\,0.79}{2\ \ 0.21}\right) - CO \quad [Nm^3/Nm^3]$$

CO_2, O_2, CO = $[Nm^3/Nm^3]$

Fig. 8 shows a graphical mode of evaluation of a gas composition based on the formulas given in this paragraph.

Fig. 8 Relation Heat Consumption/Exhaust Gas Composition for Cement Kilns (CO_2 = 0.27 Nm^3/kg cli)



* Note : If CO is present set parameter $O_2 = O_2\,eff - 0{,}4 * CO$

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Notes:

1) Although it is theoretically possible to calculate also the specific heat consumption from a given gas composition (backward calculation) this procedure is not recommended. In most of the practical cases this will be quite inaccurate because of the lack of a precise gas composition (CO_2 readings are often too low due to systematic sampling error).
2) It is not recommended to introduce calculated CO_2 values in the n-formula according to 4.2.2, because „calculation with calculated values" may finally end in a somewhat doubtful result. The air factor n should therefore only be calculated from a measured gas composition (or alternative calculations based on O_2 only should be used).

5. FALSE AIR INVESTIGATIONS

5.1 Introduction

Underpressures at various locations in the gas system may vary from 0 to -100 mbar. Leakages of the equipment can therefore cause considerable amounts of false air which increase the gas flow.

In addition to the obvious increase of gas flow, false air can also be detrimental for the heat consumption.

If the exhaust gas fan (or the mill fan) is running at full capacity any additional false air causes a reduction of the kiln production capacity. Shortage of production may become an essential financial loss for the cement factory. Therefore measures against false air must be taken.

The first step is always to locate the major leakage points. For this purpose it is necessary to measure a complete O_2 profile of the kiln system. This may e.g. include:

- suspension preheater
- cooling tower
- kiln fan
- raw mill
- E.P.
- mill fan

As already mentioned in 4.1.3 such a profile can be measured easily with a portable O_2 analyzer. A large number of sampling points and repeated checks can be realized.

Proper sampling is essential for such an investigation! During such a measurement the operating conditions of the system should be constant in order to obtain a consistent O_2 profile. Proceed quickly from one sampling point to the next and finally re-check all O_2 concentrations if they are really stable.

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5.2 Evaluation

The measured O_2 profile may already give a qualitative impression of the tightness of the gas system. For a quantitative information in form of real gas flows (Nm^3/h) the approximation formula from 4.3.2 is used here (because it is based only on O_2).

If m denotes the clinker production in [kg/h] the complete formula is:

$$V = m\,[0.28 \times (q + 1) + (0.25 \times q + 0.28)\,\frac{\%O_2}{21 - \%O_2}\,] \quad [Nm^3/h]$$

↑ [kg/h] ↑ [MJ/kg cli]

V is the gas flow at the measuring point. The amount of false air is easily calculated as the increase of V between two measuring points.

Note:

V does not include water evaporation e.g. from raw slurry or from a cooling tower. It is also obvious from the formula that V will not be influenced by any water evaporation because O_2 is based on dry gas composition.

For the purpose of false air calculation it is not required to add these additional water quantities. It would be even wrong to add such quantities to V before calculating the false air as difference between two V-valves.

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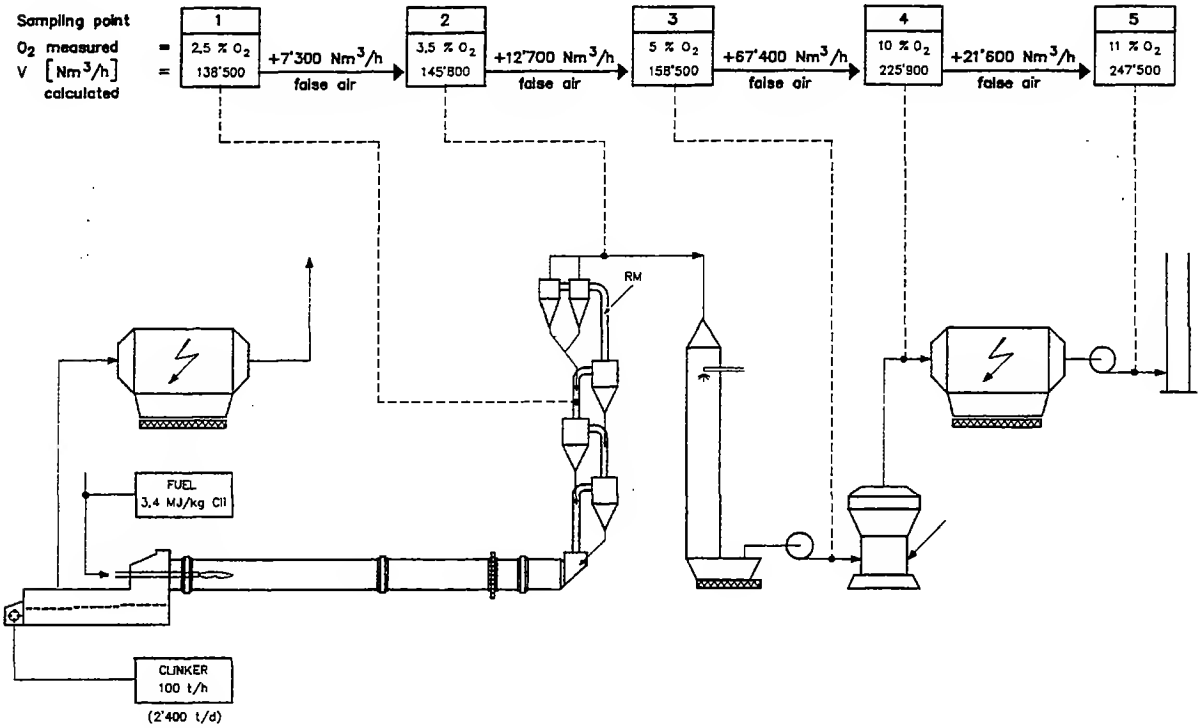
5.3 Example of an Investigation

On a dry suspension preheater kiln with cooling tower and roller mill in combined operation the O_2 profile was checked by means of 5 sampling points.

The clinker production is 100 t/h with a specific consumption of 3400 kJ/kg cli.

Fig. 9 shows the evaluation of the results by calculating the V [Nm^3/h] for every sampling point.

Fig. 9 False Air Investigation on a Kiln System



V equals to the true wet exhaust gas quantity at the sampling points 1 and 2. For the points 3, 4 and 5 the amount of water vapor from cooling tower and the amount of water evaporated in the raw mill is not included in V.

The false air flows are simply calculated as the increase of V.

The final result gives the amounts of false air inleak (Nm^3/h) for each section. These values are needed in order to evaluate the possible potential for optimization.

Conclusion:

For the example according to Fig. 9 the exhaust gas quantity after preheater is 145'800 Nm^3/h which is finally mixed with up to 101'700 Nm^3/h or 70% of false air when it reaches the chimney. Therefore clear optimization potentials exist in the area of the cooling tower, the raw mill and the electrofilter.

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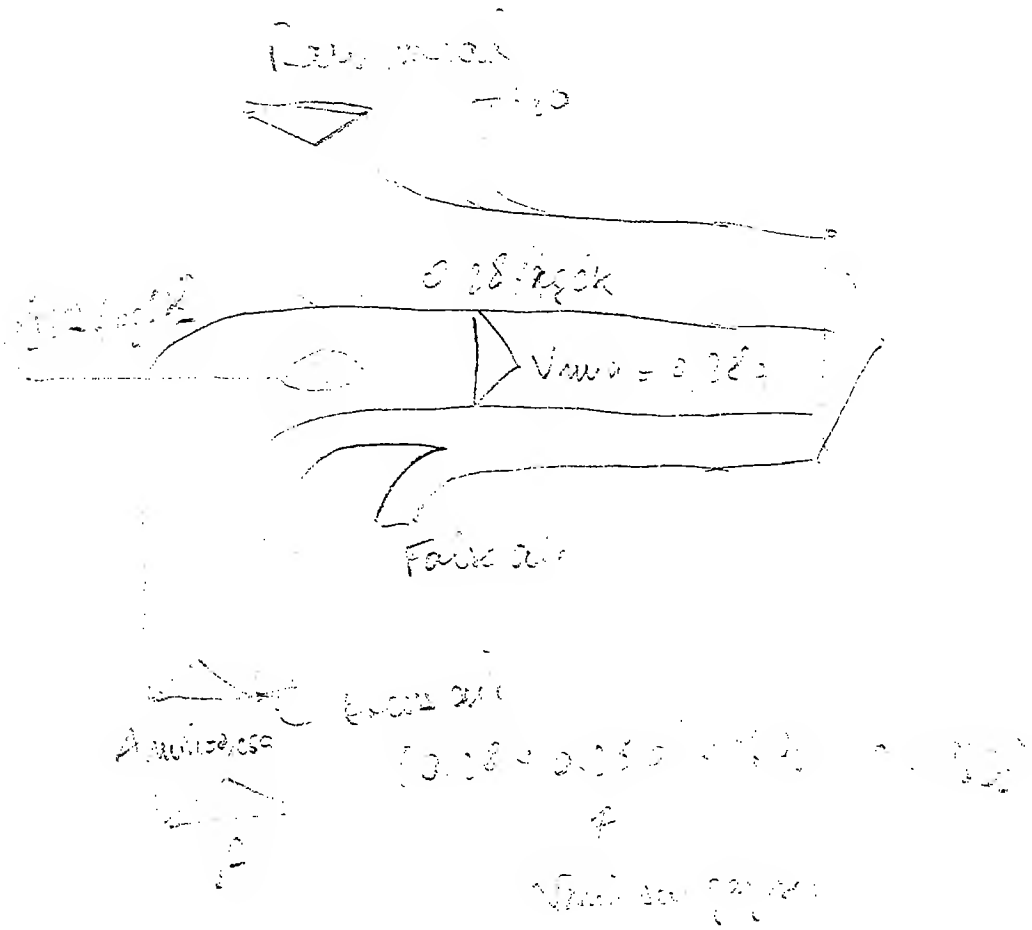

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Chapter 7

Alternative Fuels

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Use of Alternative Fuels

A. Obrist
PT 96/14024/E (Modification 2000)

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SUMMARY

The use of alternative fuels (AF) in cement kilns can save costs and contribute to the solution of environmental problems. The paper on hand concentrates on technical and environmental aspects.

Rules on how to use alternative fuels and possible impacts are given. Practical examples are attached (flowsheets).

1. INTRODUCTION

- Burning of alternative fuels (AF) in cement kilns offers unique advantages from an environmental point of view (high temperatures, long retention, no solid residues, no increase of emission, overall reduction of CO_2 emission).
- Using alternative fuels saves costs. Two main factors contribute to this
  1) Thermal substitution rate (there are technical limits)
  2) Low or even negative energy price (USD per GJ) for AF's
- Logically only fuels that are significantly cheaper than conventional fuels (USD per GJ) can create cost savings. However, even if AF's are cheaper all the additional costs involved have to be considered to make it profitable (preparation, additional production costs, maintenance, reduction of OEE, etc.)
- Within the Holderbank, Group 52 plants are using significant amounts of AF. The average thermal substitution rate of all 105 plants is 12.3% (1998).
  The fuel cost substitution rate (which is not the same as thermal substitution rate) is not yet being reported and the difficulty is to get an objective and fair consideration of all additional costs involved. From the basic principle cost substitution rates of over 100% are possible at negative fuel prices, but so far very exceptional.

2. TYPES OF ALTERNATIVE FUELS

By definition, fuels, which are not traded in the normal fuel market, are considered as "alternative fuels". Petcoke e.g. is not classified as "alternative fuel" and is listed in a separate application list (not treated here).

Alternative fuels can be roughly divided into solid and liquid fuels (gaseous is negligible).

Whether it is simple or difficult to use an alternative fuel depends much on its physical properties. E.g. it may be very simple to use waste oil which has been purified by the supplier. On the other hand it is impossible to use e.g. raw domestic refuse directly as solid fuel, because it is of poor quality and very inhomogeneous. The only practical way to burn it in a cement kiln is a sophisticated pre-treatment to produce RDF (refuse derived fuel).

Table (1) shows a list of alternative fuels in the order of their CV compared to conventional fuels. The calorific value alone does not directly indicate the potential to save costs. E.g. waste tires are as good as coal from the viewpoint of CV but require expensive handling and tend to cause negative impacts on the kiln process, so an adequate compensation must be included in the price (disposal fee).

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Table 1 **Various Alternative and Conventional Fuels, grouped according to their CV**

(*= conventional fuel)

	Material	CV [MJ/kg] net
	Pure polyethylene [3]	46
*	Light oil	42
*	Heavy oil	40
	Tar (by-product)	38
	Pure rubber (without inert material)	36
*	Anthracite	34
	Aluminium metal [1]	31
	Waste oils, various refinery wastes	30 to 40
*	Petcoke	33
	Waste tires	28 to 32
*	Bituminous coal (low ash)	29
*	Bituminous coal (high ash)	24
	Liquid mix (CSS from SCORIBEL or SYNFUEL from Safety Kleen)	20 to 30
	Landfill gas	16 to 20 (MJ/Nm^3)
	Acid sludge, acid tar (from oil reprocessing)	16 to 22
*	Lignite (10% moisture)	16 to 21
	Pot liners (from aluminium smelter)	20
	PVC [3]	19
	Palm nut shells (10% moisture)	19
	Pressed olive cake	18
	Dried wood, bark, saw dust (10% moisture)	16
	Rice husks (10% moisture)	16
	Car shredder wastes	15
	RDF (from domestic refuse, 10% moisture)	15
	Animal meal	15
	Cardboard, paper (air dry)	15
	Impregnated saw dust (25% moisture)	10 to 12
	Dried sewage sludge (10% moisture)	10
	Fuller's earth (from oil purification, LD actual)	10
	Domestic refuse (30% moisture)	8.5
	Dried sewage sludge (30% moisture)	7.5
	Pure iron [2]	7.5

1) Al metal may occur e.g. in composite packaging wastes and is oxidised to Al_2O_3

2) Fe metal occurs e.g. in waste tires and is oxidised to Fe_2O_3

3) Usually not in pure form, but contained in mixed plastics

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3. UTILIZATION IN CEMENT KILNS

3.1 List of Applications

Practical experience and practical applications are the key items in the field of alternative fuels. It is important to know where practical applications or tests have been realized and obtaining the experience from such cases.

At HMC/TPT a database on practical applications or test or projects in context with alternative fuels is used and updated regularly. It includes more than 200 plants inside and outside of the Holderbank Group. A typical printout for the first few examples looks as follows:

Database Alternative Fuels — "HOLDERBANK" Process Technology

COMPANY	WORKS	CODE	COUNTRY	AREA	ALTERNATIVE FUELS
Ciments d'Obourg S.A.	Obourg	OB	B	1	Coal wastes(DB 822), RDF(DB 829/P new), sludge coal, impregnated sawdust, liquids/solvents(via SCORIBEL), "déchets banal" ("mid kiln)
"Holderbank" Cement und Beton	Rekingen (S)	RK	CH	1	Wood wastes/waste timber(Aug 94-end 97)
"Holderbank" Cement und Beton (form.PCW)	Siggenthal	SG	CH	1	Waste tires, dried sewage sludge (Cap.25000 t/a), Plastics(T), animal meal
"Holderbank" Cement und Beton (form.SCB)	Eclépens	EC	CH	1	Waste tires, oil-contaminated soil, residue from anode production(T), RDF(from SORVAL), sawdust
"Holderbank" Cement und Beton (form.SCB)	Roche(S)	RO	CH	1	Filter tar(DB 489), charcoal tar, anthracite dust from Alu-indust(P), solvents, waste oil
Bündner Cement AG	Untervaz	UV	CH	1	Waste oil, dried sewage sludge, fuller earth (temp), oil contaminated soil,distillation residues, plastics, animal fat
Vigier Cement AG	Reuchenette	RE	CH	1	Waste oil, liquids/solvents/chlorinated(from Basel), wood wastes(P),plant wastes via (small) gasifier, Dried sewage sludge
CEVA Prachovice	Prachovice	PR	CZ	1	Waste oil
Alsen-Breitenburg GmbH	Lägerdorf	LD	D	1	Waste tires(SA/Lepol), fuller earth(DB 102/MF and SF), shredder(S), RDF(T/S), waste oil, high viscosity residue from distillation, carbon bearing flyash, paper sludge, rubber chips
Nordcement AG	Hardegsen (S)	HD	D	1	Landfill gas, waste oil(P), waste tire granulate(T)
Nordcement AG	Hoever	HV	D	1	Waste tires, tire granulate(P), fuller earth(T/S)
HISALBA	Carboneras	CS	E	1	Contam. vegetable oil (single action)
HISALBA	Jerez	JE	E	1	Waste oil
HISALBA	Lorca	LO	E	1	Waste oil, solvents, waste tires (mid kiln)
HISALBA	Torredonjimeno	TD	E	1	Pressed olive cake (S)
Groupe Origny S.A.	Altkirch	AL	F	1	Liquids, Tires, solids(MF)
Groupe Origny S.A.	Champagnole(S)	CN	F	1	Tar(residue from distillation), liquids/solvents/waste oil(via SCORI)
Groupe Origny S.A.	Dannes	DN	F	1	Landfill gas (P), waste tire chips(T,S), impregnated sawdust
Groupe Origny S.A.	Héming	HE	F	1	Waste tires(T), Solids"DIB", car shredder wastes"RBA"(T), solvents
Groupe Origny S.A.	Lumbres	LU	F	1	Liquids, solids, sawdust
Groupe Origny S.A.	Origny (S)	OY	F	1	waste oil, high viscosity liquids, coal sludges
Groupe Origny S.A.	Rochefort	RF	F	1	Tar(residue from distillation), liquids/solvents/waste oil(via SCORI)

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Database Alternative Fuels

"HOLDERBANK" Process Technology

COMPANY	WORKS	CODE	COUNTRY	AREA	ALTERNATIVE FUELS
Cementeria di Merone	Merone	ME	I	1	Fuller earth(DB 101,650), pitch("Laboni"), refinery wastes, waste oil(S), acid sludge(S/DB 878), liquids, highly viscous oil "bitoil" (P)
Cementeria di Merone	Cassago	CO	I	1	Graphite
Cementeria di Merone	Ternate	TE	I	1	Rubber chips, RDF, Plastics
Cementeria di Merone	Morano	MO	I	1	Plastics, RDF, tire chips
Hirocem A.S.	Rohoznik	RN	SQ	1	Waste oil, waste tires, impregnated saw dust
St. Lawrence Cement, Inc.	Beauport (S)	BP	Canada	2	Waste oil, spent potliners (T), sawdust, paper sludge(P)
St. Lawrence Cement, Inc.	Joliette	JO	Canada	2	Wood wastes(T,P), waste oil(baseline test 1990), waste tires(mid kiln)
St. Lawrence Cement, Inc.	Mississauga	MI	Canada	2	Waste oil(DB 103), liquids(chlorinated), PCB(S/DB 496), solvents, RDF(P)
Holnam	Ada	AD	USA	2	Waste tire chips, TDF,whole tires (Mid kiln, P)
Holnam	Artesia	AR	USA	2	Liquids (WDF)
Holnam	Clarksville	CV	USA	2	Liquids(also chlorinated)/solvents, Synfuel(DB1072), waste tires chips
Holnam	Devils slide	DS	USA	2	Tires/TDF < 2",wastes from diapers prod.
Holnam	Dundee	DU	USA	2	Waste oil, liquids(also chlorinated)/PCB(S/DB 238),waste tires chips(T), TDF(P)
Holnam	Holly Hill	HH	USA	2	Residues fro Alu-smelter (pot liners, titer tar,S), domestic refuse/RDF(T), liquids, synfuel(DB1072), tires/TDF < 3/4" (P)
Holnam	Mason City	MC	USA	2	Waste tires (P, mid kiln), tire chips
Holnam	Portland	PD	USA	2	Waste tires (TDF) (P), Asphalt(gilsonite) (P)
Holnam	Theodore	TH	USA	2	Waste tires chips
Holnam (Holnam Texas L.P.)	Midlothian	ML	USA	2	Waste tires chips(via PC)
Independent Cement Corporation	Hagerstown	HT	USA	2	Waste oil(permit P),waste tires(P complete+TDF)
Juan Minetti S.A.	Yocsina	YO	Argentina	3	Plastic-chips (T), liquids(T,P)
"Holdercim" Brasil S.A.	Pedro Leopoldo	PL	Brasil.	3	Charcoal from steel production , heavy oil with high viscosity, liquids, solids in pales and big bags, sawdust/RESOFUEL (P), waste tires (P)
"Holdercim" Brasil S.A.	Barroso	BO	Brasil.	3	solids
"Holdercim" Brasil S.A.	Cantagalo	CT	Brasil.	3	Solids, tar
CIMINAS - Cimento Nacional de Minas S.A.	Sorocaba(S)	SO	Brasil.	3	Rubber/waste tires(S,small size in SZ)
Industria Nacional de Cemento S.A.	Cartago	CG	Costa Rica	3	Palmnut shells(Coquitos,Olotes), wood wastes, fresh wood(P)

Database Alternative Fuels

"HOLDERBANK" Process Technology

COMPANY	WORKS	CODE	COUNTRY	AREA	ALTERNATIVE FUELS
Cementos Progreso S.A.	La Pedrera	LP	Guatem.	3	Liquids
Cementos Progreso S.A.	San Miguel	SM	Guatem.	3	Waste oil
Apasco S.A.	Ramos Arizpe	RA	MEX	3	Liquids, tire chips
APASCO S.A.	Acapulco	AC	Mex	3	Waste oil
Apasco S.A.	Apaxco	AP	Mex	3	Waste tires, liquids
Cementos Caribe C.A.	Cumarebo	CU	Venezuela	3	Refinery wastes (lodos) (P)
Milburn New Zealand Ltd.	Westport	WP	Neuseeland	5	Ship slops, waste oil(P)
Iligan Cement Corp.	Kiwalan	KW	Philipp.	5	Wood wastes
Alpha Cement	Dudfield	DF	RSA	6	Waste tires (T)
Eiberg Cement	Kufstein	-	A	F1	Waste tires
Gmundner Zementwerke Hans Hatschek AG	Gmunden	-	A	F1	Waste tires chips(DB 597), wood chips(T/S), waste oil/liquids
Leube Zement		-	A	F1	Plastic, waste tire chips
Perlmooser Cement	Mannersdorf	-	A	F1	Waste tires, plastics(T)
Perlmooser Cement	Retznei	-	A	F1	Waste tires
Perlmooser Cement	Rodaun	-	A	F1	Waste tires
Schretter & Cie.	Vils	-	A	F1	Waste tires
Wietersdorfer & Peggauer Zementwerke	Peggau	-	A	F1	Waste oil, solvents
Wietersdorfer & Peggauer Zementwerke	Wietersdorf	-	A	F1	Waste tires, liquids, plastics (T)
Woptinger Stein-& Kalkwerke	Waldegg	-	A	F1	Paper-sludge from peper ind.
CBR	Antoing	-	B	F1	Waste tires chips (<20mm in PC), Installation for paper,cardboard,wood("BPC"),with grinding(S),plastics(PET),car-shredder("RBA")
CBR	Harmignies	-	B	F1	Liquids ("blanfuel") (P)
Cimenteries CBR	Lixhe	-	B	F1	Waste tires, sawdust containing liquids, paper wastes, liquid wastes
Jura-Cement	Wildegg	-	CH	F1	Waste tires, Plastics(T), Gasification waste tires/wood (T,P)
Portlandcementwerk AG Olten	Olten	-	CH	F1	Oil-contaminated soil, waste oil(temp)
Anneliese Zementwerke AG	Werk I/II	-	D	F1	Landfill gas, waste tires(DB 827), pressed refuse(T)

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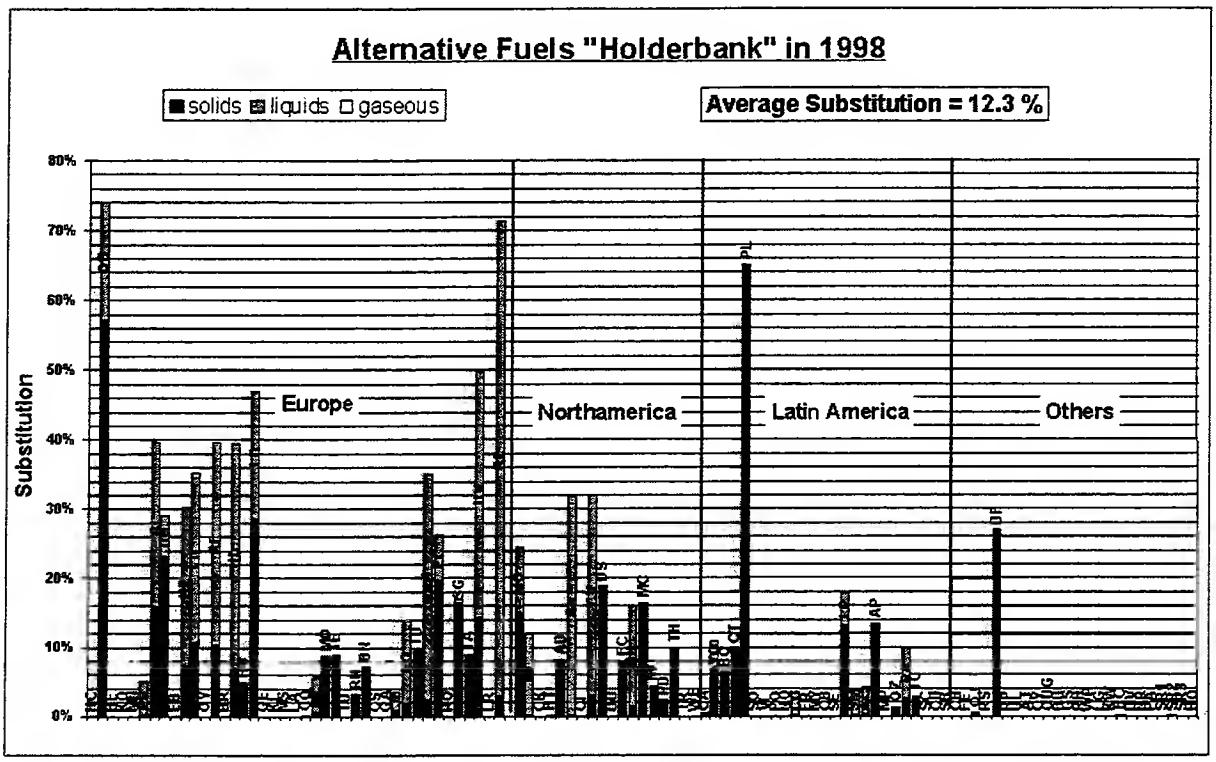
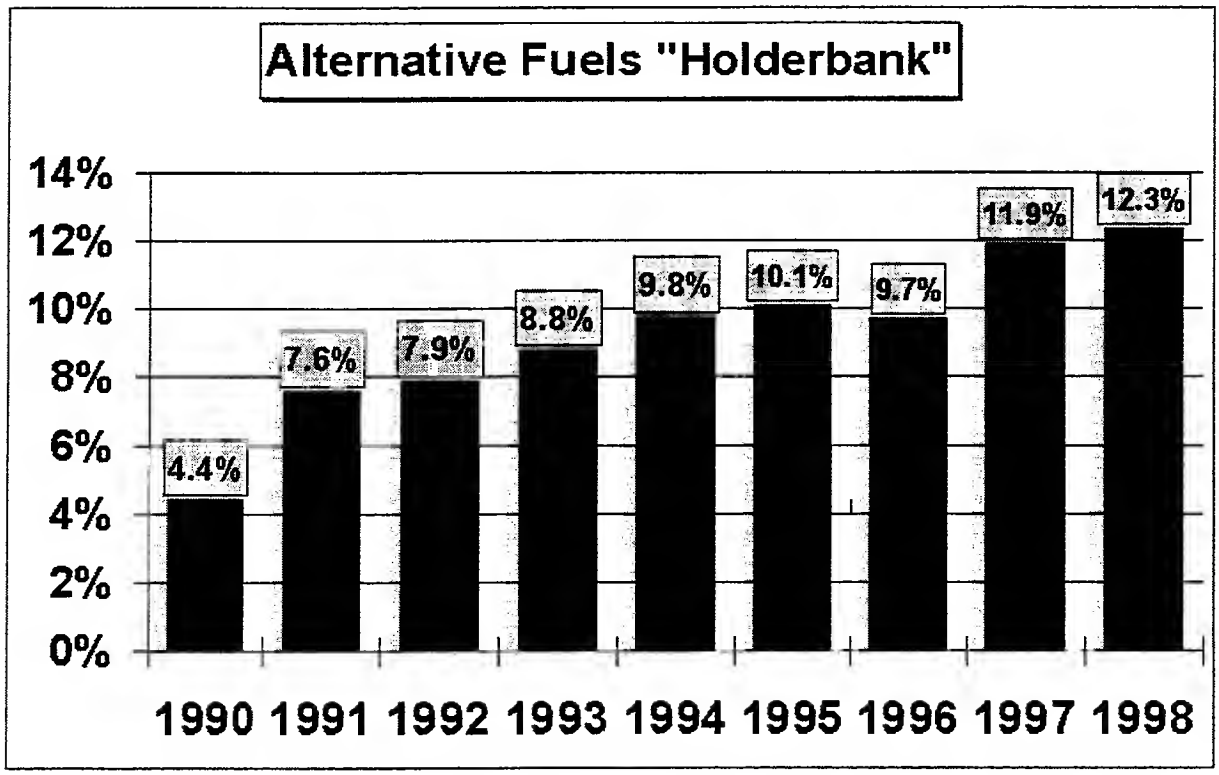
Fig. 1 „Holderbank" Alternative Fuels





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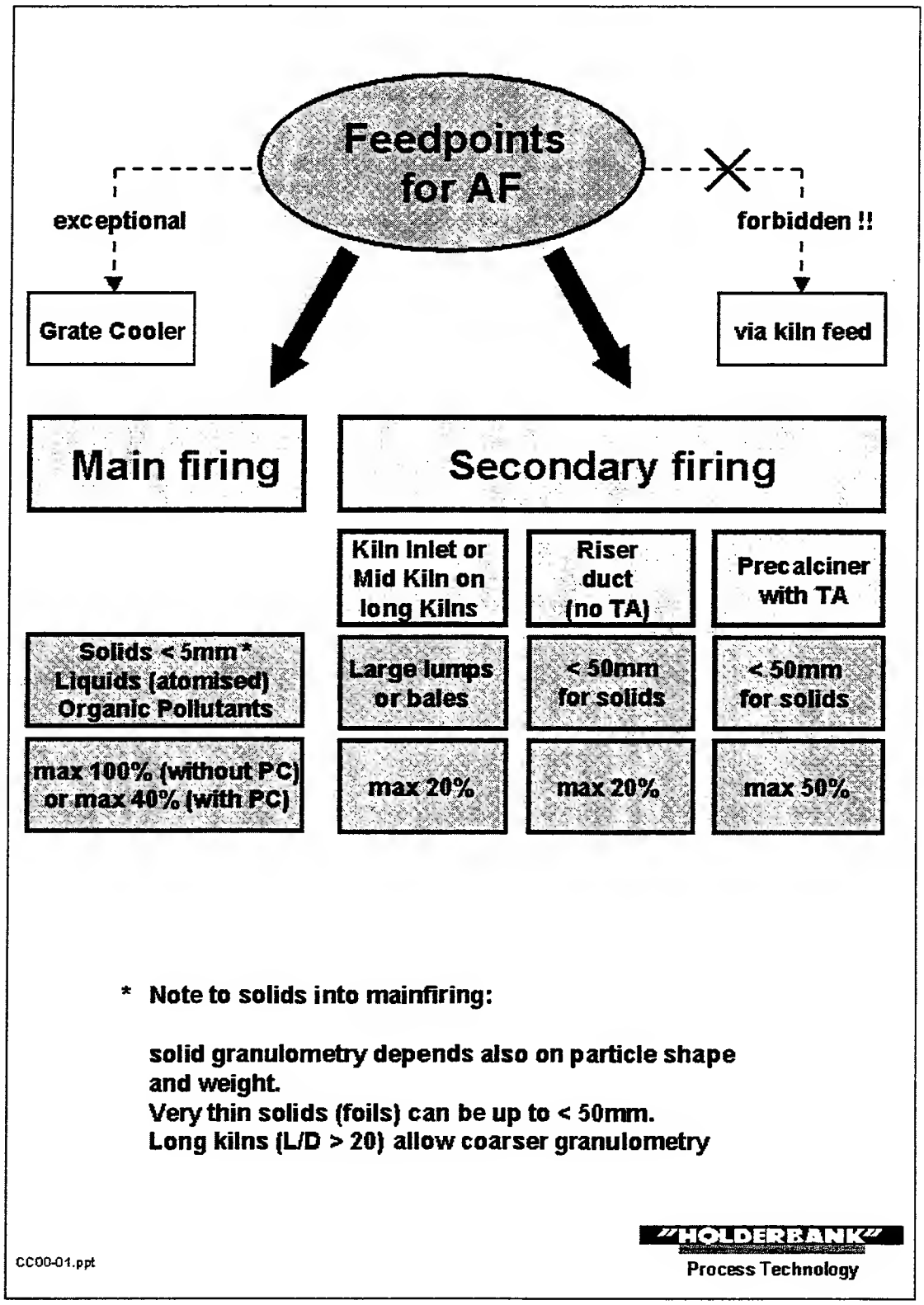
3.2 Feedpoints for Alternative Fuels

Fig. 2 Feedpoints for AF



Regarding the selection of feedpoints the following comments apply:

- Solid fuels of large size tend to produce more combustion problems especially when pushed to high substitution rates. So the practical substitution rates are often below the above optimum figures. Remedy is possible by better preparation (size reduction), if economically feasible.
- In exceptional cases solid fuels can be transferred into a combustible gas by means of a process integrated gasifier. The gasifier is then the "ultimate preparation" which allows a comparatively easy burning. Since such solutions are expensive they are reserved to special applications (the example of tire gasification is mentioned in this paper).
- The feed point via kiln feed is forbidden because of the emission problems during preheating (VOC, CO). This feed point is reserved for alternative materials with no organics. The only exception would be kiln systems where the kiln feed enters the combustion zone without preheating (one stage precalciner kiln at FC) or kilns with VOC removal system (carbonfilter SG, oxidiser at DU).

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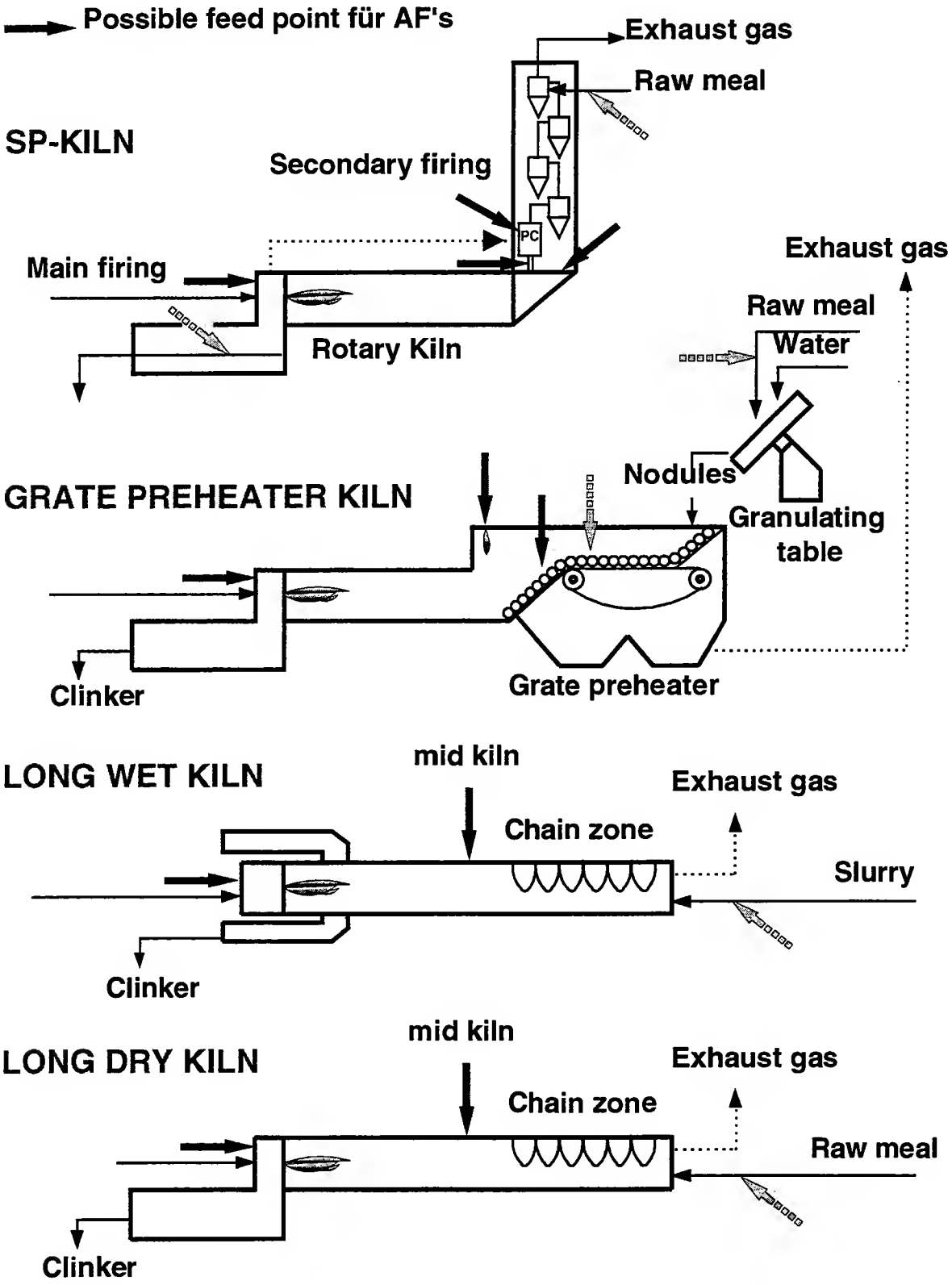
Fig. 3 Feed Points for Alternative Fuels to Cement Kilns



Regarding the different kiln systems the following rules apply:

A Circulation Phenomena

- Kiln systems with cyclon suspension preheater and without bypass are most sensitive to circulating phenomena. If the following criteria are not respected, the AF use will cause severe problems or will fail
  - Keep total chlorine input below 200 – 300 mg/kg clinker (from all fuels and raw materials). If this limit is exceeded a bypass is required. The cyclone preheater without bypass is not forgiving excessive Cl input, it will just plug.
  - Keep sulfur cycles under control! Unlike Cl the most critical factor is not the possible sulfur input by AF but the impact of poor AF combustion on sulfur volatilisation. This will promote a high sulfur cycle and sulfur pluggings. Remedies: improve combustion, higher O2 at kiln inlet, enhanced preheater cleaning.
- Kiln systems with grate preheater (Lepol) are of similar sensitivity to circulating phenomena as cyclone preheater kilns. Condensation of volatile elements in the nodule bed on the preheater can disturb its permeability and thus the kiln operation.
- Long dry kilns or long wet kilns are more forgiving in terms of circulating phenomena as they have no critical narrow cross sections. However, rings and build-ups in the rotary kiln also occur but it takes longer until they grow to a critical size. If the kiln system works with 100% dust reintroduction the sustainable chlorine limit is the same as on a cyclone preheater kiln (200 – 300 mg/kg clinker).
  The difference to the cyclone preheater kiln is that it is easy to realize a valve for chlorine on a wet kiln if the kiln is equipped with an EP dedusting. This allows to extract a highly enriched fine dust selectively that removes chlorine effectively when being discarded. Like that up to 5000 mg/kg cli chlorine input can be handled. In this case the discarded dust causes an additional disposal problem, because it cannot be added to the cement due to the chlorine limit for cement (0.1% Cl).

Chlorine limit

Regarding the chlorine input the following diagram helps to get a quick overview of what can be accepted as total input if then chlorine would come only from AFR (whether it's a Fuel or a Raw material does not matter here).

The following limits apply:

Limit A:	Normal SP kiln with completely closed dust loop
Limit B:	SP kiln with some 20% bypass or wet kiln discarding medium dust quantities
Limit C:	Maximum possible for wet kiln discarding high dust or SP kiln with 100% bypass

Example:

An AFR with 1% Cl at a relative input of 10 g AFR per kg clinker creates an input of 100 mg Cl/kg cli, which is not critical (assuming no other inputs of Cl).

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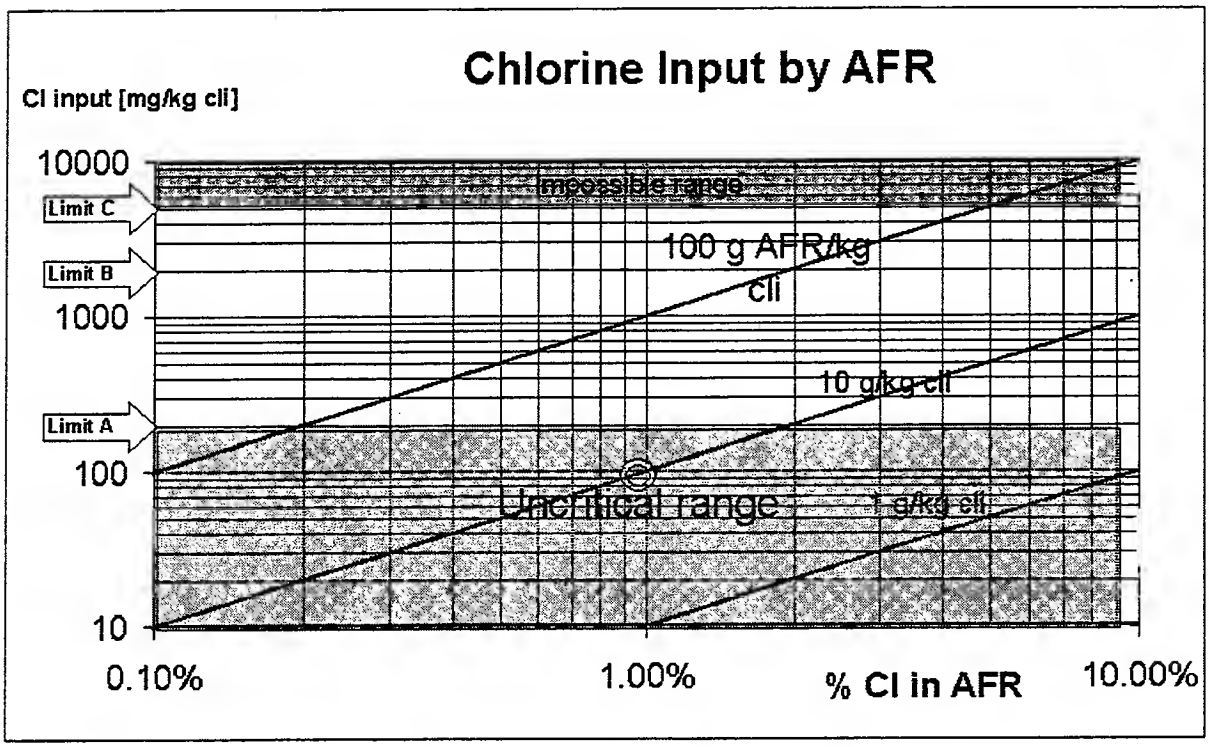
Fig. 4 Chlorine Input by AFR



B Temperature and gas residence line

If stable toxic organic compounds in AF's are an issue the main kiln features for their destruction have to be known.

- Main firing
  - Flame temperature 1800 – 2000°C
  - Total gas residence time in rotary kiln depends on kiln system as follows
    - short kiln (2 support) approx. 3 sec.
    - normal SP/PC kiln more than 5 sec.
    - long wet or dry kiln more than 10 sec.

So typically a gas residence time of approx. 5 sec. above 1200°C can be expected.

- Secondary firing (no PC), with unextended riser duct 1 sec. above 820°C
- Precalciner with tertiary air 2-4 sec. above 860°C
  (in case of hot spot design peaks up to 1200°C)

For optimum combustion and safe destruction of stable organics only the main firing shall be used. Example: waste oil that is contaminated with traces of PCB.

Other feedpoints are reserved for less critical AF's or if they are used for critical substances tests may become necessary. To avoid extensive tests it is often easier to select the main firing.

Finally the above temperatures are not valid for start up or upset conditions so critical AF's should only be used under normal operating conditions.

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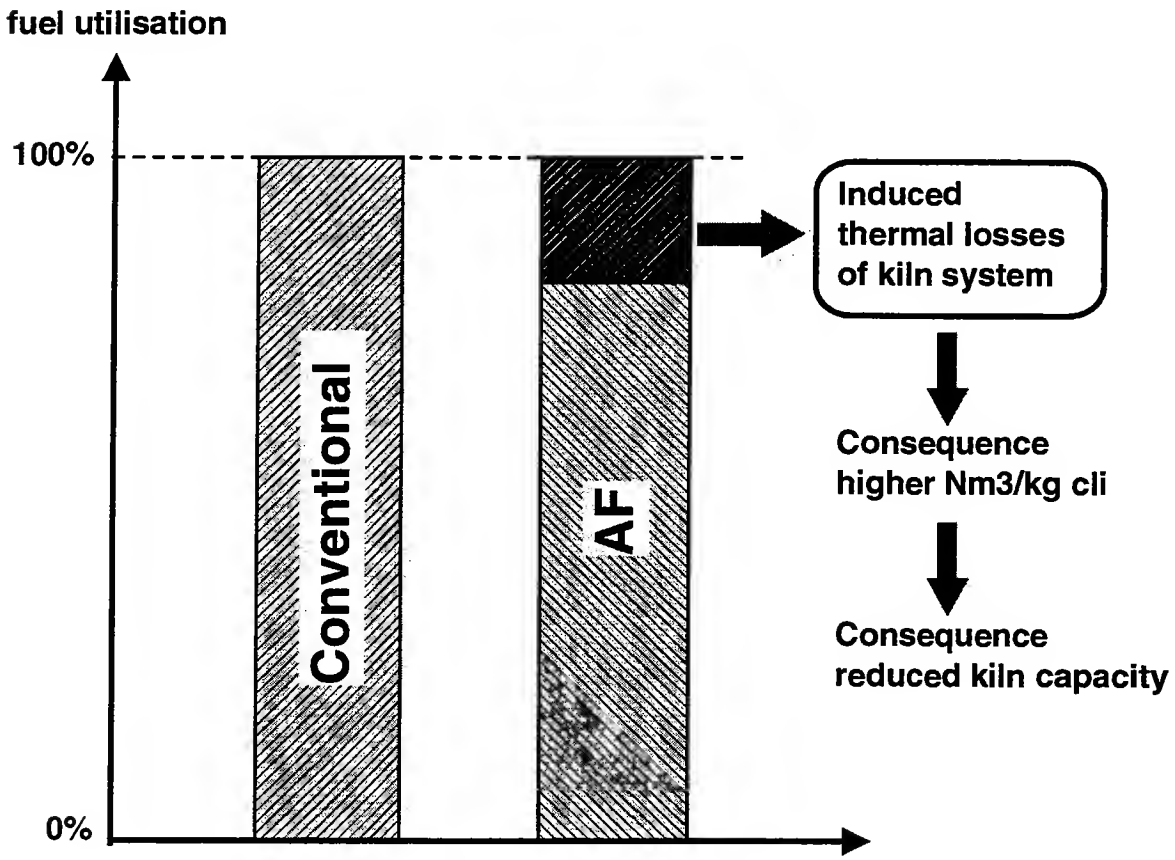
3.3 Substitution effect and potential capacity loss

Introduction

If low grade fuels are used to substitute high grade conventional fuels (coal, oil, gas) the kiln will react with certain effects that will increase the thermal consumption and decrease the maximum kiln capacity. Both phenomena are related to each other.

If the energy costs for AF's are low or even negative one may think the increase in heat consumption is not that negative because the additional consumption can be covered with low cost AF. This is only half the truth. If maximum production is required at the same time every ton of clinker that cannot be produced means a financial loss. Roughly every one % increase in heat consumption also means one % loss in potential kiln capacity. And if kiln availability is lower due to AF's the OEE can decrease even further.

Fig. 5 Conventional Fuel versus AF



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Factors for increased heat consumption

Why can AF's increase the thermal losses on a cement kiln system and thus create "induced losses"? There is a defined number of reasons that contribute to such effects as follows:

1) High water content in AF

   A high water content increases both exhaust gas quantity and exhaust gas temperature.

   Consequence: increased heat loss in the exhaust gas that needs to be compensated by more fuel.

2) High ash content in AF

   A high ash content reduces the amount of kiln feed that passes through the preheating zone and decreases the heat recovery by kiln feed. The exhaust gas temperature increases.

   Consequence: increased exhaust gas loss that needs to be compensated by more fuel.

   Note: the same effect happens if cold raw material is added directly into the precalciner.

3) Reduced combustion properties

   Certain AF's have poor combustion properties because of too coarse granulometry. Depending on the control strategy this can mean increased CO losses or increased O_2 level to compensate this.

   Example: whole tires at kiln inlet

   Consequence: No matter what the strategy is, the final result is always a higher energy loss in the exhaust gas, which has to be compensated by more fuel.

4) Fluctuating AF feed (at good combustion properties)

   Fluctuations in AF energy input can result from

   - Inhomogeneous AF properties (CV)
   - Fluctuations of the dosing rate due to more difficult handling properties

   Both items have the same impact as for item 3). What happens if a temporary excess of energy input occurs? Either a CO loss is generated or the O_2 has to be set so high that no CO is generated.

   Consequence: same as in case 3).

   (The difference to case 3) is that this can happen even with fuels that have good combustion properties.)

   Example: poorly homogenized liquids or poor performance of dosing system.

5) Cold air introduction

   Solid coarse AF usually need a high amount of air for pneumatic injection or an air leakage can occur at a poorly sealed feed chute for AF. This has the same effect as an increase of the primary air on a burner.

   Consequence: Inleak of additional cold air (due to AF) causes higher fuel consumption.

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6) More circulation phenomena due to AF

Circulation phenomena do not only occur because AF can introduce circulating elements but also be a result of poor combustion (local CO formation). Although the evaporation and condensation of circulating elements induces a heat transport from the hot zone to the colder zones, the direct effect on balance heat consumption is not that critical. What really disturbs is the unstable kiln operation, which results in reduced availability and higher average fuel consumption.

Consequence: reduced availability due to unstable operation/stoppages and increased average fuel consumption.

The majority of the above factors (1, 2, 4 and 5) can be quantified of predicted and the other factors are based on experience. (One of the services that can be provided by HMC/TPT.)

Practical substitution effects (examples):

- Liquids with < 10% H_2O and good homogeneity 95 – 100%
- Very low grade waste as raw domestic refuse 70% or lower

So the substitution effect would be typically between 70 – 100% as long as the applications are approached in a engaged and professional way and no unusual difficulties occur. For low grade AF (high ash, high water, coarse, inhomogeneous) the lower limit of 70 % would be typical.

Conclusion

1. The potential fuel substitution value of an AF (USD/GJ) cannot be calculated by just using its net CV. A reduced effect of typically 70 – 100% can occur. This is only on basis of immediate additional thermal losses (not yet taking into account other costs that emerge when using AF).
2. When reporting the true heat consumption of a kiln, we have to accept higher consumption when using low grade AF's. Manipulating CV's for AF to get the same consumption (on paper) is physically incorrect and not a good reporting practise.
3. The potential capacity loss when using AF's is directly linked to the % increase in heat consumption. Increased heat consumption and possibly also reduced kiln availability can have an important impact on OEE.

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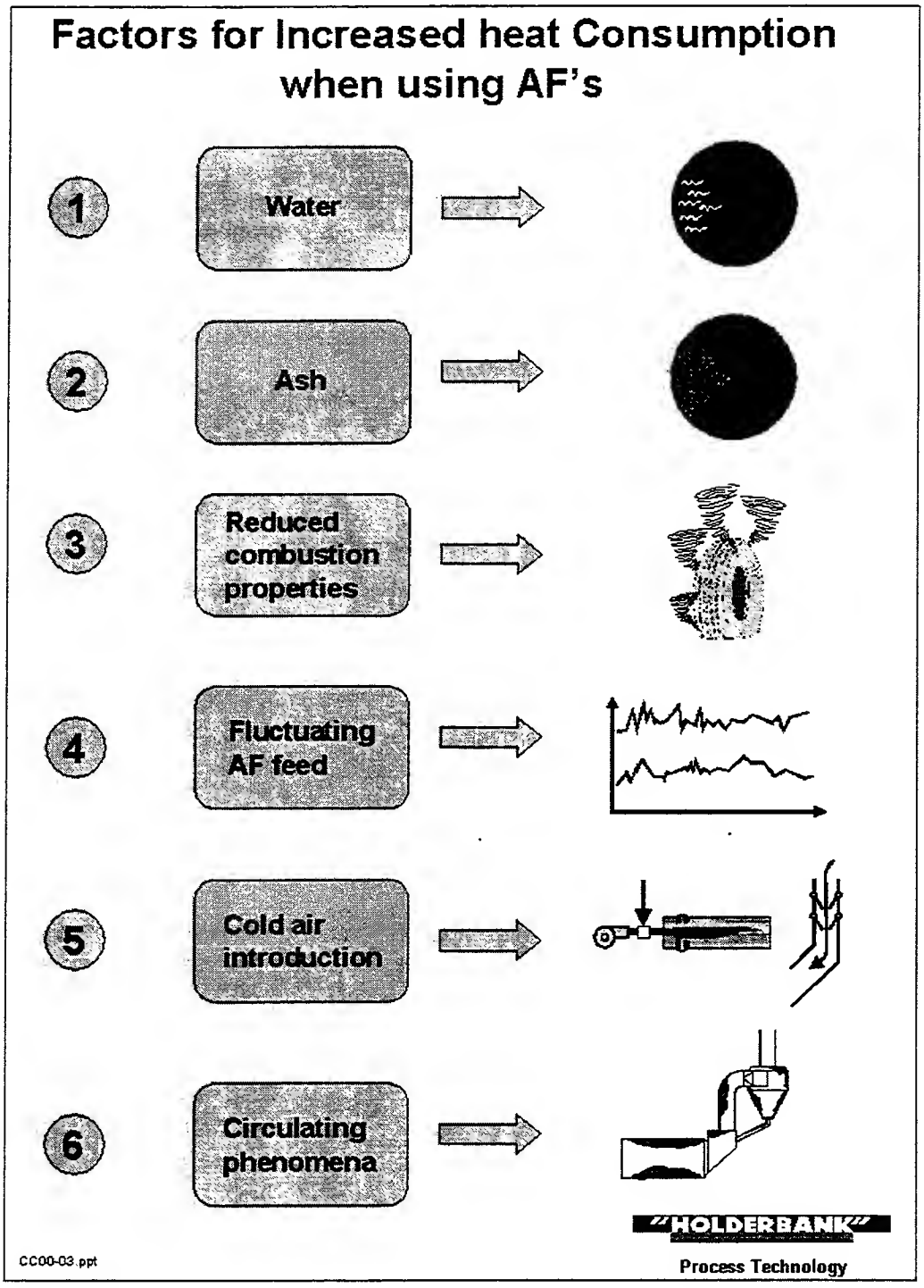
Fig. 6 Factors for Increased Heat Consumption



3.4 Supply and Inlet Control

3.4.1 Organisation of supply

To get AF to the cement plant can be basically done in two ways:

1) Get AF directly from a waste source.

2) Get AF through a specialised company that prepares an adequate waste blend for the cement.

Both ways are being used. The possibility 1) is adequate for certain wastes that can be used with minimum or no pre-treatment before shipped to the cement plant. A standard example would be tires.

The possibility 2) is a more professional approach, which involves also better integration in the whole waste market. Specialized companies emerged in the previous years and their number is still growing. Classical examples were

- SCORI (France)
- SCORIBEL (Belgium/Obourg)
- SYSTECH (USA/Lafarge)
- SAFETY KLEEN (USA/partly active for Holnam)

With the exception of Safety Kleen all these organisations were controlled by the cement industry.

The new companies that have emerged within Holderbank recently are mostly orientated on the SCORIBEL/Obourg model. From a technical viewpoint of a cement plant these companies provide the following functions:

- Allocating adequate waste categories to the cement plant
- Control of properties
- Preparation/pretreatment
  (in particular blending/homogenising)

The preparation of waste into a useful cement kiln AF is done externally. This seems to become the preferred approach. The supply companies within Holderbank have a common platform: the VESTA Forum.

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3.4.2 Delivery control

A delivery control at the cement plant is essential. In case of hazardeous wastes this is anyway a must (given by the permit) and this does not need to be explained further.

What is less obvious is that even harmless or non-hazardeous AF's need to be checked when delivered to the cement plant. The problem is that AF's can be contaminated with undesirable inpurities. Whether this happens intentionally or not, it needs to be excluded.

Examples:

- Waste oil

  The original motor oil would not be critical from its properties but the waste product that is finally delivered to the cement plant may be contaminated, e.g. with

  - Solvents (a small quantity of solvents decreases the flash point drastically and thus the safety can become a problem)
  - PCB (PCB contaminated oils have a high disposal fee and the cement kiln could be abused to get rid of them cheaply, PCB is not detectable by simple test methods)

- Waste tires

  When accepting waste tires it is mandatory to have a visual delivery control and the receiving area must not allow uncontrolled access for various suppliers. Some plants that believed they could do it without any control finally paid a high price because they were abused as a dumping area for non usable sizes, rims and other materials (for which they had to pay for the disposal).

- Chlorine

  It happened from time to time that suppliers came up with new solid waste mixes (RDF) or mixed plastics where they claimed very low chlorine contents. This is not always true, but difficult to disprove. There is in fact a problem that sampling of solid RDF is hardly representative and the chlorine analysis are often lower than the average bulk. So do not believe, but check what you get for chlorine, it could hurt your kiln operation.

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3.4.3 Check-list for Properties of Waste Fuels

Table 2 Checklist for Properties of Waste Fuels

• Type of waste	Name, trade name origin
• Physical state: solid liquid gaseous solid/liquid	 Size, form, grindability viscosity at ...°C, impurities mixing proportions
• Density	kg/m^3
• Calorific value (net)	MJ/kg
• Proximate analysis	Moisture, ash, volatiles, C_{fix}
• Ultimate analysis	C, H, O, N, S
• Halogens	Cl, Br, F
• Ash composition	CaO, SiO_2, Al_2O_3, Fe_2O_3, K_2O, Na_2O, P_2O_5, etc
• Heavy metals	Hg, Cd, Tl, Be, As, Co, Cr, Pb, Zu, V, etc.
• Flashpoint	°C
• Explosivity	non-explosive
• Toxicity	toxic /non toxic, safety precautions, warnings
• Legal restrictions containing transport and storage	
• Storage	Chemical or natural degradation, putrefaction phenomena, segregations, precipitations,
• Corrosivity	Construction materials required
• Mixing possibilities	Mixing with oil, water, solvents
• Quantities to be used	min, max, average (now, in future)
• Fluctuations in quality	Quality specification

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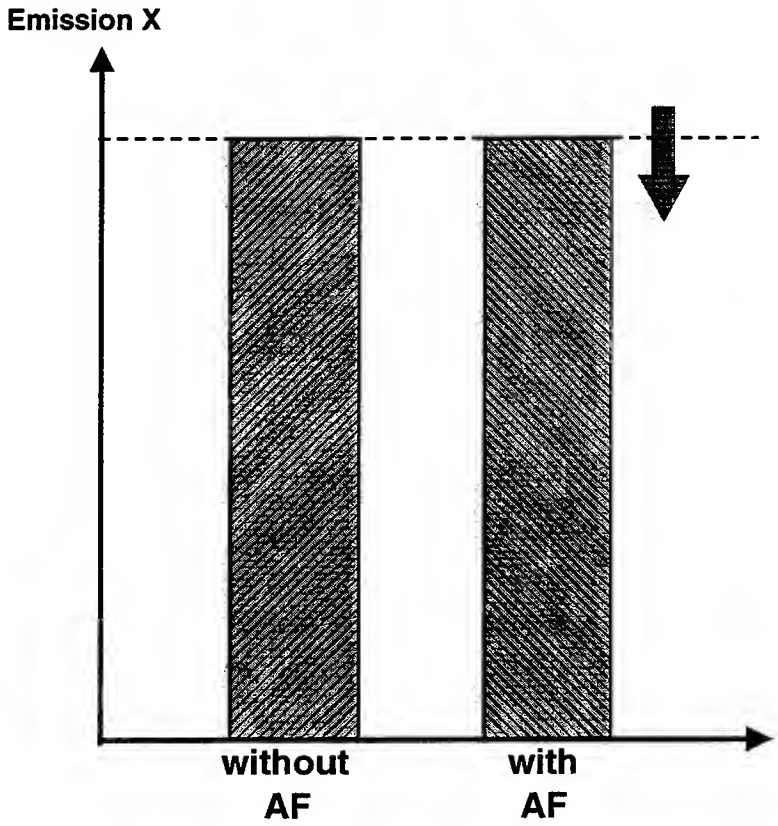
4. EMISSIONS IN CONTEXT WITH ALTERNATIVE FUELS

4.1 Introduction

If alternative fuels are used to substitute conventional fuels the cement kiln emissions are often not increased and may even drop.

Fig. 6 Emission influence AF



Emissions do occur but they are hardly caused by alternative fuels. Emission results mainly from the raw material and from the high temperature process (NOx) and the fuels have only a limited influence. E.g. the SO_2 emission on a suspension preheater kiln does not depend on the sulfur in the fuel. The difficult part can be how to handle the normal emissions if burning AF's attracts public interest and implies more stringent emission rules.

4.2 General Features of Cement Kiln Systems

"Barriers" which prevent toxic substances from being emitted or becoming harmful to the environment (see figure 7)

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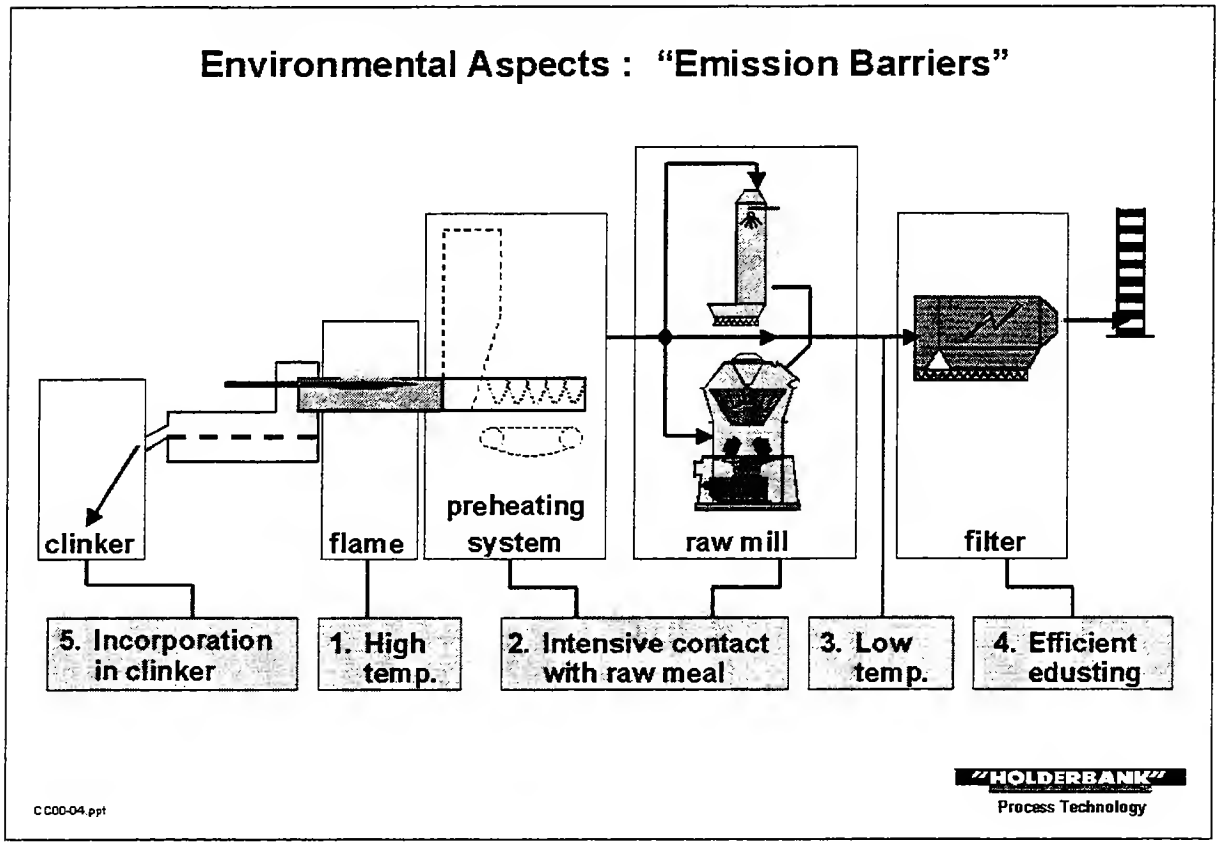
Fig. 7 Environmental Aspects: „Emission Barriers“



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1 High incineration temperature

In the sintering zone flame temperatures of some 2000°C are required for process reasons. Even very stable organic compounds (e.g. PCB) are completely destroyed.

This argument does not apply for secondary firings.

2 Contact with Fine, Dispersed Raw Meal

Intensive contact of gas and raw meal is required for process reasons (heat transfer). This produces gas purification through absorption of toxic compounds while contact occurs in counter-current pattern. Excellent retention of acid gases (e.g. HCl, SO_2) and also of most of the heavy metals is achieved in SP-preheaters and raw mills. The key is the contact of gas with fine suspended particles.

This does not apply for the bypass gas extraction, which must be considered e.g. in an emission estimate.

3 Low Final Gas Temperature (favourable equilibrium)

Condensation or absorption on surface active raw meal reduces the concentrations of toxic elements according to the physical/chemical equilibrium. This effect strongly depends on the gas temperature. The lower the stack temperature the lower will be the equilibrium concentrations of the vapours of toxic compounds. Examples for very low stack temperatures are:

- Kiln gas after passing the raw mill (during combined operation) ~ 100°C

It is therefore possible to keep emission levels low, while the gases are passing the raw mill (argument #2 and #3 are equally important).

4 Efficient Dedusting Equipment

The high absorption capacity of the kiln system avoids emissions but on the other hand can cause enrichments of the filter dust for certain elements that can reach the outer dust cycle (e.g. Tl). An efficient dedusting prevents enriched dust from getting into the atmosphere. Also no EP shut offs are acceptable. Moreover, excessive dust emissions have an over proportional negative psychological impact as all neighbours can see it.

5 Safe Disposal of Trace Elements

Trace elements or heavy metals cannot be destroyed nor can they disappear. If they are fed into the cement clinkering process and are not emitted they must have an outlet. Unlike other incineration systems, which produce concentrated and often toxic by-products, a cement kiln with complete dust reintroduction offers the unique possibility to incorporate trace elements in the clinker production in diluted and immobile form. These trace elements occur in concentrations which are usually not different from clinker, which is produced without alternative fuels and they are not leachable (exception: hexavalent Cr during make up with water).

If the dust is not completely reintroduced into the kiln or if a bypass is required the above argument - in its simple form - is no longer true.

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4.3 Special rules regarding emission behaviour on cement kilns

The reality with emission is usually too complex for a safe and accurate prediction. However, from an engineering point of view it is better to have some ideas or rules about the emission behaviour to roughly identify what could be critical or not.

- Low emission of Cl, F and Br, < 0.1% of balance input.
- Low volatile heavy metals are not critical. Emission usually < 0.1% of balance inputs.
- Medium or high volatile heavy metals can reach the outer dust cycle (Cd, Tl) or even escape in form of vaporous compounds (Hg). Whereas Cd and Tl are still below < 1% emission Hg can be emitted almost totally (again this depends on process conditions).
- SO_2 emissions are not depending on fuel sulfur in case of a SP kiln. Wet kilns however show a moderate influence by S on emissions.
- AF burning in the secondary firing usually decreases the NOx (0 – 30%).
- CO as discussed previously is often increased when using the secondary firing. Fluctuating energy input may also cause CO peaks.
- Virtually no organics result from AF burning (even in cases of CO formation it does not necessarily correlate with organics). The improper use of AF via kiln feed is of course excluded from this consideration.
- Dioxine/furan emissions on SP kilns are not critical in view of a limit of 0.1 ng TE/m^3. Again, there is no correlation with AF burning.

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5. ADVANTAGES / DISADVANTAGES

Nowadays the destruction of wastes in special incineration plants is being improved e.g. by addition of more effective gas cleaning. Under this aspect the question may arise whether it still makes sense to incinerate wastes in cement kilns instead of in special incinerators. To answer this question the advantages and disadvantages of a cement kiln must be compared as follows:

Advantages	Disadvantages
Possibility of high temperature incineration (up to 2000°C) which destroys toxic organic compound completely	
The majority of heavy metals can be captured at > 99.9% in the kiln system (absorption by raw meal)	Some limitations have to be considered, e.g. Hg, Tl, Cr
Acid gases are retained efficiently (e.g. no HCl emission)	Because of kiln internal circulation phenomena, cement kilns and especially SP-kilns cannot accept high inputs of chlorine
No solid residues occur because the ash is incorporated in the clinker. No landfill is required	In the case of wet kilns or bypass installations solid residues in the form of dust may need disposal
If the necessary rules are observed there will be no influence on emissions and clinker quality	For psychological reasons some customers may not accept cement which is produced by using „waste“
No necessity for a new incinerator since the cement kiln is already existing	
High environmental awareness helps to allocate certain wastes to cement plants	Realization of a project may be difficult and time consuming at the level of public discussion and obtaining of permission

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6. PRACTICAL APPLICATIONS

6.1 Waste Tires

Energy potential

- Calorific value (depends on quantity of steel included) 28 to 32 MJ/kg

Comparison for an industrialized country (per capita):

a) Energy required to burn clinker at 500 kg cement/cap. a	(at 3.6 MJ/kg)	=	1800 MJ/a cap.
b) Energy from waste tires at 6 kg tires/cap. a	(at 30 MJ/kg)	=	180 MJ/a cap.

Theoretical overall fuel replacement
(if tires were fully available for cement industry) 10%

Conclusion:

Tires are an important energy source and so far the most frequent application of AF. The practical attractivity depends on the disposal fee that is available. Current values (in different areas) range from 0 – 60 USD/t.

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Typical Composition of Tires

Constituents	
Rubber	36.0%
Filler (soot, SiO_2)	37.0%
ZnO	1.2%
Softeners	3.0%
Sulfur	1.3%
Steel, textiles	18.0%
Rest	3.5%
Total	100%

Chemical analysis	
C	70%
H	7%
S	1 ...3%
Cl	0.2...0.6%
Fe	15%
ZnO	2%
SiO_2 + rest	5%
Cr	97 ppm
Ni	77 ppm
Pb	60 to 760 ppm
Cd	5 to 10 ppm
Tl	0.2 to 0.3 ppm

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Incineration in Cement Kilns

Incineration of waste tires in cement kilns has nowadays become a frequent method. At least 40 cement plants are known to do so. They can usually substitute 10 - 20% of their fuel requirements.

From an environmental point of view this method is considered as proven and advantageous (energetical recycling, low emission, no solid residues). It is often well accepted by the authorities.

Application of feeding methods:

1) Whole tires
This is the most frequent application, based on the secondary firing principle. Originally, this was first developed and used on dry SP-kilns but then also extended to long dry or wet kilns.

   a) Kiln inlet of suspension preheater kilns
   A feed system according to the figure 8 involves an investment of some USD 2 Mio. for a fully automatic installation.

   b) Mid kiln introduction device on long dry or wet kilns
   The principle is similar to a) but the introduction chute is rotating with the kiln shell, i.e. the tire feed is coupled with the kiln revolution. Figure 9 shows the introduction chute of the "Cadence" system as used at the Joliette plant.

2) Shredded tires / TDF
TDF = tire derived fuel

   Shredded tires allow for a more regular fuel input into the kiln and have a higher density (advantages for transport and storage).

   Shredding costs are some 30 - 60 UDS/t. Sometimes this is already paid by disposal fees. Shredding is normally not operated by the cement producer.

   The use of shredded tires < 300 mm on suspension preheater kilns as in figure 10 is rare because it would cost less to use complete tires at the kiln inlet.

   The use of shredded tires < 50 mm has some applications on long kilns in North America where still many long dry and wet kilns are in operation. The tire chips are injected into the burning zone. Figure 11 shows the example of the Seattle plant with 15% substitution. Shredded tires or TDF < 50 mm are successfully used on precalciners according to figure 12 in the plants Midlothian, Theodore, Ramos Arizpe and Lägerdorf.

3) Ground tires (< 5 mm):
Theoretically, ground tires would be the ideal fuel for any primary firing (without compromise). However, the costs for grinding are usually prohibitive. Ground rubber as granulate is normally too expensive as fuel. Nevertheless, a Group Plant in Germany and HCB have tried this and gone through a learning process.

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4) Pyrolisis/Gazification of tires:
The Japanese have realised gasifications for cement kilns and reported more than a decade ago. It was based on a reactor (shaft) with understochiometric air addition at 700°C. The hot gas produced was directly sent to the cement kiln.

1999 a new gasifier for whole tires (industrial scale) was commissioned by Polysius at Jura Cement in Wildegg (Switzerland). The hot combustible gases are used in the precalciner. Investment for a 3 t/h installation is in the order of 3 Mio. USD. (Fig. 13)

The gasification can potentially help to optimize the use of tires, which still needs to be demonstrated on a long-term basis. The costs are significantly higher than for burning the tires directly. Direct burning - not gasification - should always be the first option to be investigated.

Fig. 8 Whole Tire Handling and Lump Fuel Kiln Feed (HCB-Eclépens)

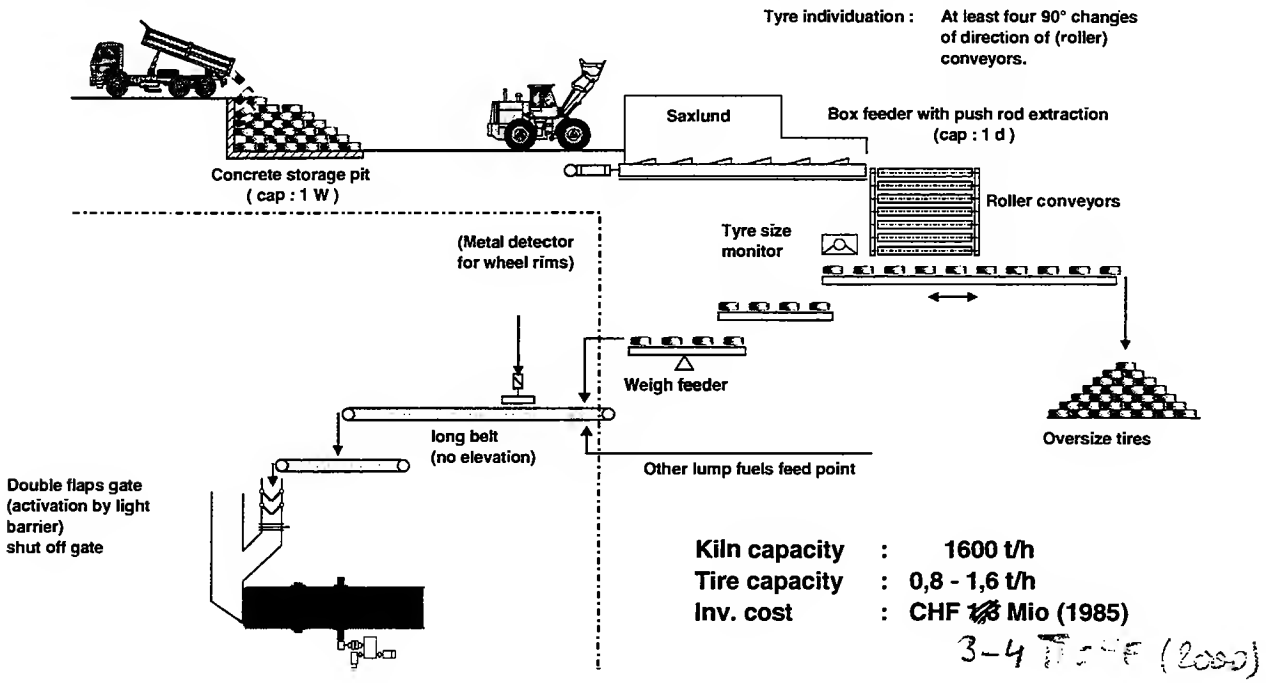

Fig. 9 "Cadence valve" on Joliette kiln

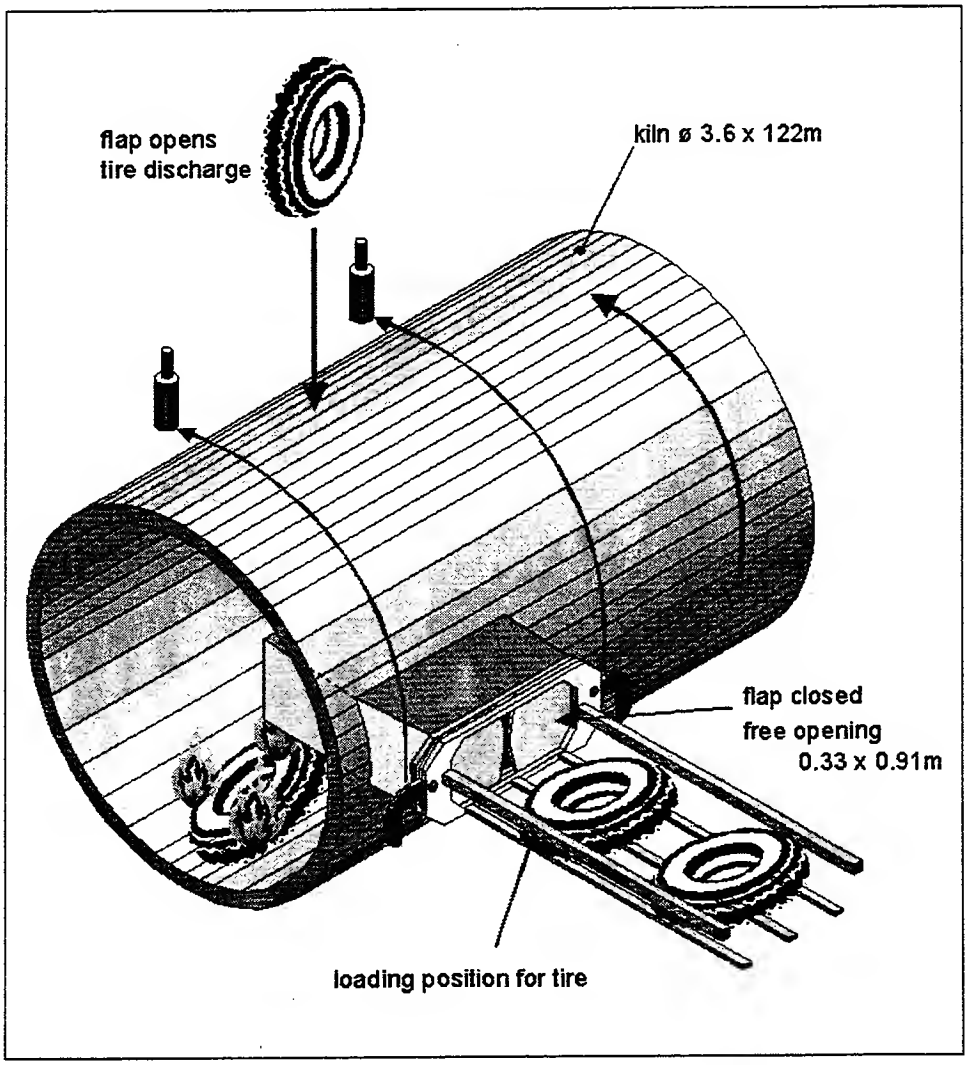

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Fig. 10 Handling of Tyre Chips and Lump Alternative Fuels at Altkirch

Handling of Tyre Chips and Lump Alternative Fuel at ALTKIRCH

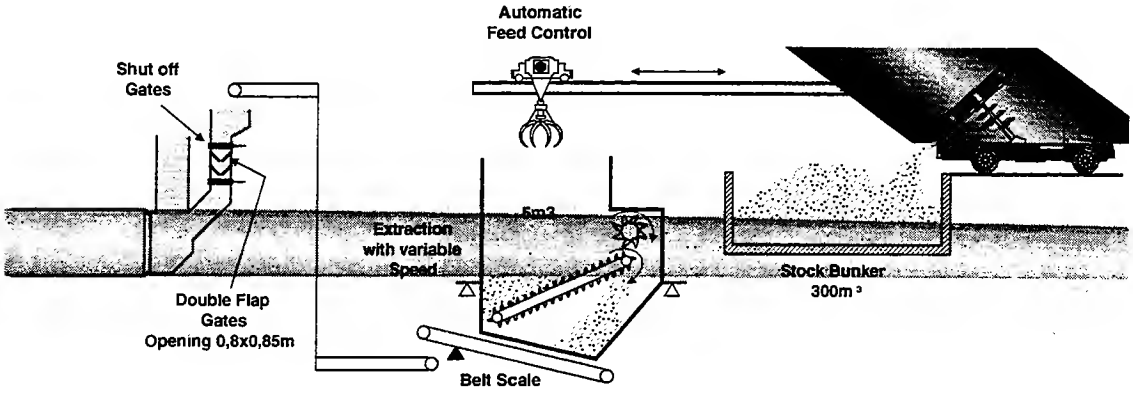

Fig. 11 Burning of TDF (Tire Derived Fuel) at the Seattle plant

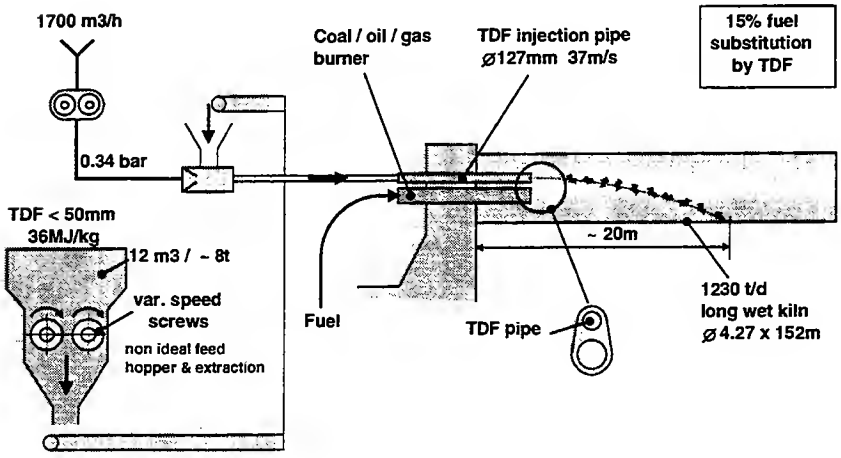

Fig. 12 Generic tire chip feeding system

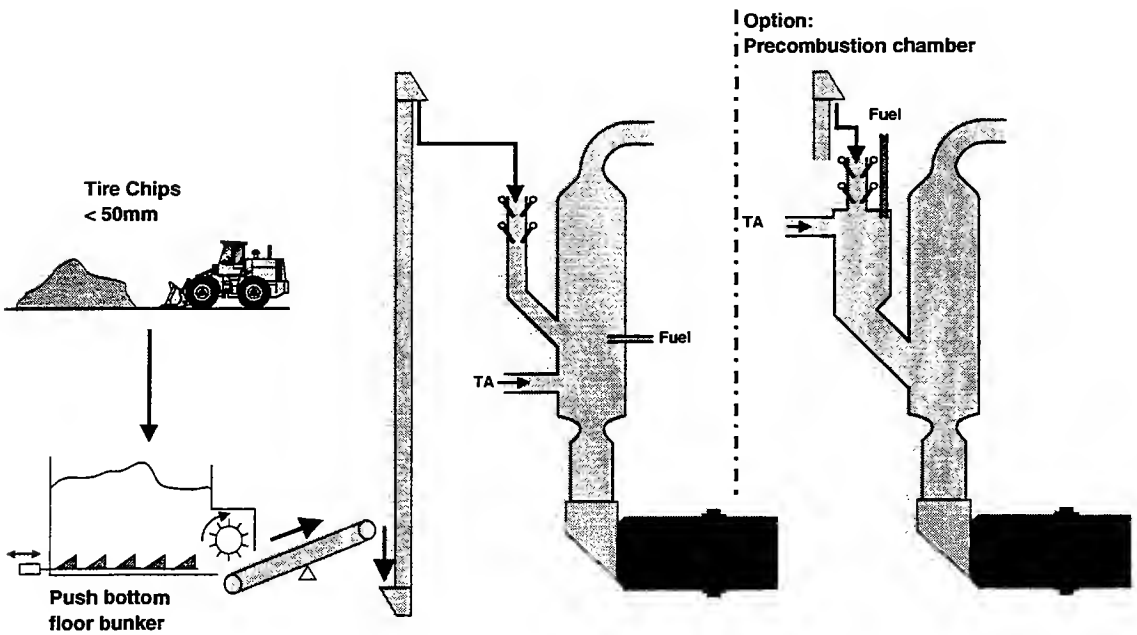

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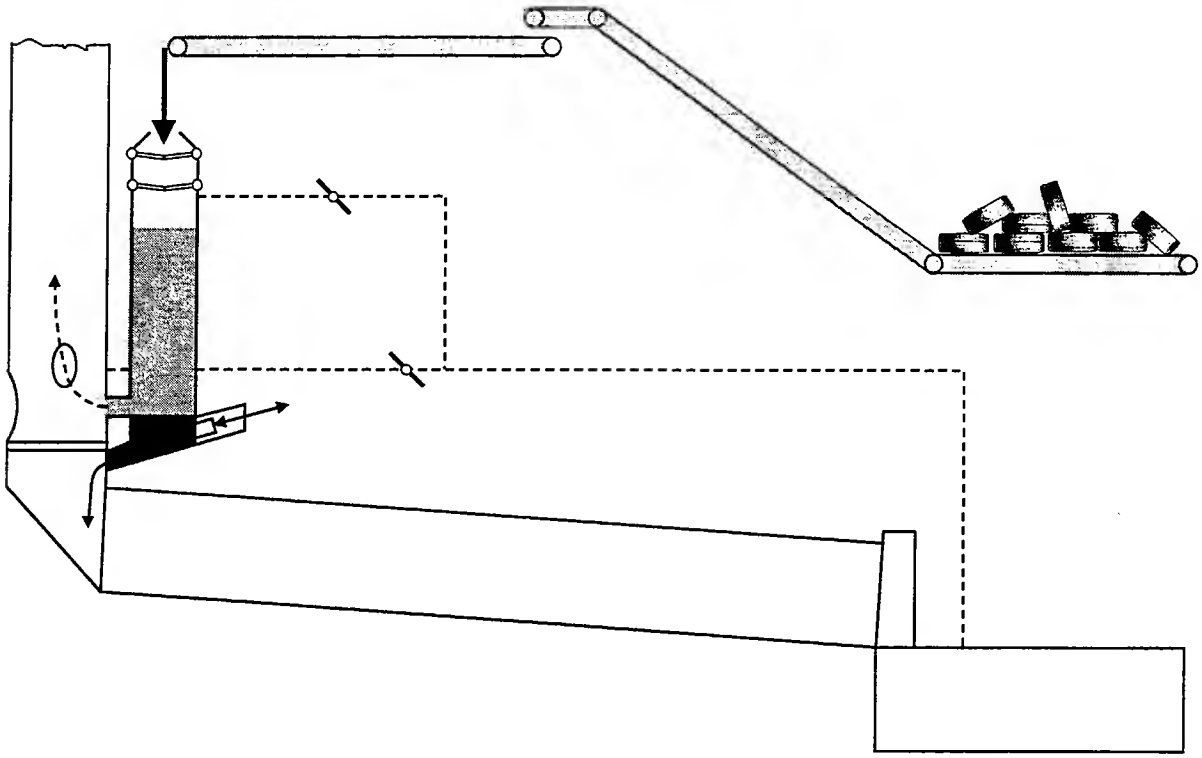
Fig. 13 Integrated Gasifier (Polysius) for whole tires



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6.2 Domestic Refuse / RDF

- Example Germany:

	Domestic refuse	Cement consumption
Quantity	400 kg/cap a	450 kg/cap a
Energy content (heat energy)	3.4 GJ/cap a	1.7 GJ/cap a

- The energy content contained in the domestic refuse is twice the energy consumption of the cement!
- However, a complete use of the energy from raw domestic refuse in the cement industry is by no means feasible. Reasons:
  - poor homogeneity, inadequate size, difficult handling
  - Cl-content of 0.5...1% Cl which can cause clogging problems in the kiln
  - low calorific value (8 to 10 MJ/kg)
  - low density and high transport costs per heat unit
  - competition to existing incinerators

Conclusion:

Domestic refuse needs intensive processing in order to eliminate undesired fractions and to obtain a reasonable burnable fraction. Such a fraction may represent 30...50% of the original refuse, the rest needs further disposal. The burnable fraction is called RDF (refuse derived fuel) and offers somewhat better properties, e.g. a CV of 12...16 MJ/kg.

Experience:

The first application was in the early 80's at BCI/Westbury, now stopped.

The most important application today is the Wittekind plant in Erwitte (Germany) according to figure 14 with 50% substitution and a chlorine bypass. Otherwise very few plants have realized major applications.

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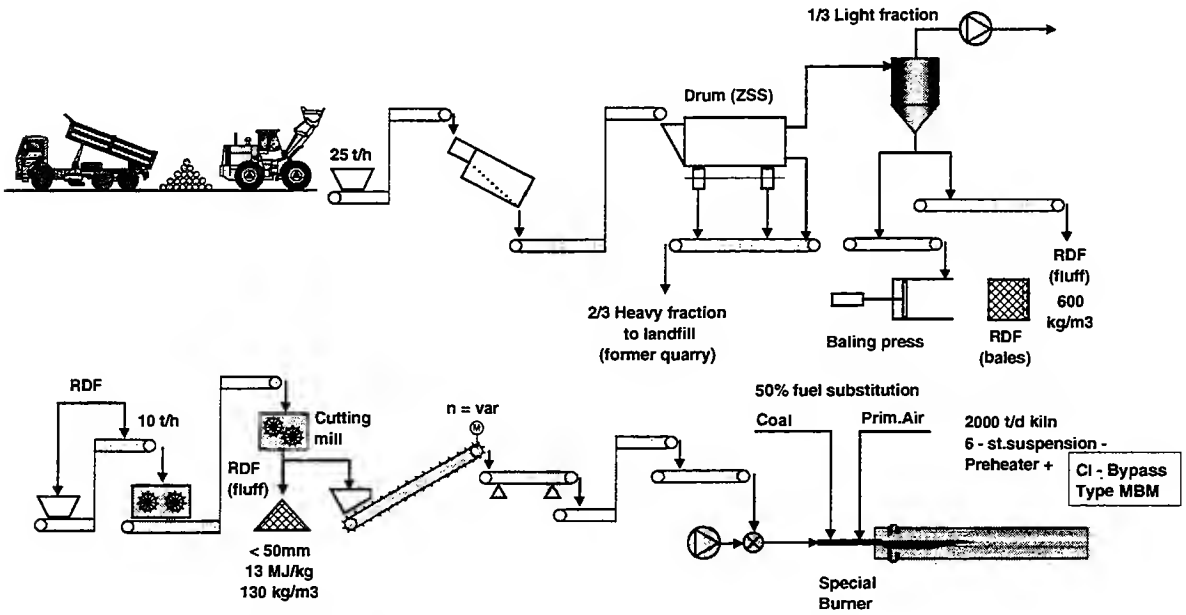
Fig. 14 Processing of Domestic Refuse and Incineration of RDF in a SP Kiln



Example of the 2000 t/d suspension preheater kiln of Wittekind in Erwitte (Germany)

6.3 Burning of Contaminated Waste Oil

The burning of waste oil in cement kilns has a long tradition. In the late 80's and early 90's new efforts have been made to investigate the influence of contaminants. Extensive measurement programs have been performed to demonstrate all possible impacts on the environment.

Important examples come from Germany and Austria.

One of the first plants to publish the results of its measurements in 1988 was Phoenix in Beckum/Germany. They burnt waste oil, which was contaminated with PCB (0 to 1000 ppm). Emissions of dioxins were also measured. This project was 50% sponsored by the German Umweltbundesamt (UBA).

The Lägerdorf plant and the Gmunden plant (Austria) have also followed this at an even higher degree of perfection. The design of the original Lägerdorf installation is shown in the figure 15. A large program of measurements has been done and published. This program also includes emission measurements of SO_2, NOx, heavy metals, F, chlorinated organics, PCB, Dioxins/Furans. It could be demonstrated that a limit for dioxins of 0.1 ng TE/m^3 (toxic equivalent) could be easily met and that these emissions are not influenced by burning contaminated waste oils. A similar result was obtained at Gmunden.

Special efforts were also made with regard to the delivery control, which meant a considerable extension to the existing laboratory. Limiting values for the waste oil in Gmunden:

Pb	<	5000 ppm
Hg	<	2 ppm
Tl	<	10 ppm
Cd	<	60 ppm
PCB	<	100 ppm
F	<	600 ppm
S	<	5%
Cl	<	1%
N	<	5%
H_2O	<	15%
Sediments	<	5%
CV net	>	25'000 kJ/kg

The original installation Lägerdorf according to figure 15 is designed for low flashpoints (< 21°C), therefore, use of solvents is also possible.

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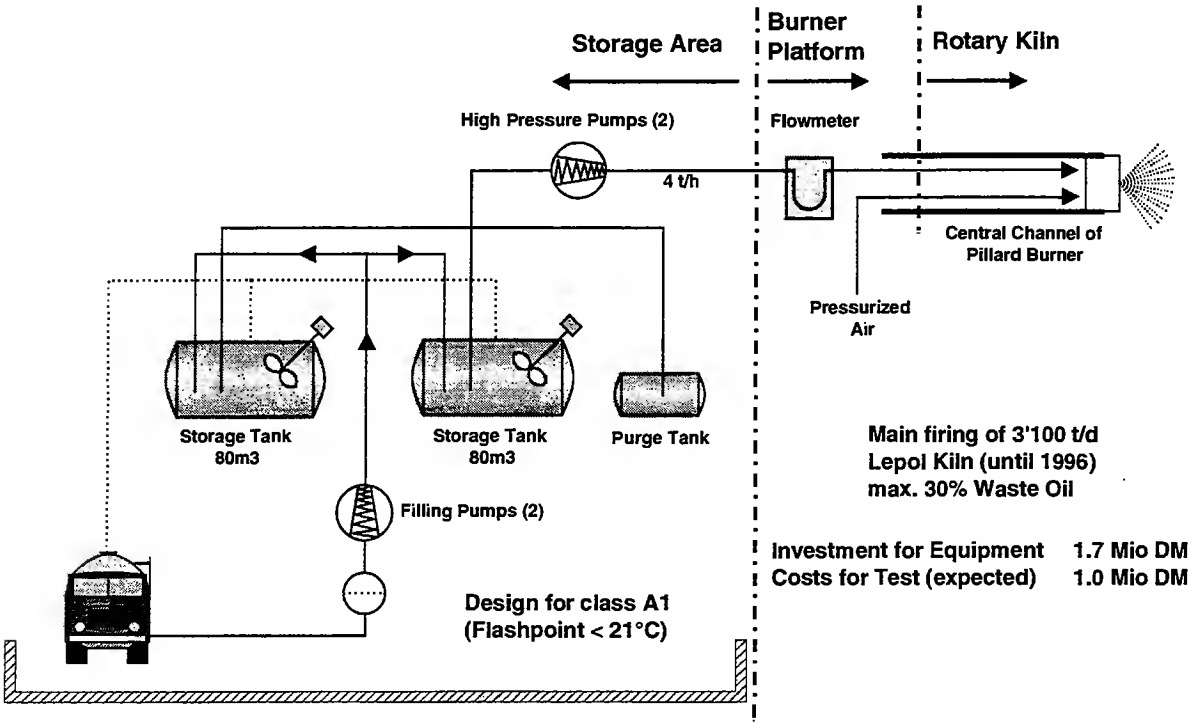
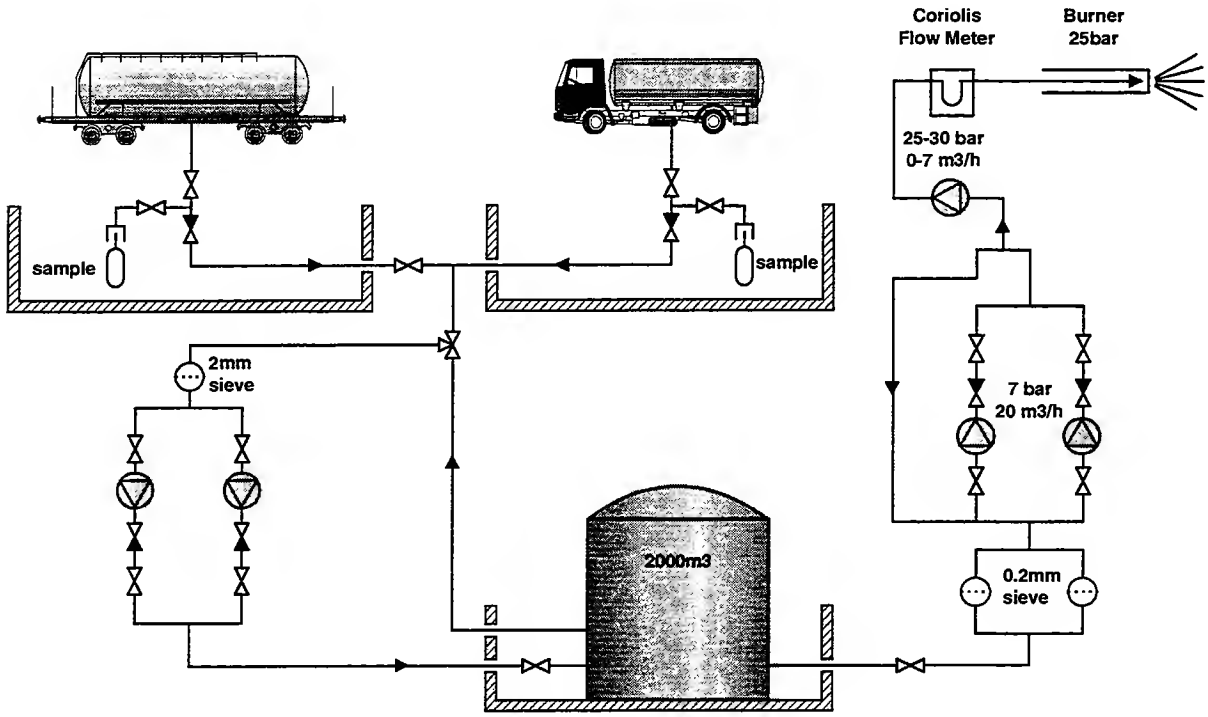
Fig. 15 Burning of Waste Oil at Lägerdorf Plant



6.4 Burning pure waste oil

The installation from Untervaz according to figure 16 results from a former heavy oil system and is adequate for high quality waste oil with high flashpoint (> 55°C).

Fig. 16 Handling of Waste Oil at BCU Untervaz



6.5 Burning of Waste Wood at Rekingen

This installation to process waste wood to a dry saw dust and burn it in a cement kiln went into operation in 1994. It was stopped in 1997 because the Rekingen plant was closed (market reasons). The final stage of this project would have been 70'000 t/a of processed waste wood or 50% fuel substitution. The treatment of the incoming wood consisted of delivery control, primary crushing, sorting fine grinding and drying.

The projected costs were 25 Mio. Swiss Francs and therefore among the highest ever realized for a single project for an AF. During the project phase the fees for waste wood were overestimated. When it came into operation, the actual market prices for waste wood were much lower and the installation could not be amortized.

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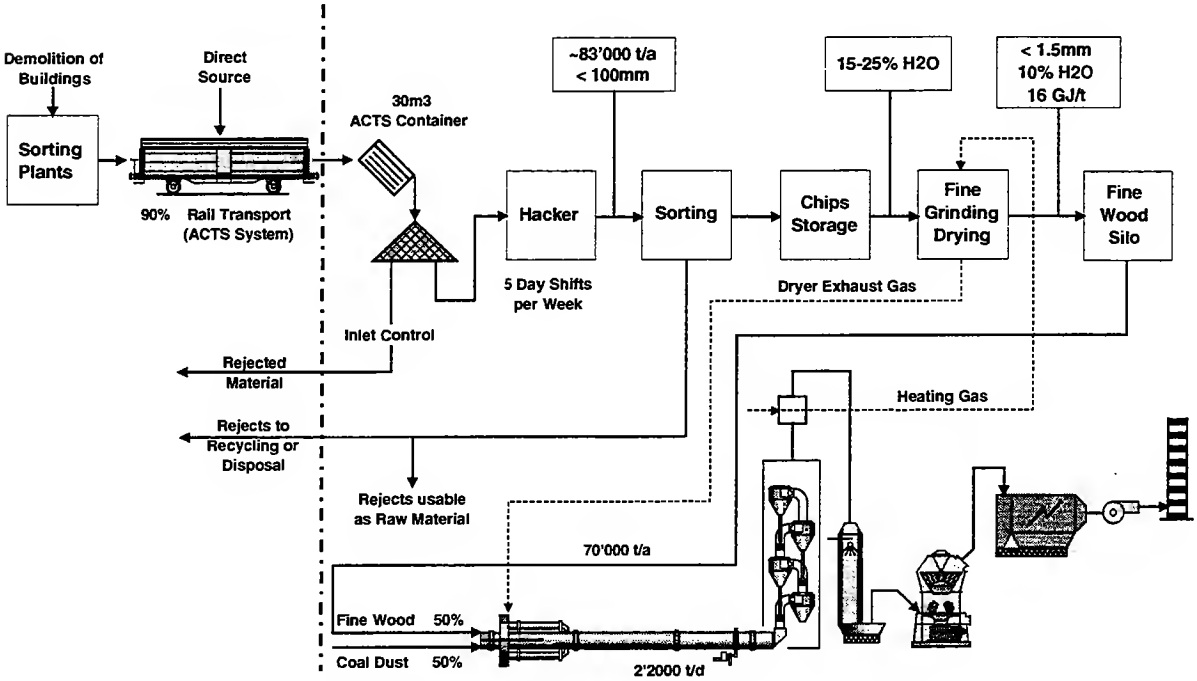
Fig. 17 Use of Waste Wood as Fuel at the Rekingen Plant



Note: Data as indicated refer to original project design

6.6 Mixed examples

The following illustrations originate from our reports describing practical AFR applications in our Group plants. The illustrations are self-explaining.

Content:

- Liquid AF at Altkirch
- Liquid AF at Obourg
- Distillation Residues and Animal Fat at Untervaz
- Mid kiln firing for bales at Obourg
- Dried Sewage Sludge and Animal Meal at Untervaz
- Solid AF (Impregnated saw dust) at Eclépens
- Tire Chips and Ferrocarbon at Lägerdorf
- Fly ash at Lägerdorf

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Fig. 18 Handling of Liquid Alternative Fuels (CSL) at Altkirch (Solvents, Oil-Emulsions, Pasty Liquids)

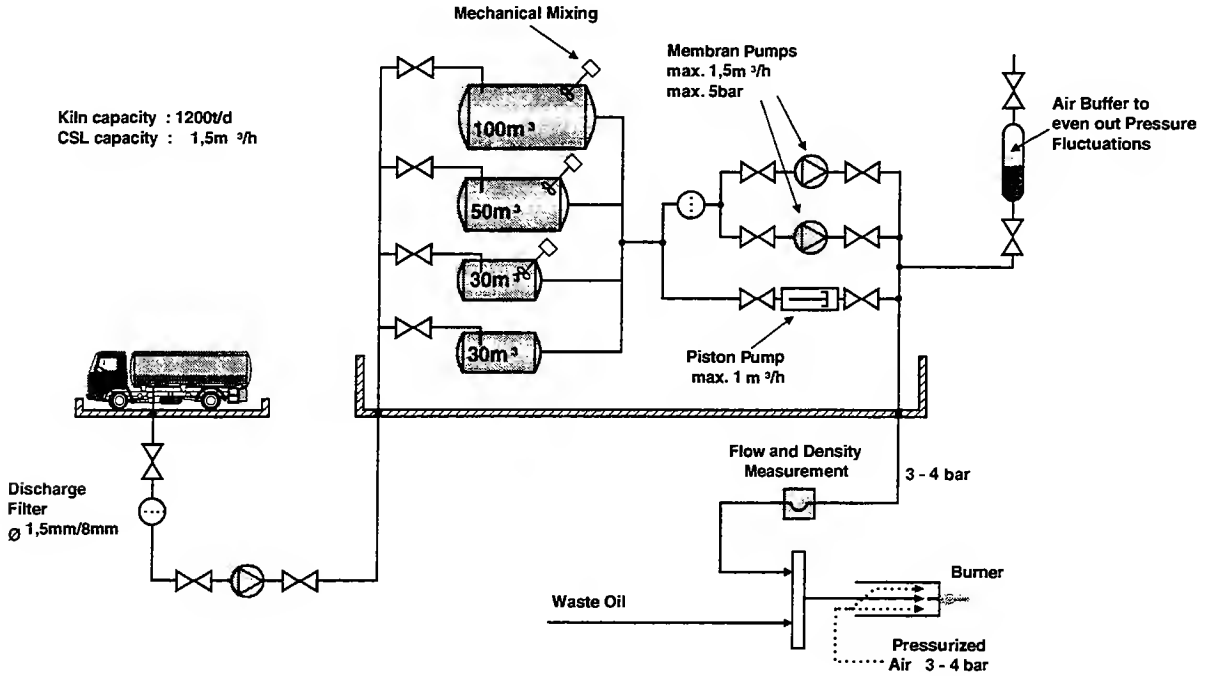

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Fig. 19 **Handling of Solvents (CSL: Combustible de Substitution Liquide) at Ciments d'Obourg**

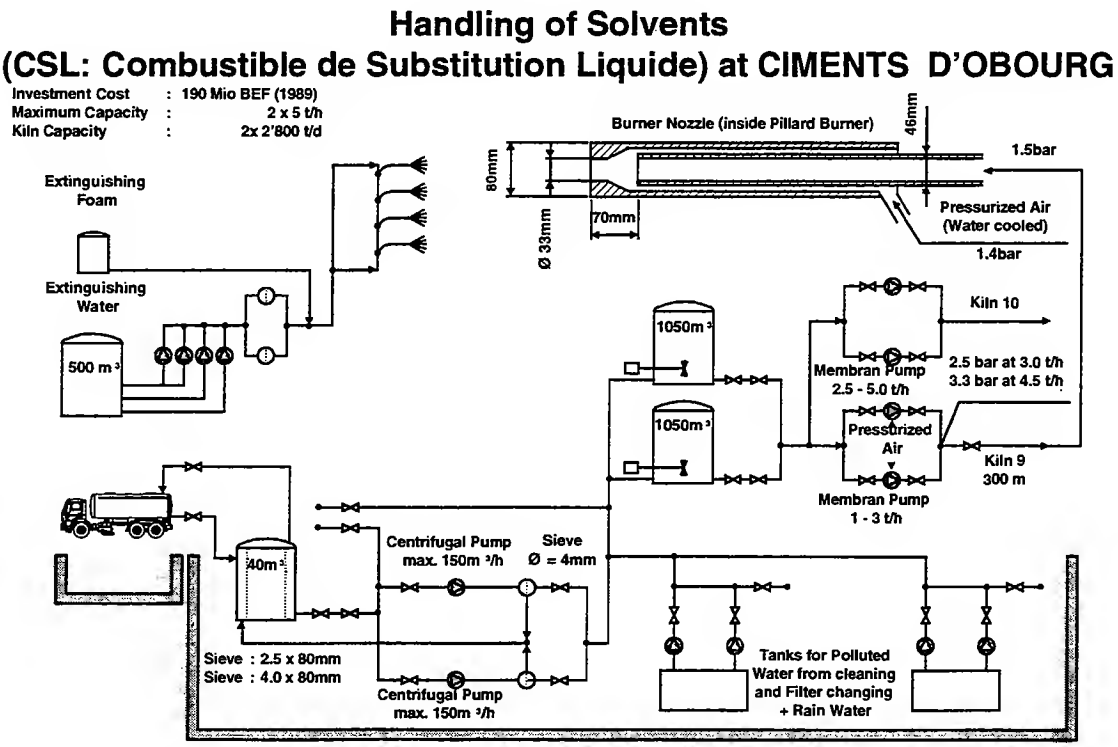

Fig. 20 Handling of Destillation Residue and Animal Fat at BCU Untervaz

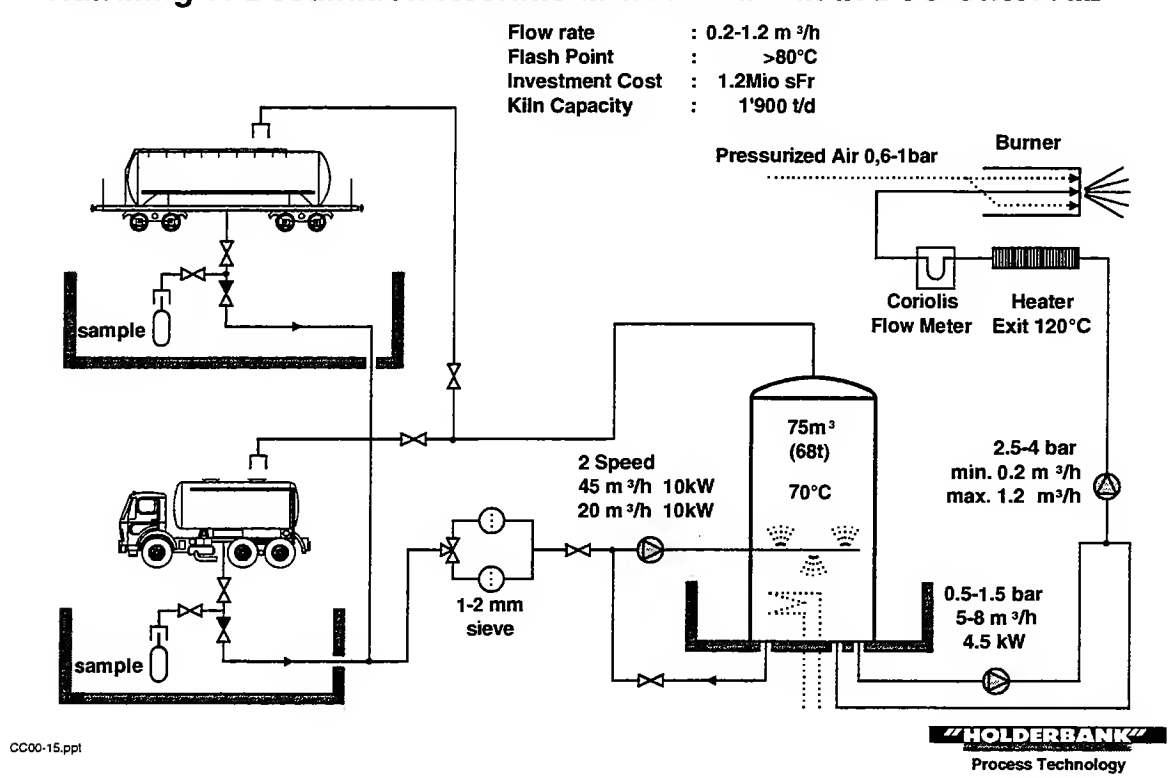

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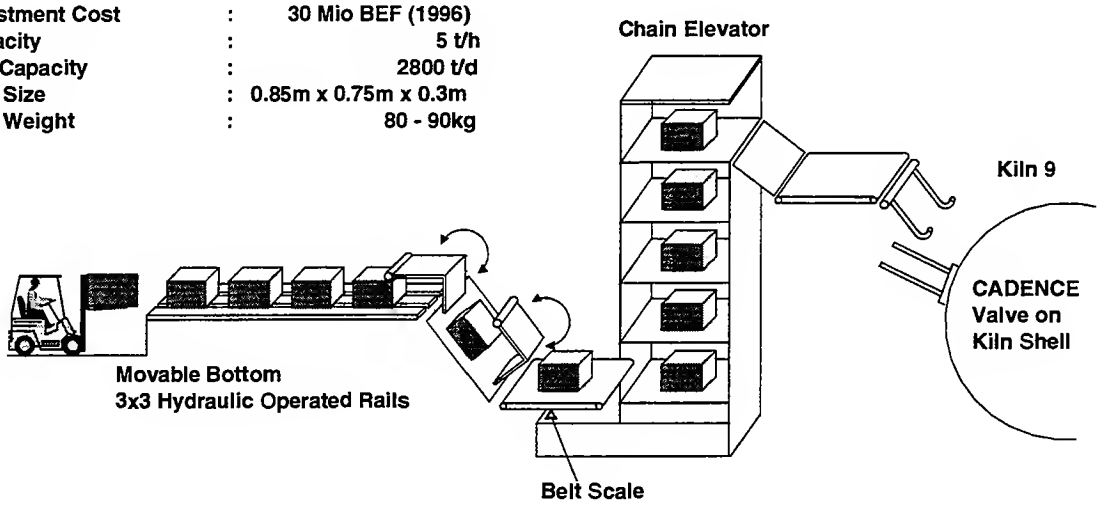
Fig. 21 Mid Kiln Installation at Ciments d'Obourg

Mid Kiln Installation at CIMENTS D'OBOURG

Investment Cost	:	30 Mio BEF (1996)
Capacity	:	5 t/h
Kiln Capacity	:	2800 t/d
Bale Size	:	0.85m x 0.75m x 0.3m
Bale Weight	:	80 - 90kg



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Process Technology

Fig. 22 **Handling of Dried Sewage Sludge and Animal Meal at Untervaz**

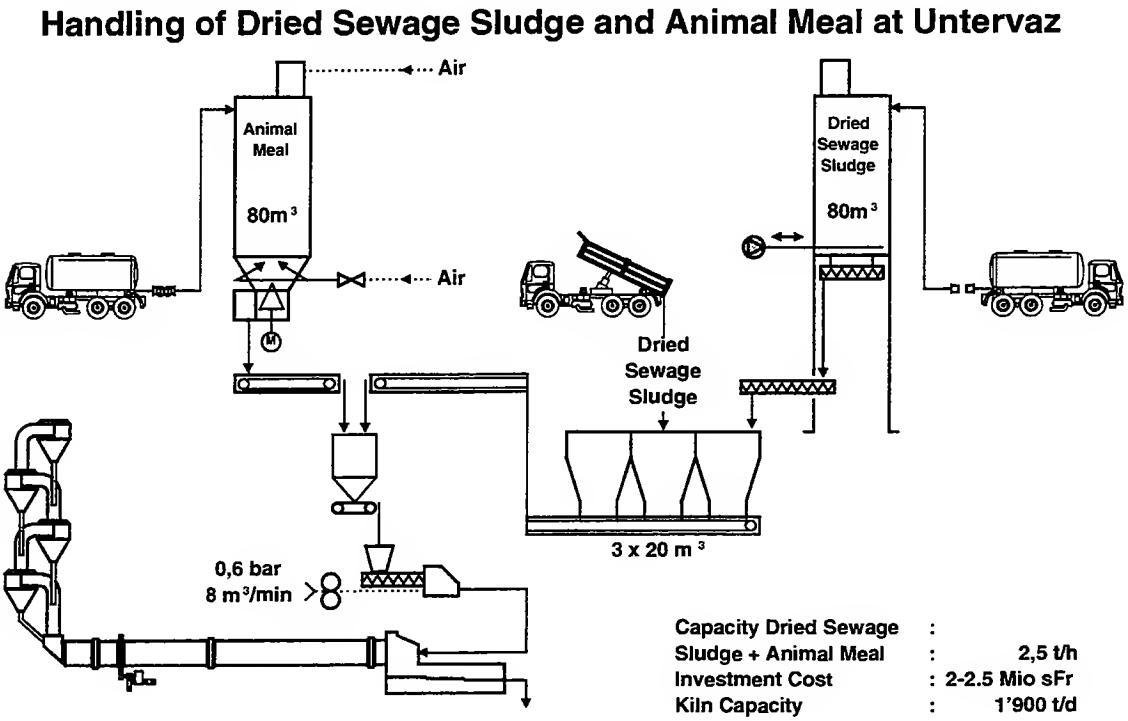

Fig. 23 **Handling and injecting of solid fuel at "HCB Eclépens (impregnated saw dust, shredded plastic, animal meal)**

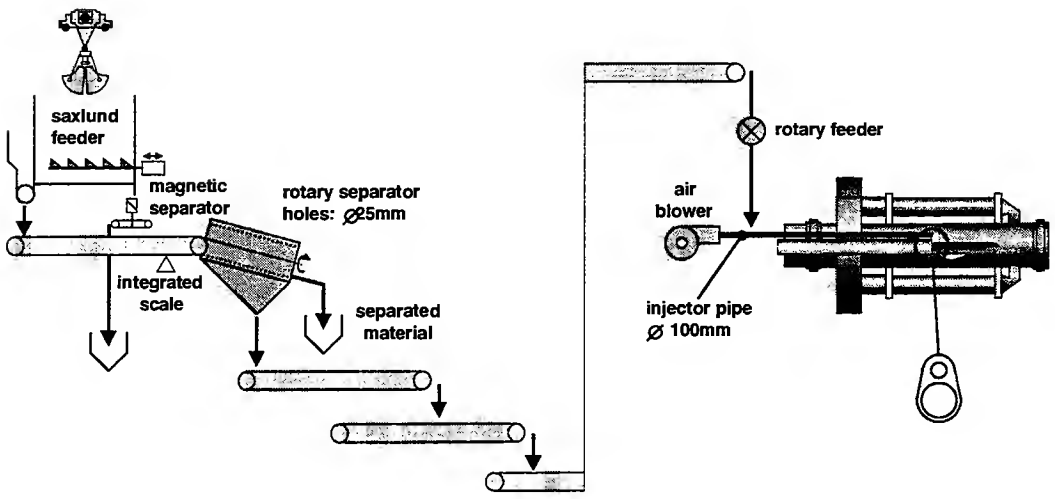

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Fig. 24 Handling of Tire Chips and Ferrocarbon at Lägerdorf

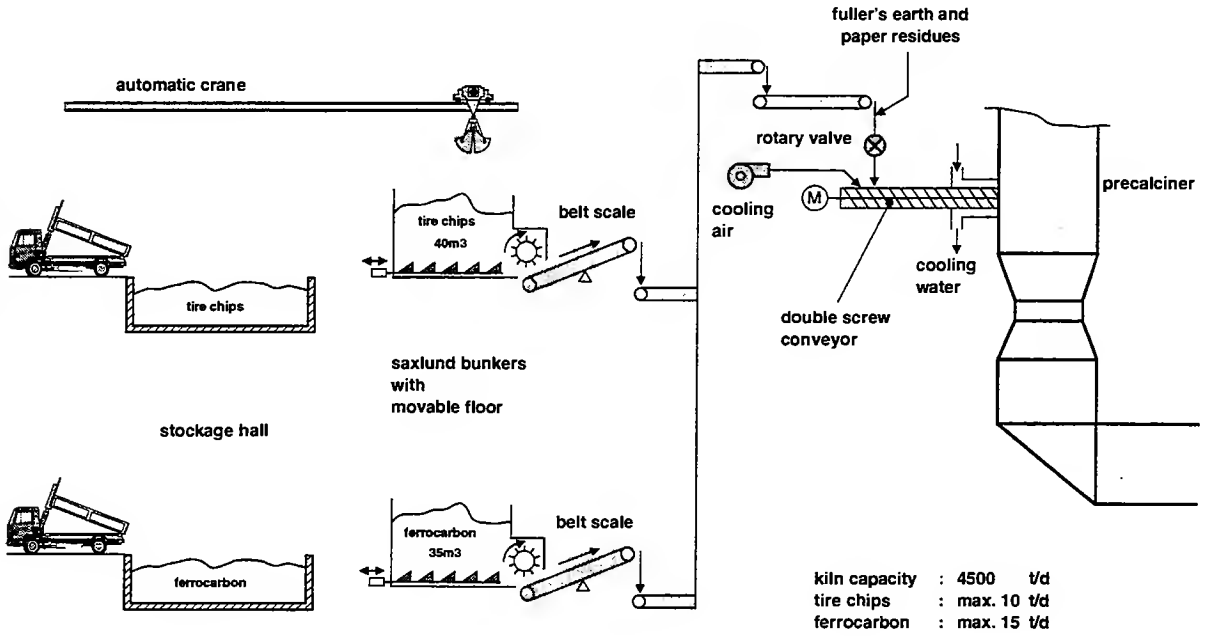
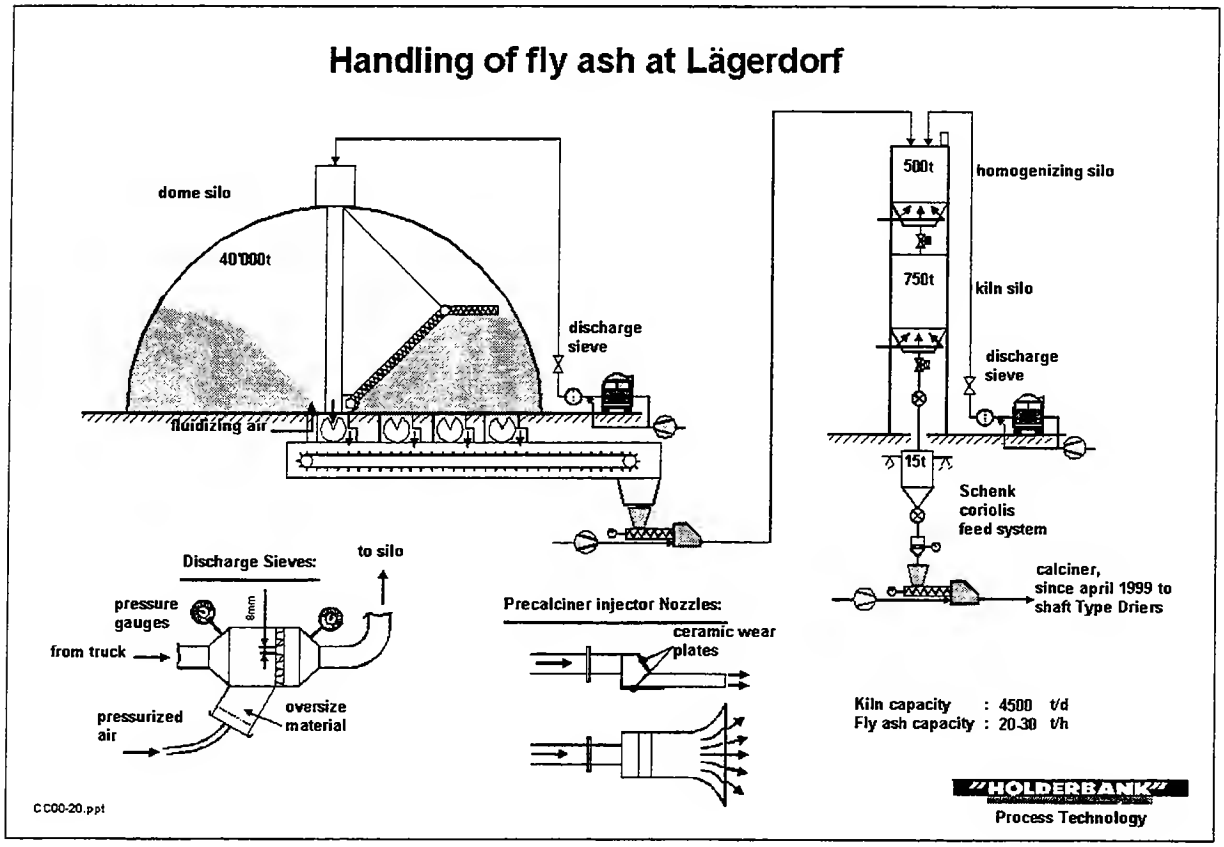


Fig. 25 Handling of fly ash at Lägerdorf



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Preparation of Alternative Fuels

A. Flacher
PT 00/14667/E

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SUMMARY

Alternative fuels used in the cement manufacturing process originate from waste, which often doesn't have suitable physical properties for the kiln process. This is where alternative fuel preparation is needed.

This paper focuses on common waste treatment techniques for alternative fuel production. For a complete understanding of the subject, the lecture of the Cement Course paper "Use of Alternative Fuels" is recommended.

1. INTRODUCTION

Waste is a residual product of some sort of process. Its physical properties are characterized by this process. Alternative fuel, on the other hand, is a combustible for clinker manufacture. It must have physical properties suited for incineration in a cement kiln. In many cases the properties of waste and alternative fuel do not correspond. This is where alternative fuel preparation is required.

2. FROM WASTE TO ALTERNATIVE FUEL

2.1 Waste Properties

When waste material is considered for alternative fuel use, first of all, its **chemical composition** (circulating elements, ash composition) must be suited for the kiln process. Then there are other properties and aspects, which need to be looked at:

- The **viscosity** of liquid waste determines the design of the storage and handling installation.
- Corresponding to the **granulometry** of solid waste, a size reduction process might have to be considered and the suitable kiln feeding point is chosen.
- **Water** contained in liquids requires mixing for better homogeneity of the alternative fuel. Water contained in solid waste has an impact on the handling as it might change the flowability of the material.
- **Foreign bodies and impurities** need to be separated in order to reduce wear and the risk of blockages.
- **Safety** is very important when dealing with waste, e.g. **toxicity** and **flashpoint** need to be considered.
- Additional aspects refer to the waste supply, such as **annual tonnage** as well as **form and schedule of delivery**.

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2.2 Processing of Waste

As for any other process, for the preparation of alternative fuel the most simple and inexpensive but also safe process is suited best. A mechanical waste treatment is therefore chosen in almost any case.

Similar to the mechanical processes in cement manufacture the preparation of alternative fuel comprises of the following main processes:

- **Screening** to separate foreign and oversized material which might cause handling or wear problems in fuel preparation and firing installations.
- **Size reduction** to fit the fuel to the foreseen firing point (e.g. preheater, main burner), as well as to allow a proper dosing and to ensure a good combustion.
- **Homogenizing and mixing** to feed to the kiln process a fuel of constant quality.

Picture 1 gives and overview of the different processes from waste to alternative fuel. Explanation is given in the following chapters.

Picture 1: Processes from waste to alternative fuel

Thermal waste processing (gasification/pyrolysis) is applied in a few cement plants. It is operated in conjunction with the kiln. Gasification allows separating the combustible fraction of a waste in a gaseous form from the non-combustible part. The investment cost for such installation is comparably high, which makes this process worthwhile only when considerable disposal fees are received.

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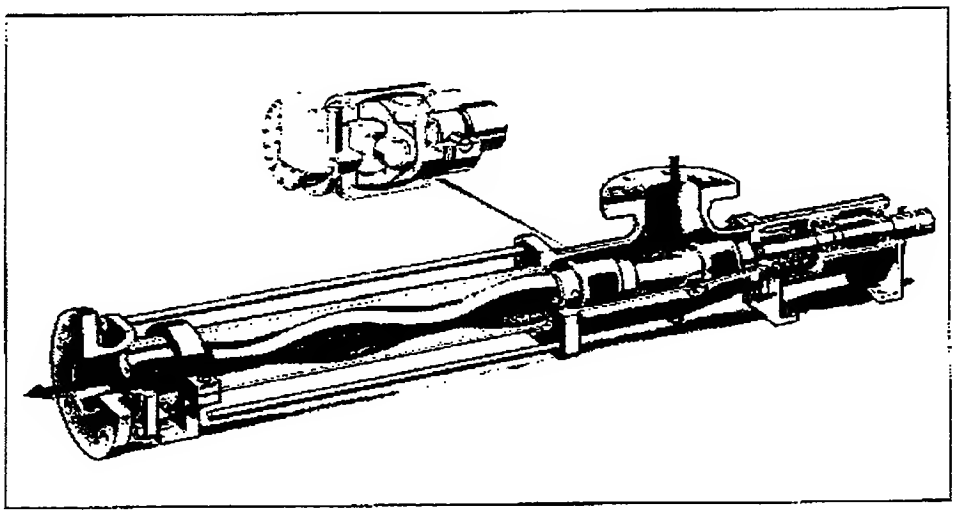
3. LIQUID ALTERNATIVE FUEL PREPARATION

In many cement plants, liquid waste has been the first waste material to be accepted and used as alternative fuel because little preparation is necessary and, in certain cases, existing installations may be used. However, there are special aspects that need to be considered when handling liquid waste such as waste oil and solvents.

3.1 Pollution of Liquid Waste

Liquid waste is likely to be polluted with foreign bodies (metal pieces, sand, plastics etc.). A coarse filtering upon reception is therefore necessary. With in-line shredders and grinders remaining particles can be crushed to avoid blockages. Even after a second filter in the kiln firing line, there are still solid particles present in the liquid. The pumping and kiln injection system must therefore be designed accordingly.

Picture 2: Screw pole pump, suited for waste oil containing solid particles



3.2 Homogenizing

Due to the presence of solids and often also mixtures of liquids with different densities, a mixing tank is required. A recirculation circuit or mechanical mixer ensures a good blending.

3.3 Safety

The handling of solvents or liquid waste containing solvents represents a safety risk since there is a high potential for explosion. The decisive parameter for the safe design of a liquid installation is the liquid's flashpoint (the flashpoint is the temperature at which the evaporations of a combustible liquid form an inflammable gas). Pure motor oil has a flashpoint of up to 200°C whereas other liquids, such as solvents, can have one below 0°C.

Another safety risk to workers and the environment represent toxic substances. Special protection and safety procedures are required. The guideline for this should always be the safety standards applied in the chemical industry.

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Picture 3: Protected worker sampling liquid waste

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4. PREPARATION OF ALTERNATIVE FUEL FROM PASTY WASTE

In industrial areas there are often considerable amounts of pasty or sludgy waste materials available, such as

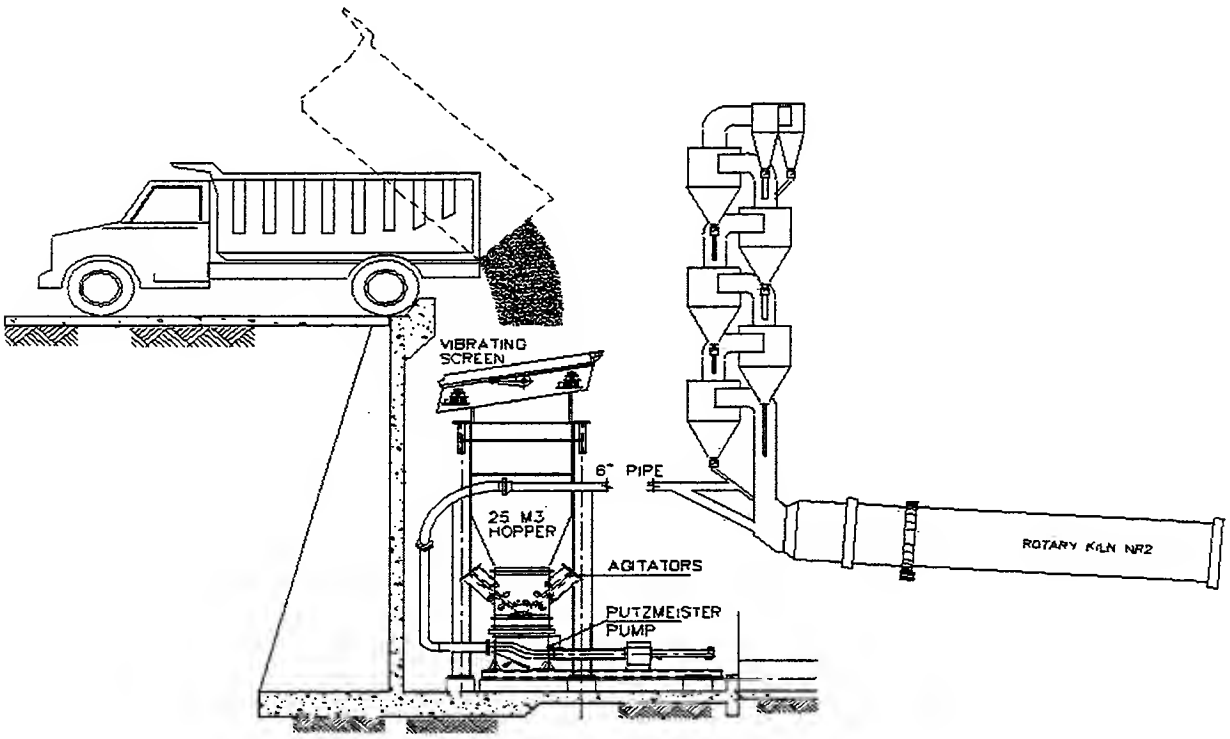
- resin, paint, varnish
- oil sludges from tank cleaning
- destillation residues
- grease, soap.

Many of these materials are hazardous. With regard to the variable clinker production cost, they can therefore be of particular interest as considerable disposal fees may be received.

From the point of view of incineration, it is important that such waste is burnt at high temperatures with sufficient residence time (The combustion criteria for destruction of halogenated waste for example is 1'200 °C with 2 seconds residence time). This is only ensured at the main burner of a cement kiln. Therefore, the hazardous, pasty waste must be pre-treated for incineration in the primary firing.

There are two common ways of sludge pre-treatment for incineration in the primary firing, see below. For mainly non-hazardous sludges there is the possibility of kiln inlet feeding. With such application, the sludge is screened to take out coarse foreign bodies before being pumped to the kiln by means of a concrete pump. Sludge feeding at the kiln inlet is limited due to the high water input.

Picture 4: Sludge feeding installation to kiln inlet (Apaxco plant)



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4.1 High Viscosity Liquid Facility

A so-called high viscosity liquid facility aims to liquefy the sludgy waste. This is achieved by mixing the sludge with liquid fuel. The important aspect of this process is the compatibility of the substances involved. To avoid chemical reaction, sludge and liquid need to fulfill certain requirements. This however, limits the use of such pre-treatment system to certain substances, which is a drawback with regard to flexibility.

4.2 Impregnation Facility

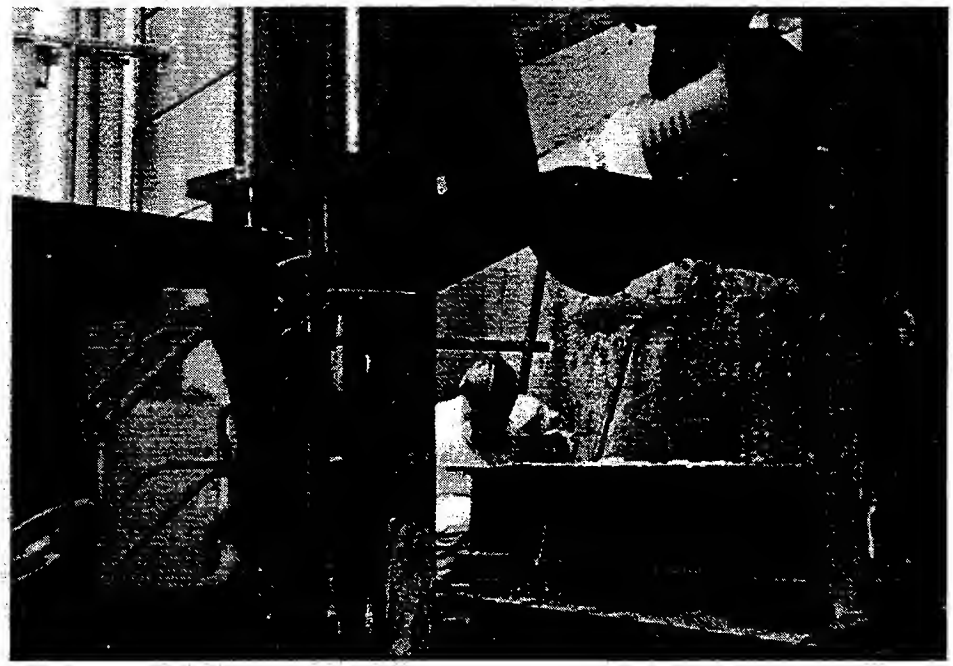
The other common sludge pre-treatment process aims to prepare a pourable, fine, solid alternative fuel. This is achieved by mixing the sludges with an impregnation support. Saw dust is commonly used for this purpose. It has a good absorbency and the impregnated product shows little stickiness. Alternatively, filter cakes and also animal meal have been used as impregnation aid.

In what follows the different steps of an impregnation facility are explained.

4.2.1 Reception and Preparation of Primary Materials

The sludgy waste is delivered either in bulk or small containers (typically 200 l steel drums) and emptied into pits. There are drums, which cannot be emptied by gravity or reasonable manual effort. For such containers a robust shredding facility is needed.

Picture 5: Drum emptying



According to chemical analysis of the incoming waste and the product requirements, the unloaded materials are mixed and then shredded. The prepared mix is conveyed to a mixing station by means of a high pressure pump.

For an optimum product quality and minimum use of impregnation aid the viscosity of the batch mix is adjusted. If necessary, liquid phases are extracted from the drums or added to the mixing pit.

4.2.2 Production of Impregnated Alternative Fuel and Dispatch

Once there is a batch of sludge prepared it needs to be mixed with sawdust. The mixing ratio is about one third of saw dust and two thirds of sludge.

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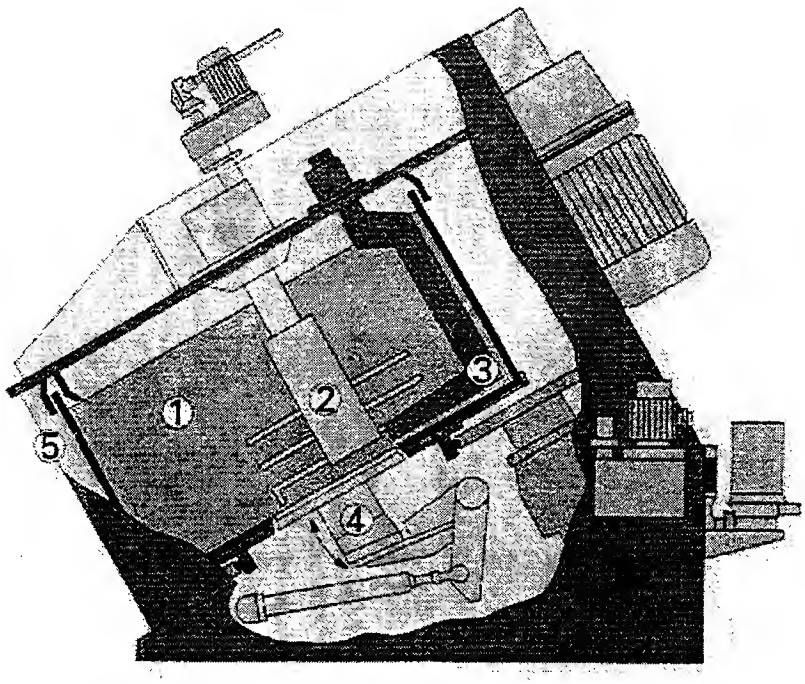
For a simple installation, it is possible to mix the components by means of a wheel loader. The more sophisticated process is to use a continuously operated intensive mixer.

Legend:

1: mixing container
2: mixing tool
3: material guiding arm
4: discharge opening
5: housing

Picture 6: Intensive mixer (type Eirich)



Before storage, metals are separated and the product is screened in a drum screen. The product is fine (mostly – 10 mm) and well suited for incineration at the main firing.

For storage moving floor type silos are best suited.

4.2.3 Organic Emission

Due to the handling of unconfined chemicals there is a lot of organic emission. This requires an appropriate protection of the workforce on the one hand. On the other hand, an aspiration and thermal treatment of the emission might be necessary.

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5. SOLID ALTERNATIVE FUEL PREPARATION

Solid waste can be fed in a coarse form to the kiln (e.g. whole tyres) but often needs to have a finer granulometry. A size reduction is necessary and determined mainly by the following process factors:

- transportation cost
- handling properties
- kiln feeding point
- thermal substitution rate
- kiln behavior.

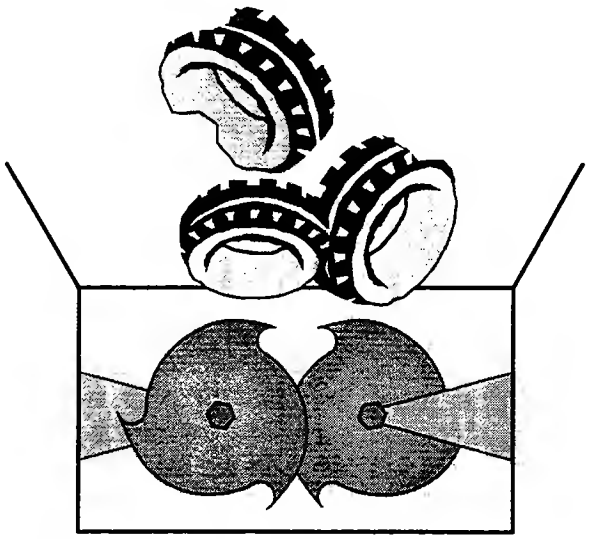
The mechanical processes used for the size reduction of solid waste are shredding and granulation. They are typically applied for materials such as wood, rubber, plastics or waste tyres.

5.1 Shredding and Granulation of Solid Waste

5.1.1 Shredders

A shredder is a machine for primary size reduction of solid materials. Its operating principle is based on a tearing and shearing action of slowly rotating tools (10 – 40 rpm) agitated by a high torque drive.

Picture 7: Typical shredder design with two counterrotating shafts



Typically, a shredder consists of two counterrotating shafts. On these shafts engaging disks with hook type knives are mounted to grab and tear the material fed. Such shredder is preferably used for the size reduction of high-density waste (e.g. wood and tyres). Waste materials with a low bulk density such as plastic films require a shredder with large feed opening and a mechanism to push the material towards the cutting rotor. For such application single rotor shredders are used.

Depending on the knife size and operation (multiple stage, closed circuit) a minimum chip size of 20 – 50 mm can be achieved.

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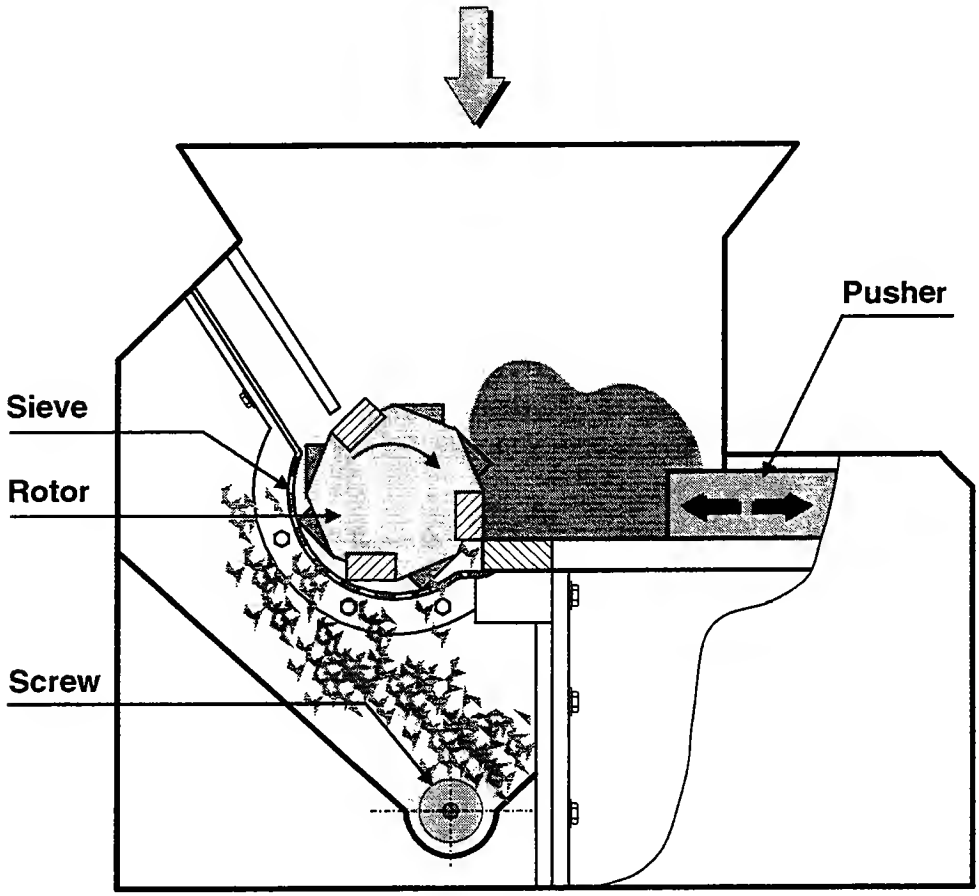
5.1.2 Granulators

Granulators (also called grinders) are generally used for secondary size reduction. The typical design consists of a single rotor with a fixed counter knife and an outlet sieve. There is a mechanism to push waste material towards the rotor where knives grab it. After being cut at the fixed counter knife the waste chips are further milled before they pass the outlet sieve. A granulator operates at higher rpm than a shredder, above about 100 rpm.

Picture 8: Typical granulator design with one shaft and outlet sieve



Due to their high rotor speed, granulators are used for low abrasive material. Waste containing considerable amounts of metals and/or mineral material should not be granulated because of excessive wear. The spacing between the rotor knife and the fixed counter knife needs to be small (about 0.5 mm) to efficiently cut fine material like plastic films.

5.1.3 Multi-Stage Size Reduction

For the firing of waste materials at the preheater/precalciner, a single stage size reduction is sufficient in most cases. If bulky waste material is to be fired at the main burner a multiple stage shredding and granulation is required for the efficient production of an alternative fuel of a few millimeter size.

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6. EXAMPLES

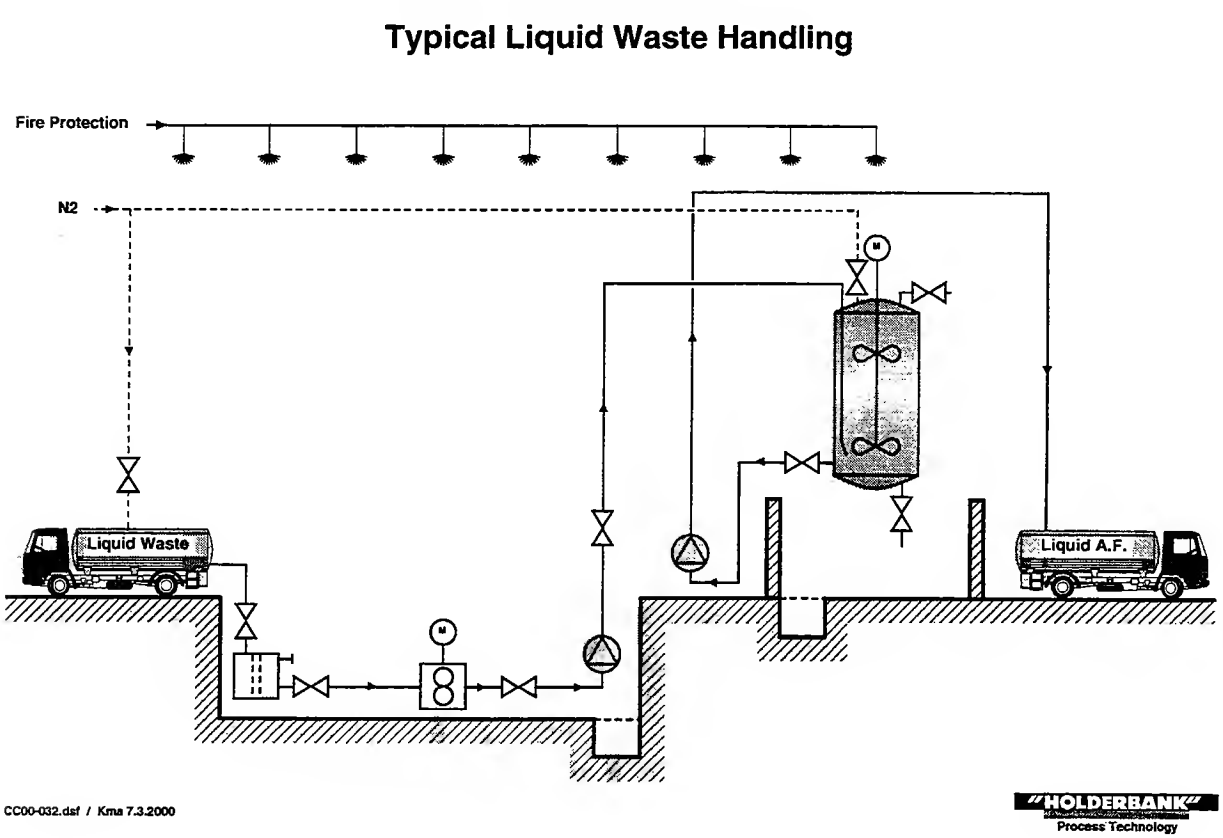
6.1 Typical Liquid Waste Handling

A typical handling facility for the reception of liquid waste and the production of liquid alternative fuel is shown in picture 9.

The installation consists of the following main elements:

- Liquid waste decanting tank with primary filtration
- Shredder to crush oversized particles for protection of the pump and to avoid blockages.
- Discharge pump, typically of centrifugal type
- Liquid storage tank with mechanical mixer
- Loading pump, typically of centrifugal type
- In case of low flashpoint liquid, a nitrogen inertisation system
- Sprinkling system for fire suppression

Picture 9: Typical liquid waste handling installation



6.2 SCORIBEL Impregnated Sawdust Production

SCORIBEL is a Belgium based subsidiary of Ciments d'Obourg. At the impregnation workshop of its Seneffe plant solid, pasty and liquid waste is mixed with an impregnation aid to form a fine, solid alternative fuel for incineration at the main firing of cement kilns.

The installation consists of three main parts:

- A reception section for industrial waste in bulk or containers as well as the impregnation aids sawdust and filter cake. In a closed building the liquid, pasty and solid waste is premixed and shredded.
- By means of a continuously operated intensive mixer the waste sludge is mixed with the impregnation aid before being screened.
- In two moving floor type silos the product is stored and homogenized before being loaded to trucks.

For reduction of organic emission air is aspirated at the main sources of emission and sent to a thermal treatment unit.

The plant's design production capacity is 15 t/h, actually it is operated at more than 20 t/h.

Investment cost for the impregnation facility (excl. air treatment): BEF 250 Mio.

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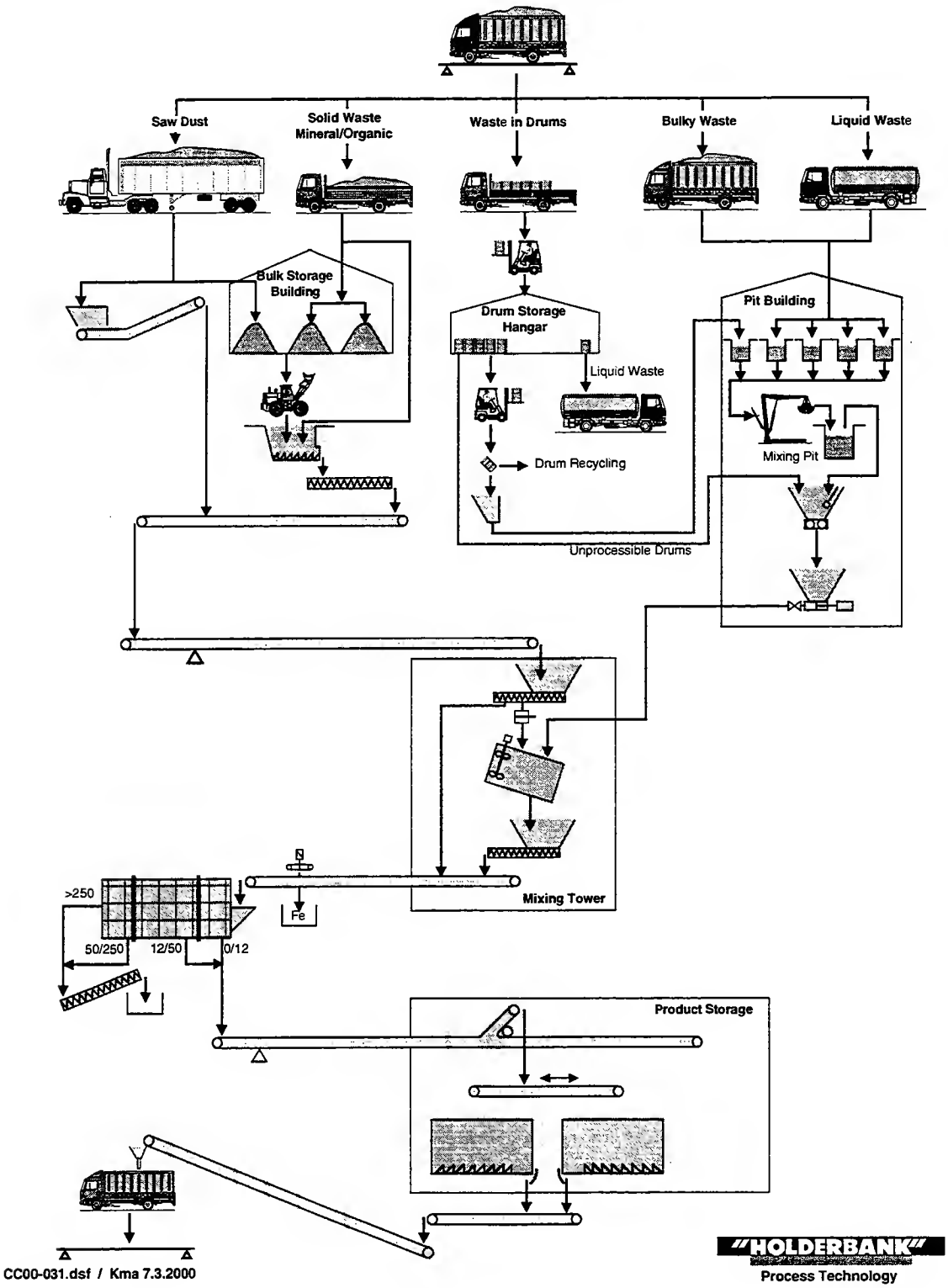
Picture 10: flowsheet of SCORIBEL's impregnation workshop



6.3 Industrial Plastic Waste Shredding at PLASTREC AG

PLASTREC AG, a subsidiary of HCB, operates a facility for the shredding of industrial plastic and rubber waste to a product for use at the primary firing of cement kilns. The waste materials received are cables, textile reinforced plastic, residues from manufacture of hard plastic items, plastic films and rubber.

The facility is designed to mechanically process the incoming waste to a product of 10 mm size. This is achieved by pre-shredding and subsequent grinding of the material. Two shredders and two granulators, each operated in parallel, are used for the two-stage process. Material transport is done by belt conveyors for raw and pre-shredded material, and pneumatically for the finished product.

The plant has an hourly production rate of 1.5 – 2.0 t. The annual production is about 5'000 t. The consumption of electrical energy is 100 kWh/t for the whole facility.

Investment cost for whole installation: CHF 1.2 Mio.

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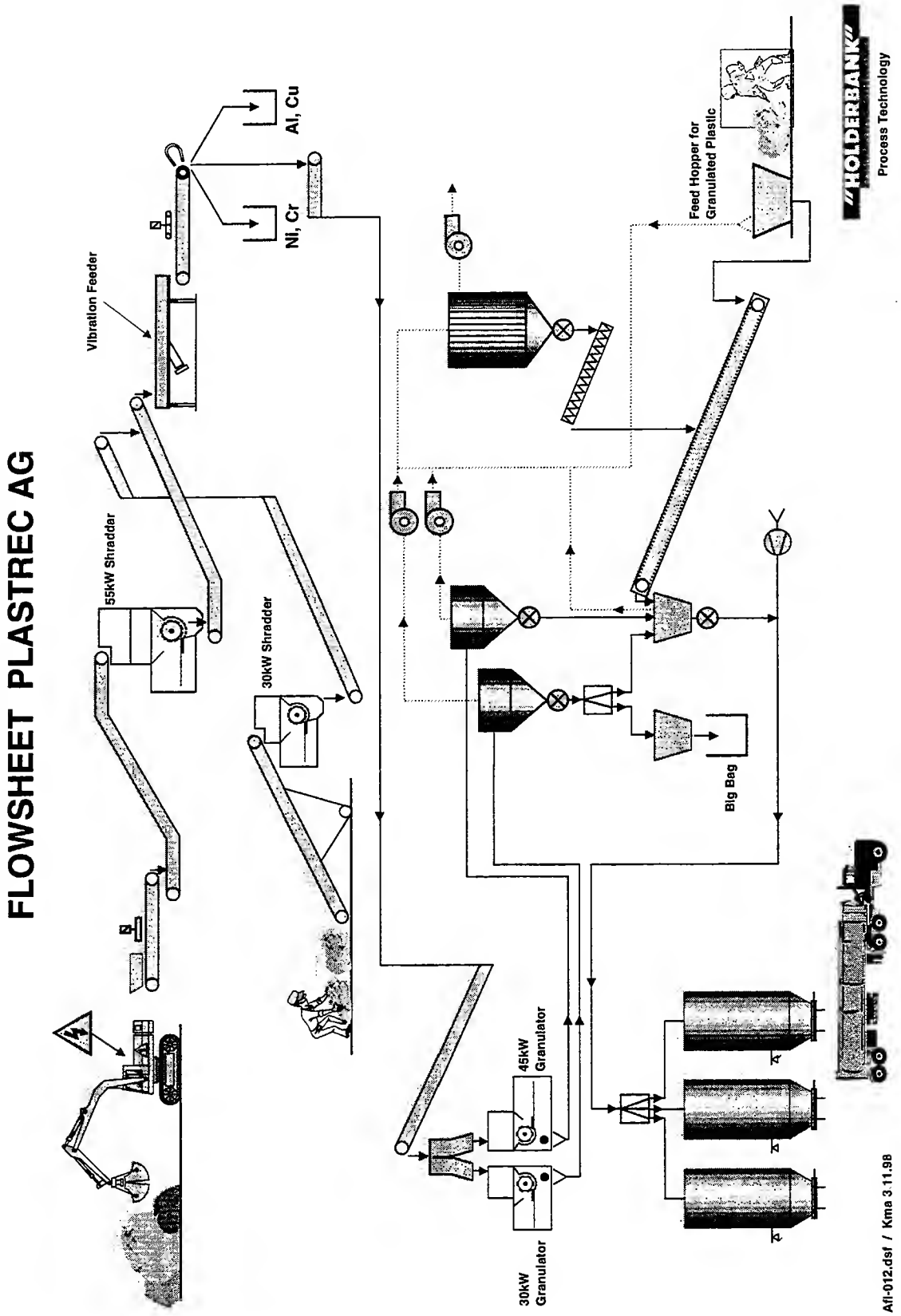
Picture 11: flowsheet of PLASTREC plastic shredding plant



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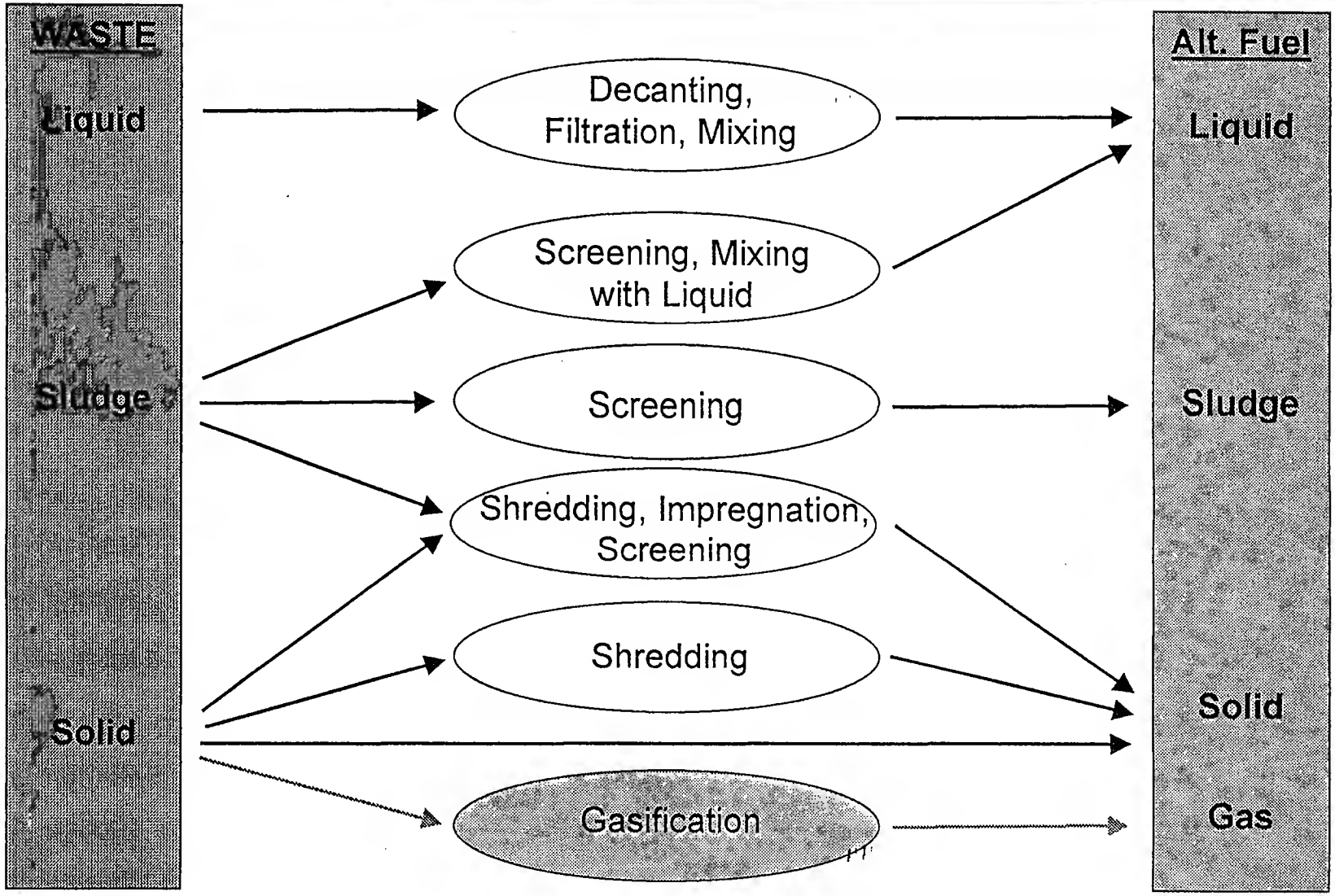
Physical Waste Properties

Alternative Fuel preparation is required to change the waste's physical properties.

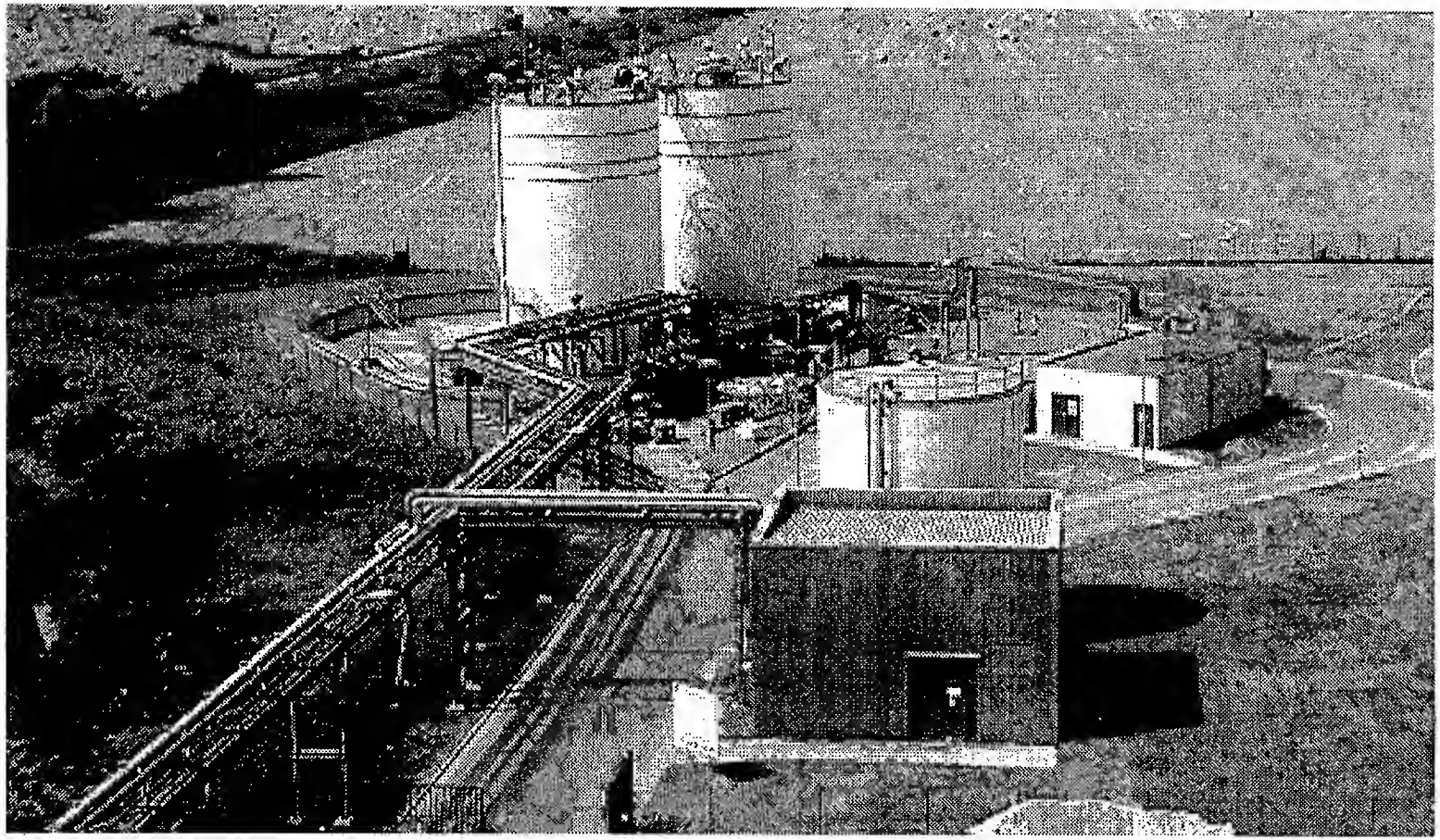
- Granulometry (Size)
- Foreign Bodies/Substances
- Water Content
- Non-Homogeneity
- Viscosity

From Waste to Alternative Fuel



Liquid Waste

Waste Oil - Solvents - Waste Water



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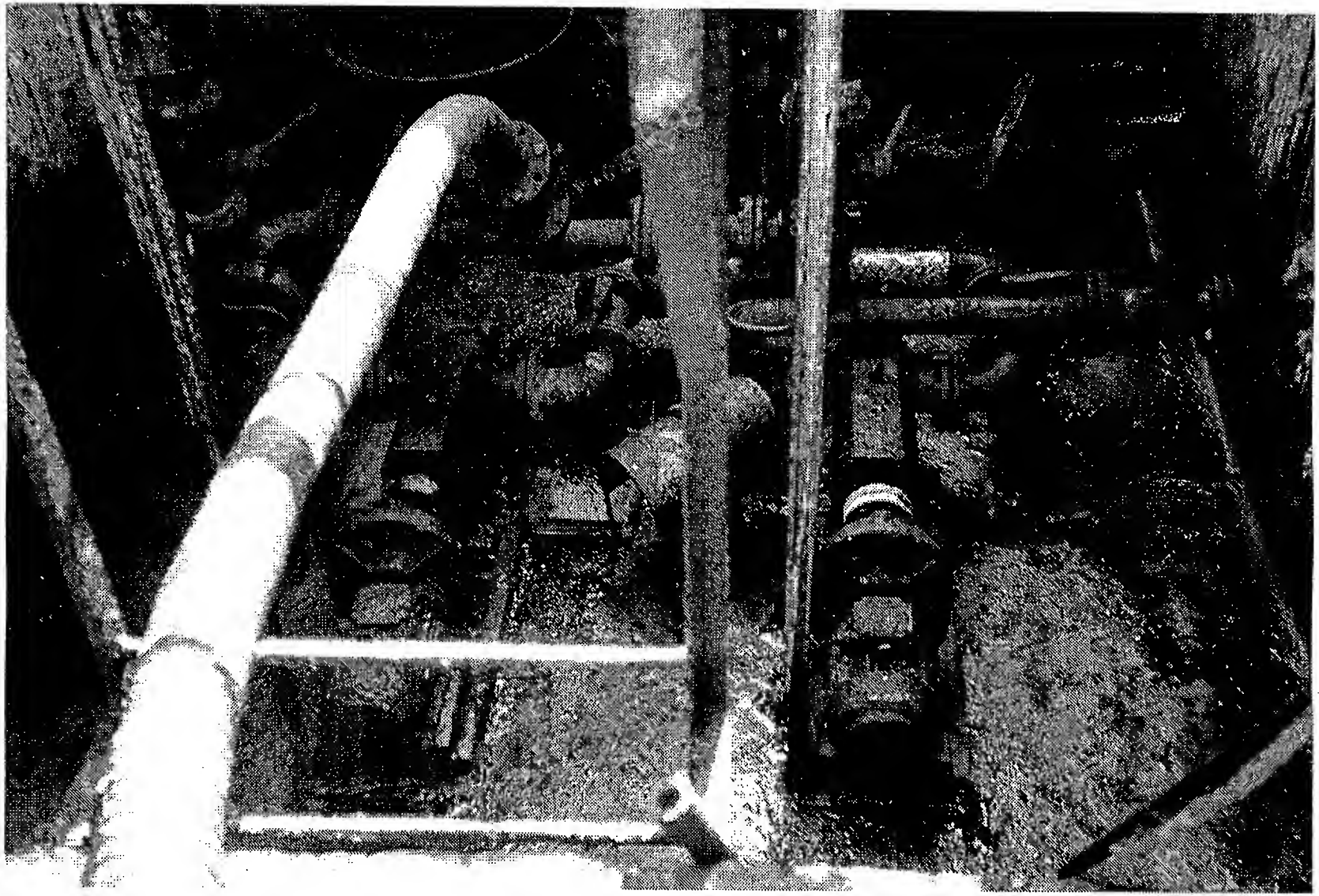
Liquid Waste - Handling Aspects

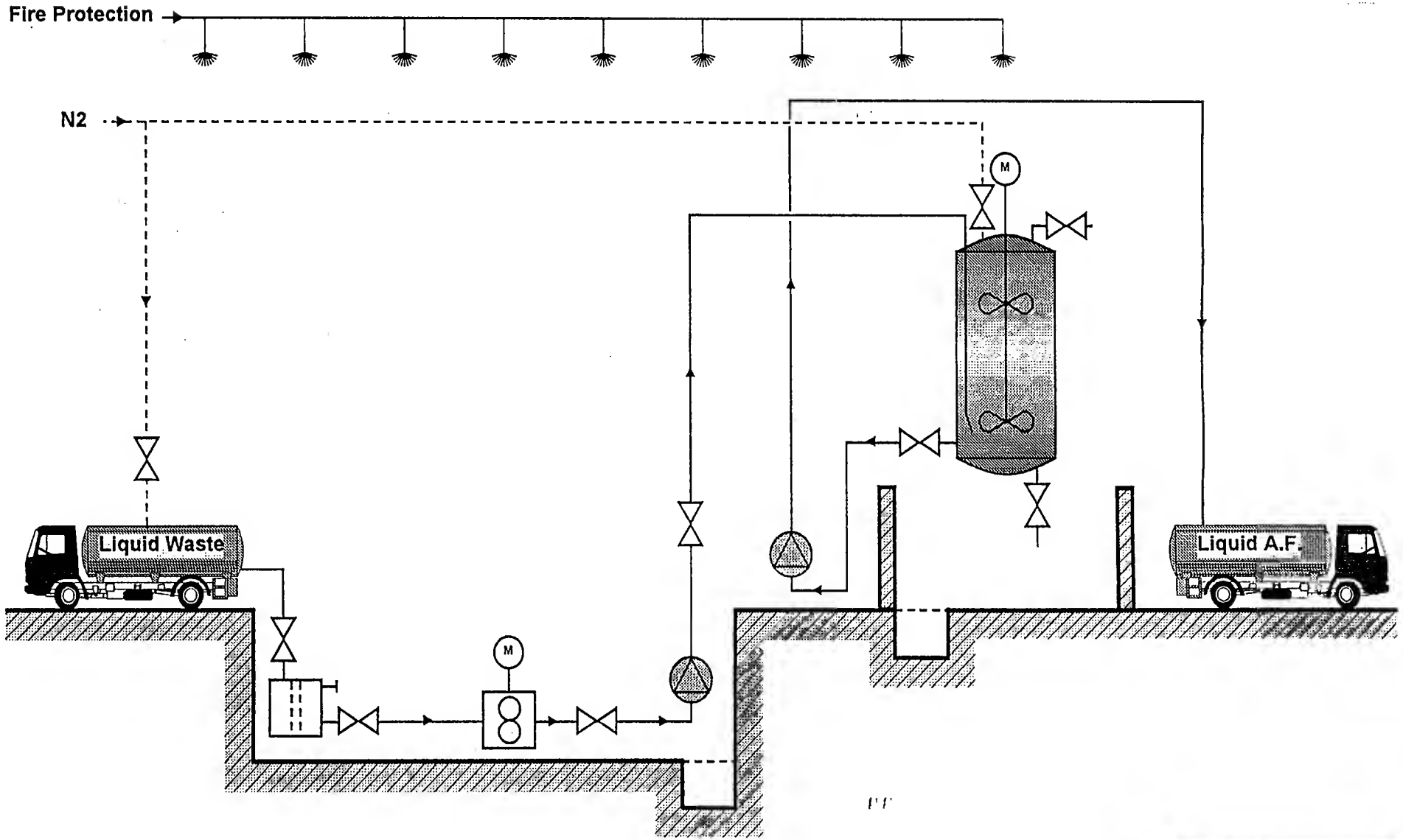
Liquid waste can be...

- polluted with solids (metals, minerals, plastics, ...)
  - decanting, filtering, grinding
  - proper selection of pump
- inhomogeneous
  - mixer, tank recirculation
- hazardous
  - protection of personnel
  - safety of installation

Flashpoint: temperature at which the evaporations of a combustible liquid form an inflammable gas.



Typical Liquid Waste Handling



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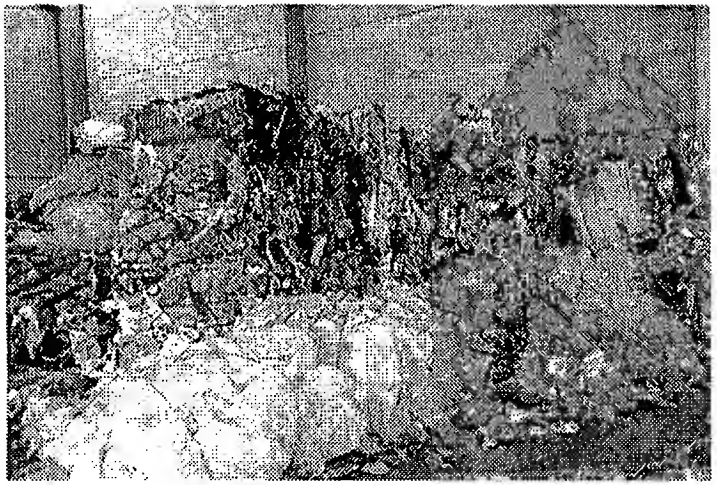
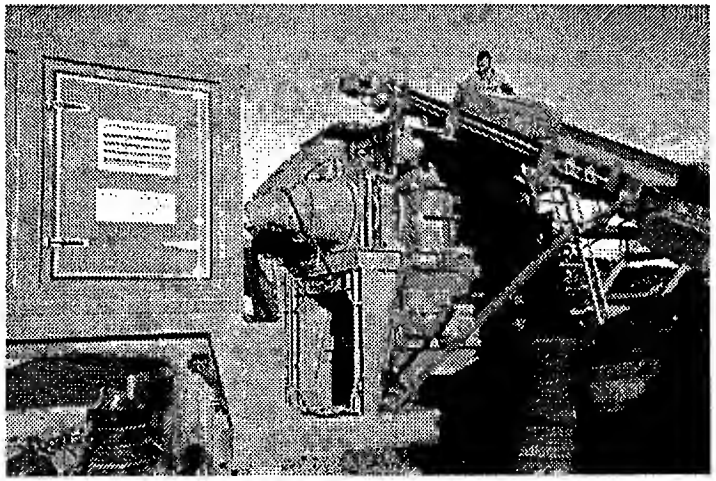
Liquid Waste - Safety

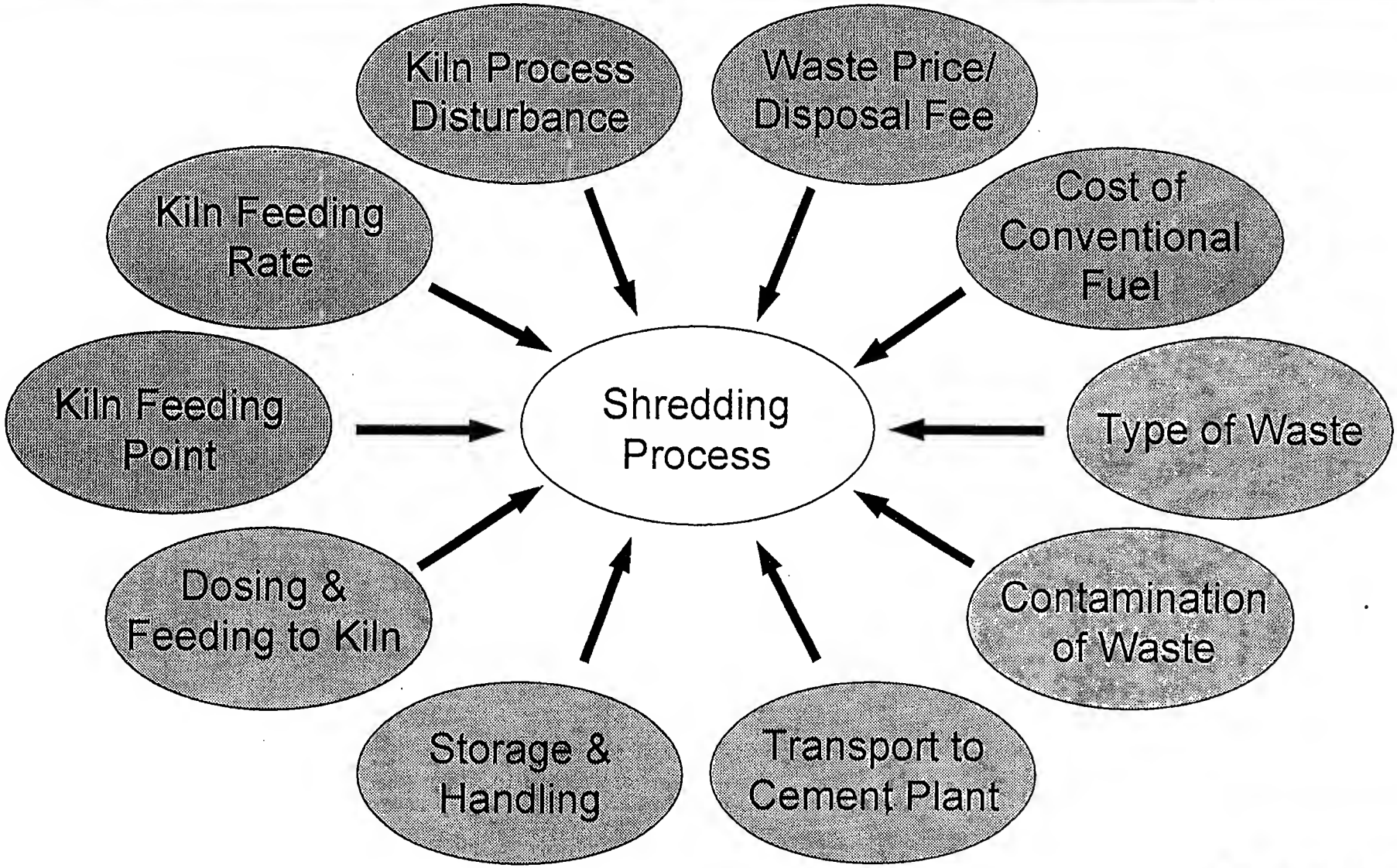
- Safety of installation:
  - Design of mechanical & electrical installation according explosion risk
  - Inertisation & explosion suppression (N_2, CO_2, ...)
  - Fire fighting facility (detectors, water/foam supply, sprinkling installation, fire fighting water basin)
- Protection of personnel:
  - Education, Instruction
  - Physical protection (overall, boots, gloves, masks with active carbon filter)

Waste Shredding



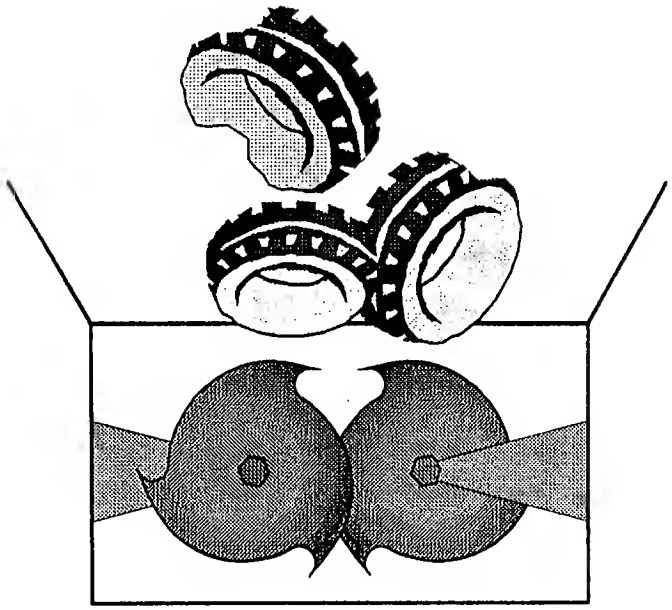


Waste Shredding - Selection Criteria



Coarse Shredder

- robust design
- low rpm drive (< 40 rpm)
- high torque drive (electric or hydraulic)
- typically two shafts



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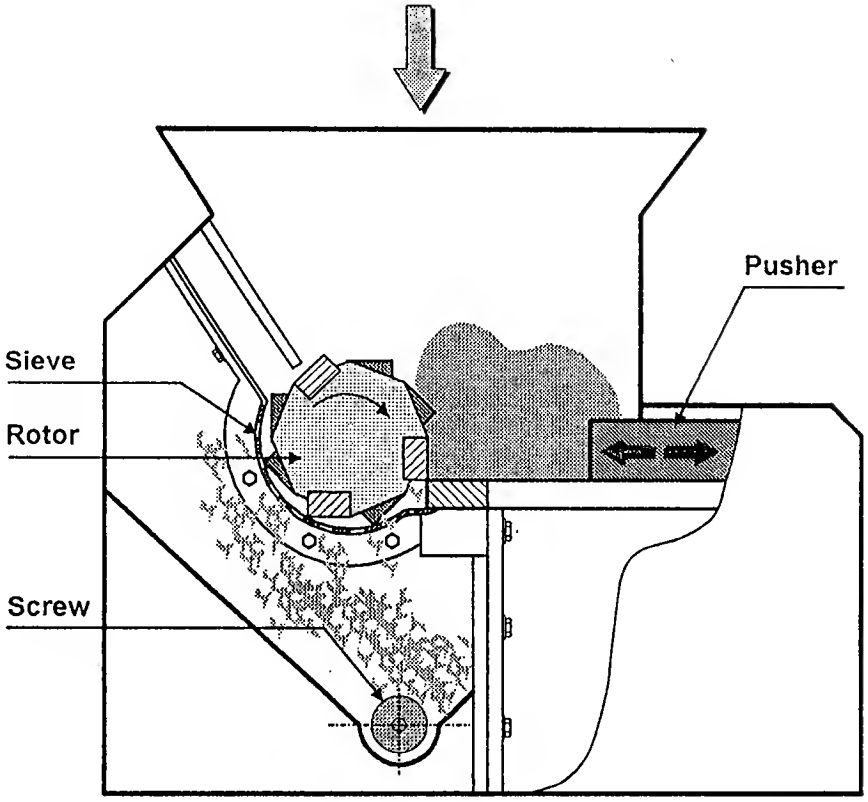
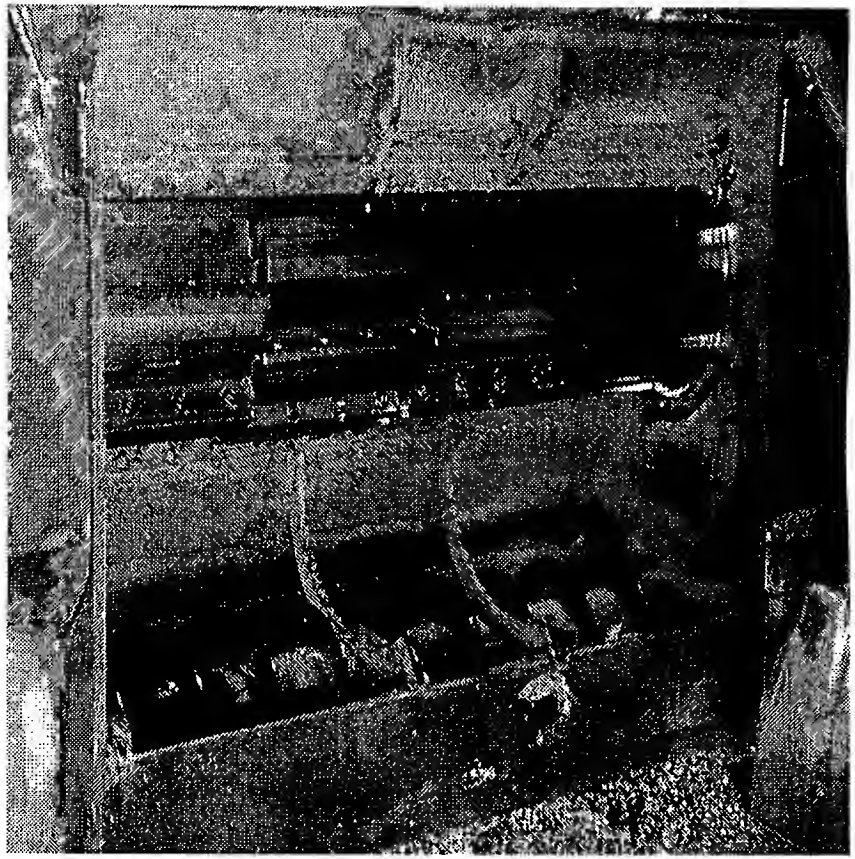
Fine Shredder

- secondary size reduction
- high rpm drive (> 100 rpm)
- sensitive to foreign bodies and abrasive material



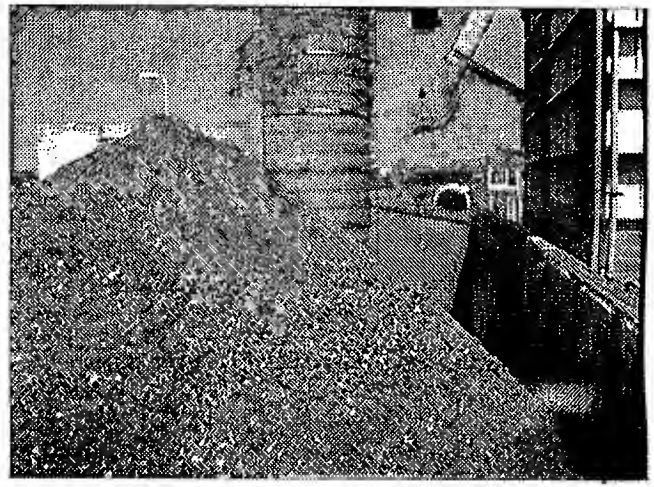


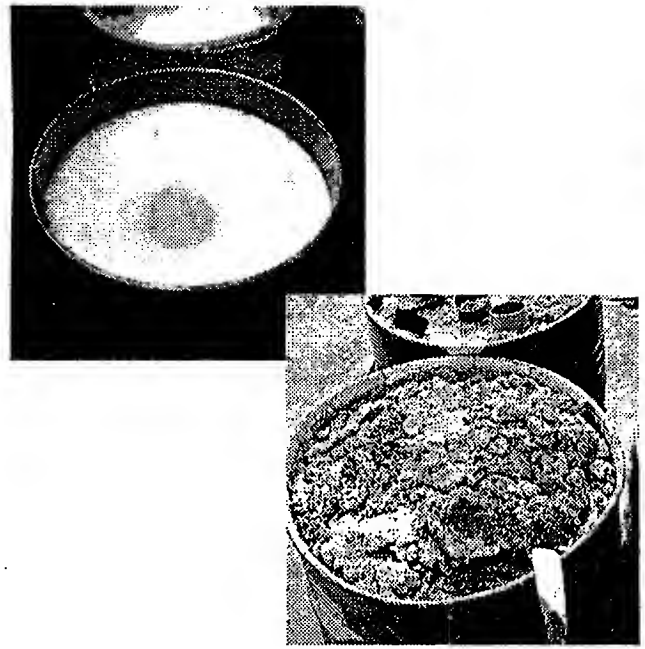
Coarse & Fine Waste Shredding

	Coarse shredding	Fine shredding
Kiln firing point	PH/PC (main burner)	main burner
Suited waste	tyres, plastics, wood, drums, ...	clean plastics, low abrasive material
Process	single stage	multiple stage
Chip size	< 150 mm	films < 30 mm granules < 5 mm
Contamination, Foreign bodies	visual check usually sufficient	metal separation, low mineral content
Power consumption	≈ 40 kWh/t (tyre)	≈ 100 kWh/t (plastic)
Operating cost	30 – 50 USD/t	depending on contamination
Versatility for other waste	yes	no

Sludge Treatment - Impregnation

Production of fine solid alt. fuel from pasty/solid waste and impregnation aid





by means of:
- sludge premixing
- coarse shredding
- impregnation
- screening
- homogenisation

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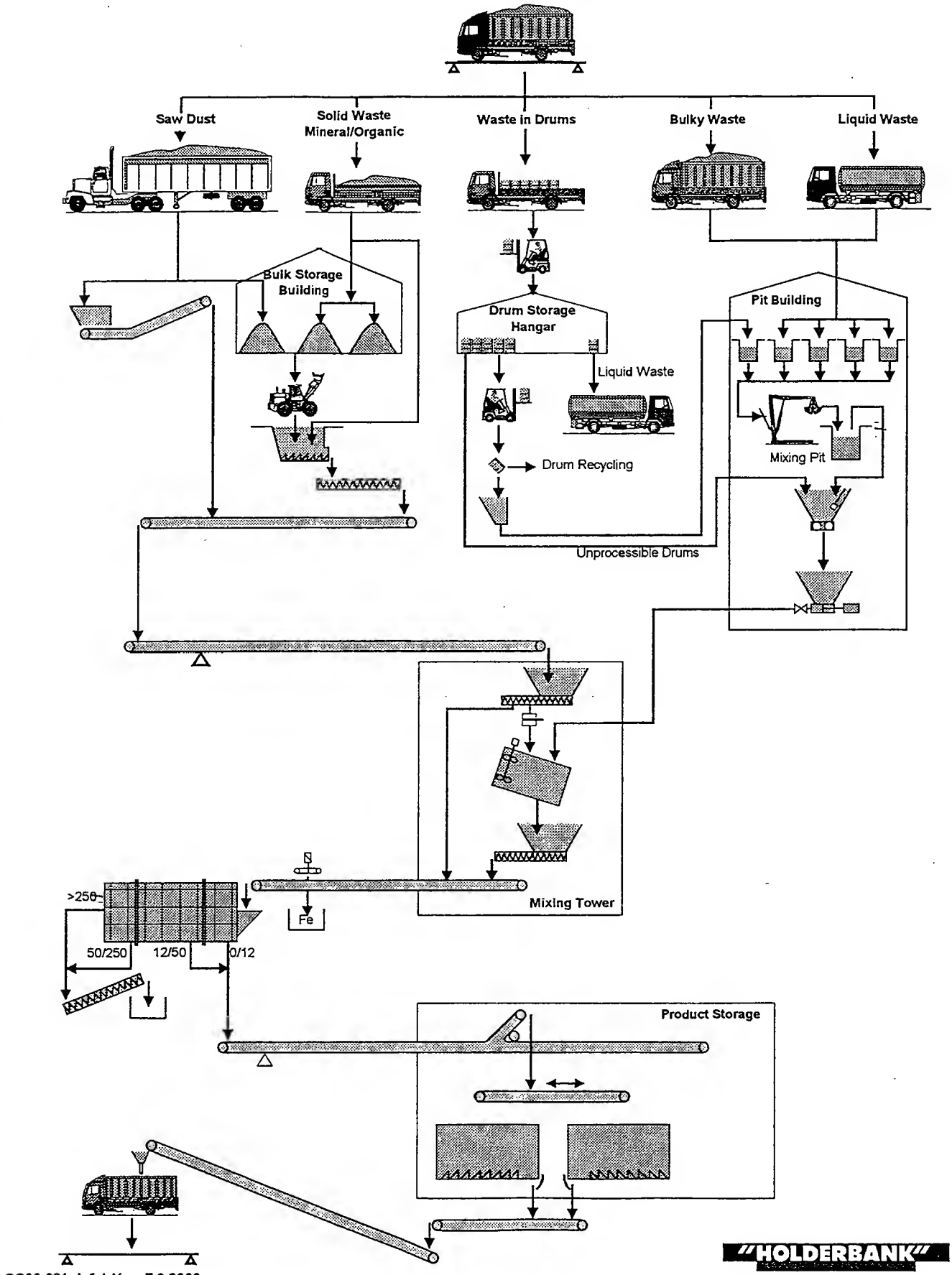
Impregnation Platform - SCORIBEL S.A. (1)

Features of SCORIBEL Impregnation Platform

- **Input materials**
  - contaminated earth, resin/paint, sludges of ink glue varnish oil, soap, ...
  - impregnation aid: sawdust, filter cake, animal meal
  - not accepted are: explosives, radioactive material, pathological substances, pharmaceutical waste, materials exceeding toxical limits
- **Products**
  - fine, solid AFR for main burner firing
    size: <12 mm of which 80% < 5mm
    density: 0.4 - 0.6 t/m^3
    PCI: ± 10.5 MJ/kg
- **Quality control of incoming waste and product**
  - incl. heavy metal and halogen analysis

"HOLDERBANK"

SCORIBEL - Impregnated Saw Dust Production



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Process Technology

Chapter 8

Circulation Phenomena

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Circulation Phenomena in the Clinkerization Process

René Hasler, Daniel Brassel
PT 99/14503/E

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SUMMARY

This chapter describes the circulation of volatile elements in the kiln system. It indicates the tolerable inputs of circulating elements so that no excessive build-up and clogging problems arise.

In particular it shall serve as guideline how an encrustation problem, caused by volatile elements, is systematically solved.

1. INTRODUCTION

Alkali, sulfur and chlorine compounds (hereafter called circulating or volatile elements, see below) in raw materials and fuels utilized for the cement manufacture, when present in high concentrations often given rise to difficulties in kiln operation with build-up formation, mainly in the preheater and the kiln inlet section.

Volatile Elements (VE):

Sulfur	SO_3
Potassium	K_2O
Sodium	Na_2O
Chlorine	Cl

The build-up formations cause cyclone blockages or restrictions of the kiln inlet area so that the kiln has to be stopped for cleaning. In extreme cases more than 200 annual kiln stops due to blockages may occur, so that the impact on kiln availability and productivity can become a severe issue. Suspension preheaters armed with dozens of shock blowers (Fig. 1a, 1b) to prevent build-up formation illustrate the importance of this problem.

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Fig. 1a

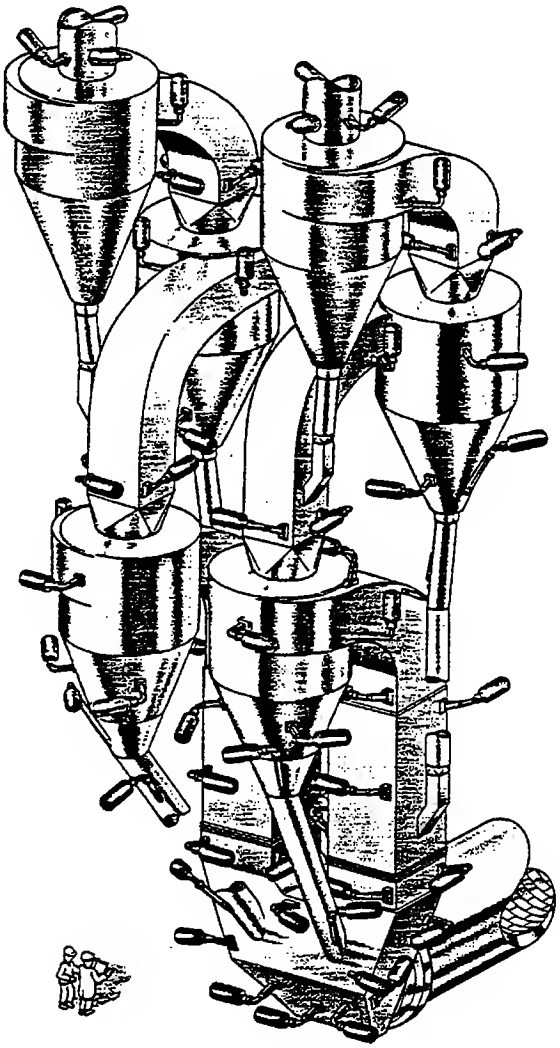

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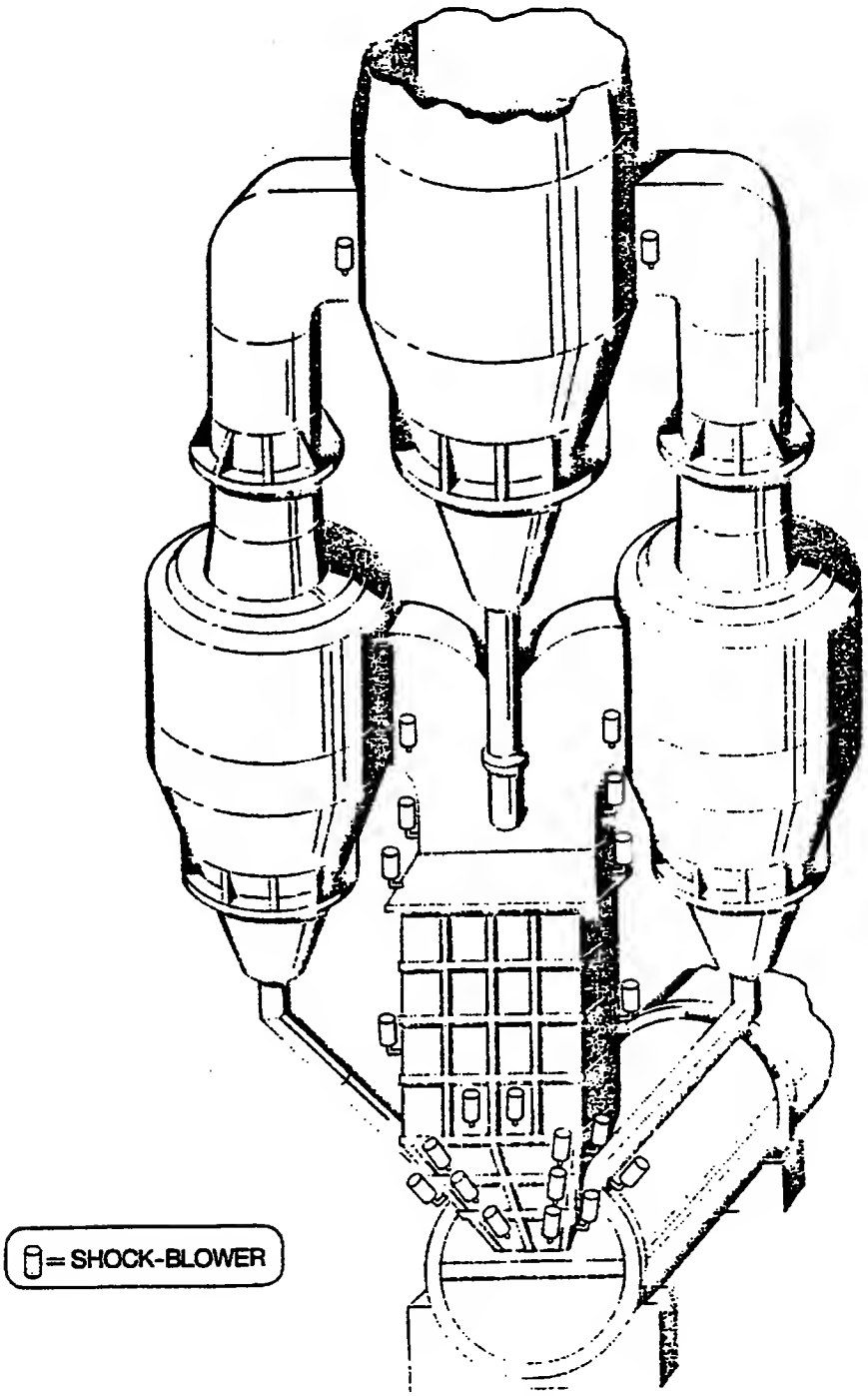
Fig. 1b



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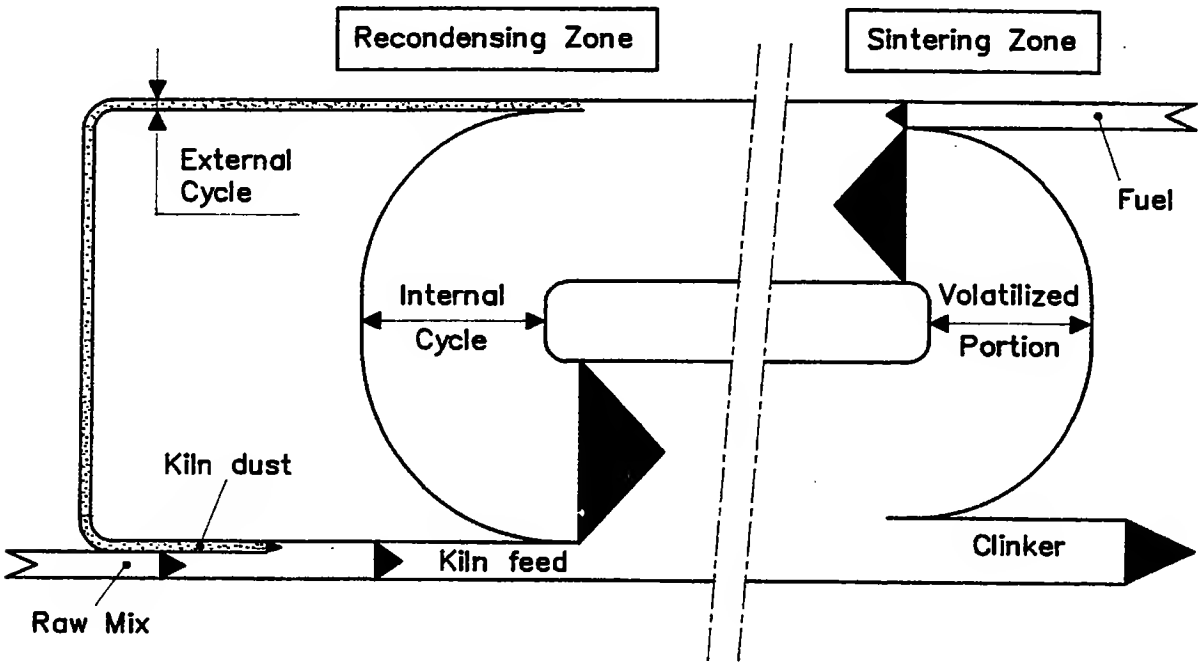
2. MECHANISM OF THE CIRCULATION PHENOMENA

Depending on the degree of volatility, the circulating elements evaporate in the sintering zone of the cement kiln and are carried by the gases to colder zones, where they mainly condense on the raw meal and partly also on the surrounding walls.

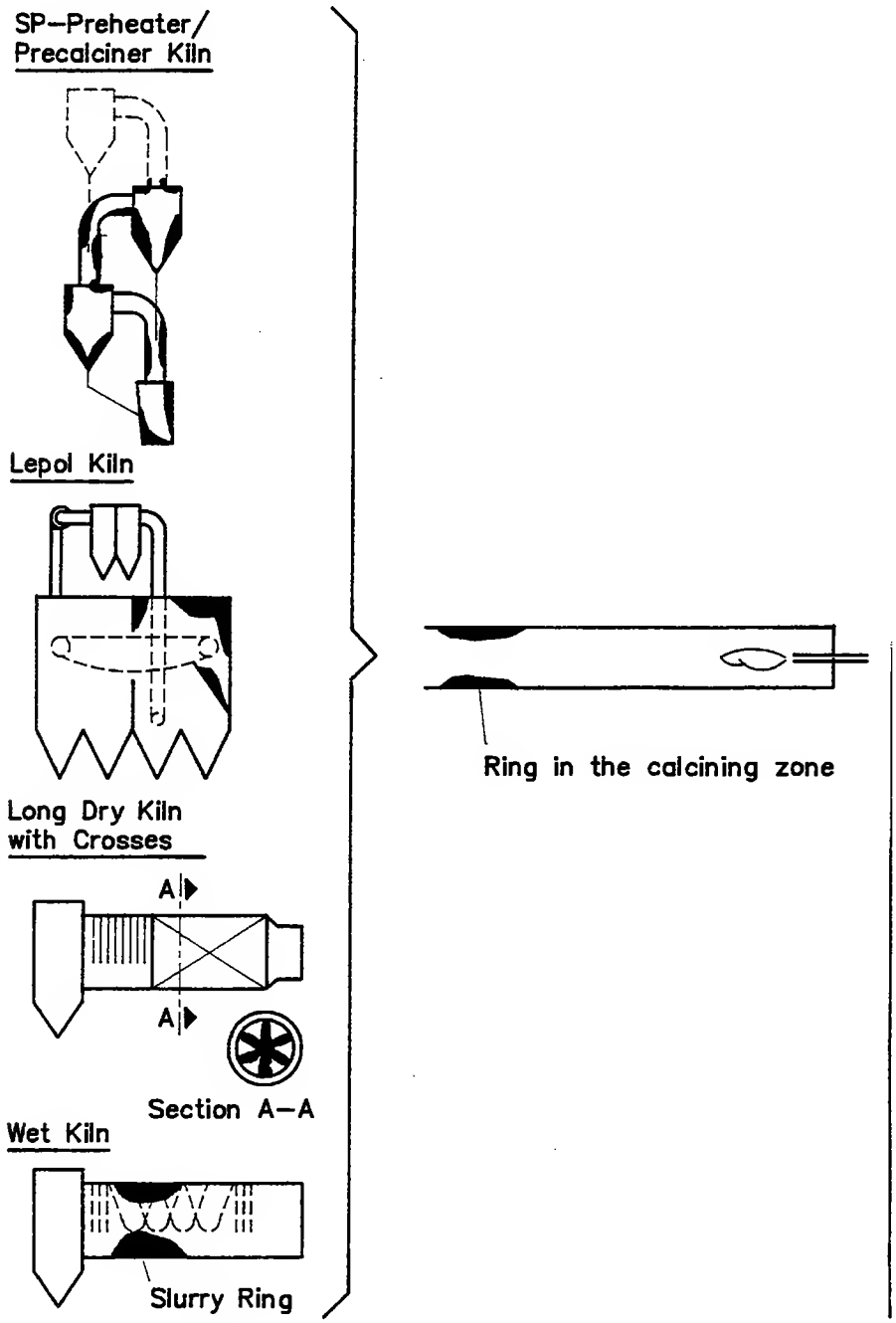
Afterwards they return with the raw meal into the sintering zone where they partly reevaporate depending on the degree of volatility. This repeated process through the kiln leads to the establishment of internal cycles (see Fig. 5). Finally the cycles reach equilibrium so that the output of circulating elements is equal to their input by the raw materials and the fuels (cp. Fig. 2).

Fig. 2 Circulation of Volatile Elements within the Kiln System



Almost all the circulating elements finally leave the system with the clinker. However, this is only the case when beforehand sufficiently high internal cycles of the volatile elements have been formed. The extents of these cycles depend on the degree of volatility of the circulating elements. As the latter recondense on the colder raw meal and the surrounding walls, the formed sticky molten salts are able to reduce the fluidability of the raw meal and, if present in sufficient quantities, to glue it finally on the walls. From time to time, especially during a change of the temperature profile, pieces of build-ups fall down and mainly block the cyclone outlets (Fig. 3).

Fig. 3 Build-ups due to Circulating Elements in the Various Kiln Systems



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If the amount of molten salts becomes too high, either because of an excessive input of volatile elements or due to a high degree of volatility, the installation of a kiln gas bypass becomes necessary in order to extract part of the circulating elements from the kiln system (Fig. 4a, 4b, 4c).

Fig. 4a Conventional Bypass with Water Injection

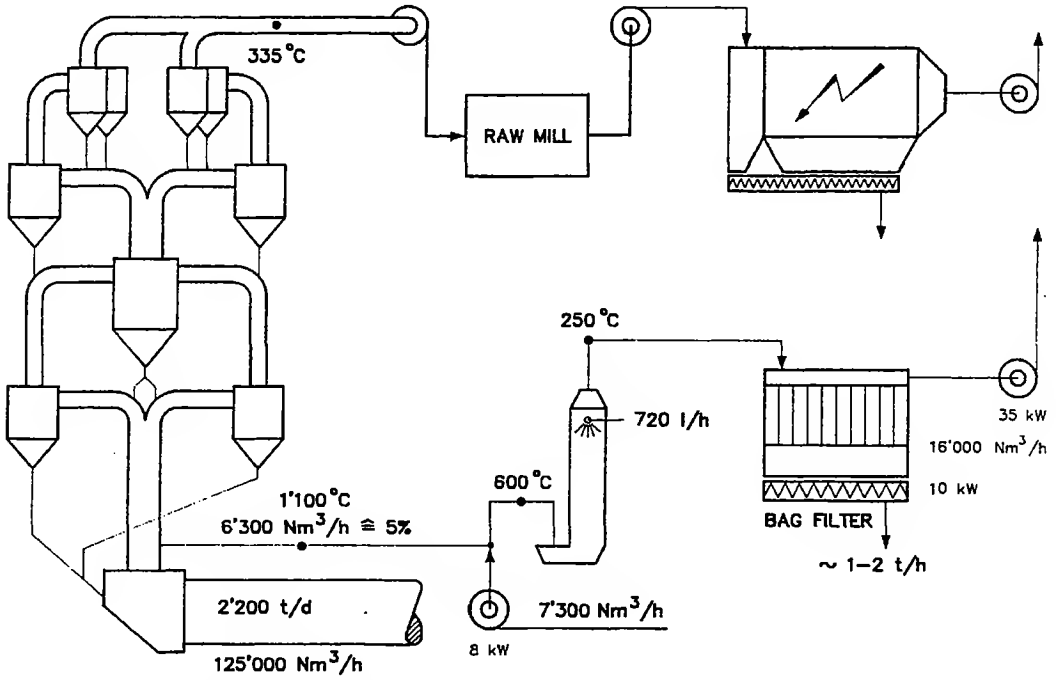

Fig. 4b Hot Gas Bypass without Water Injection

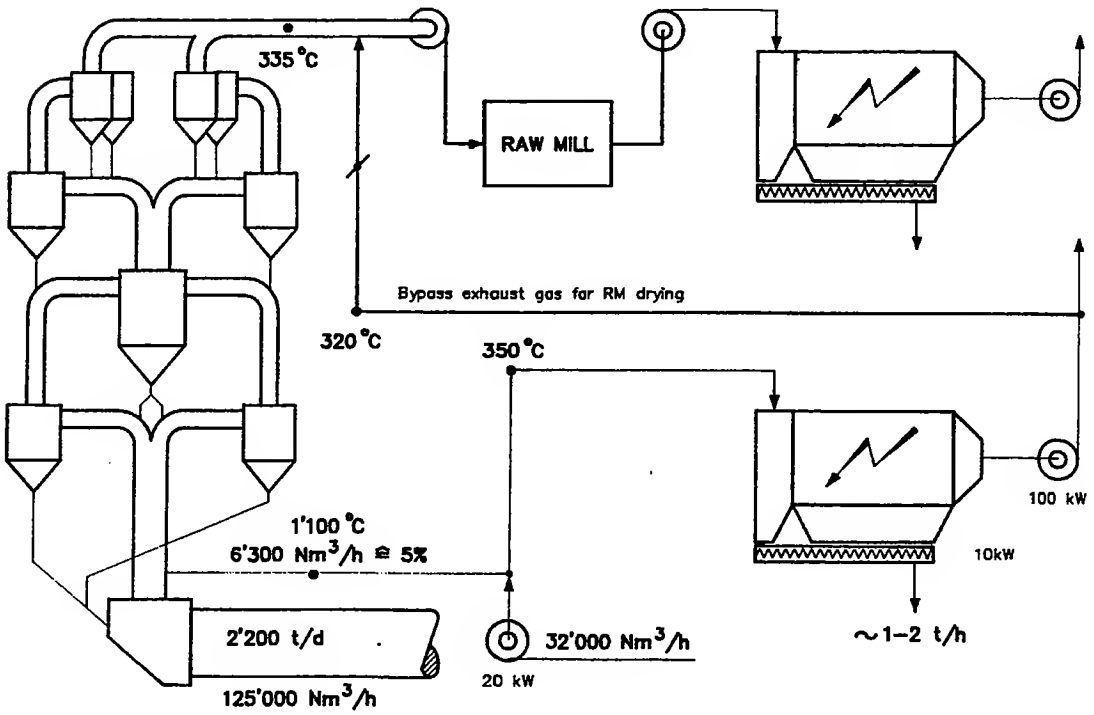

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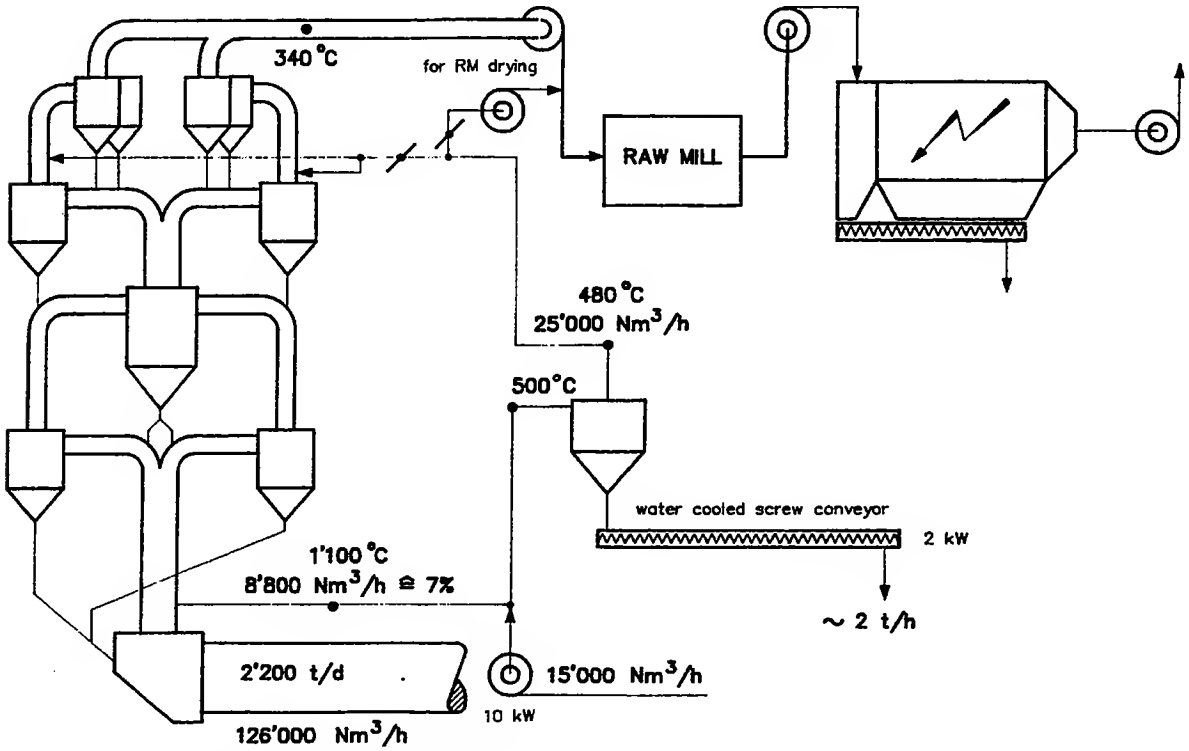
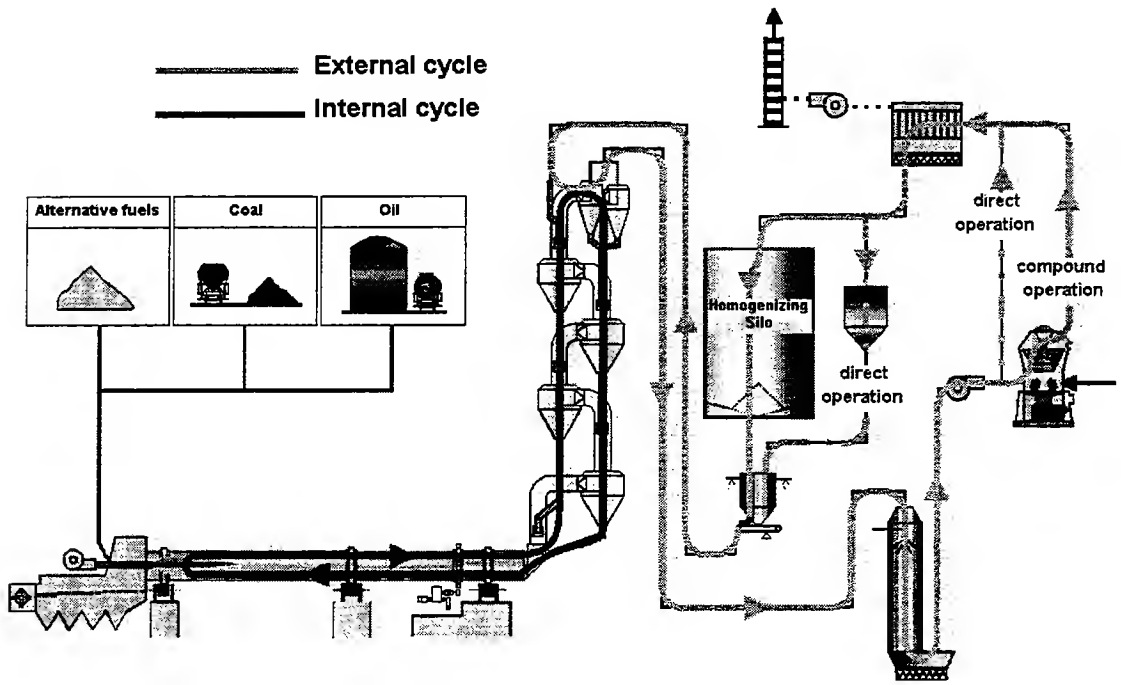
Fig. 4c Bypass with Gas Feedback



A small part of the circulating elements leave the kiln system with the main exhaust gas dust. The latter is efficiently precipitated in the dedusting unit and is normally reintroduced into the kiln system. This is called the external cycle of the circulating elements (Fig. 5).

Fig. 5 Circulation Phenomena: Internal and External Cycle



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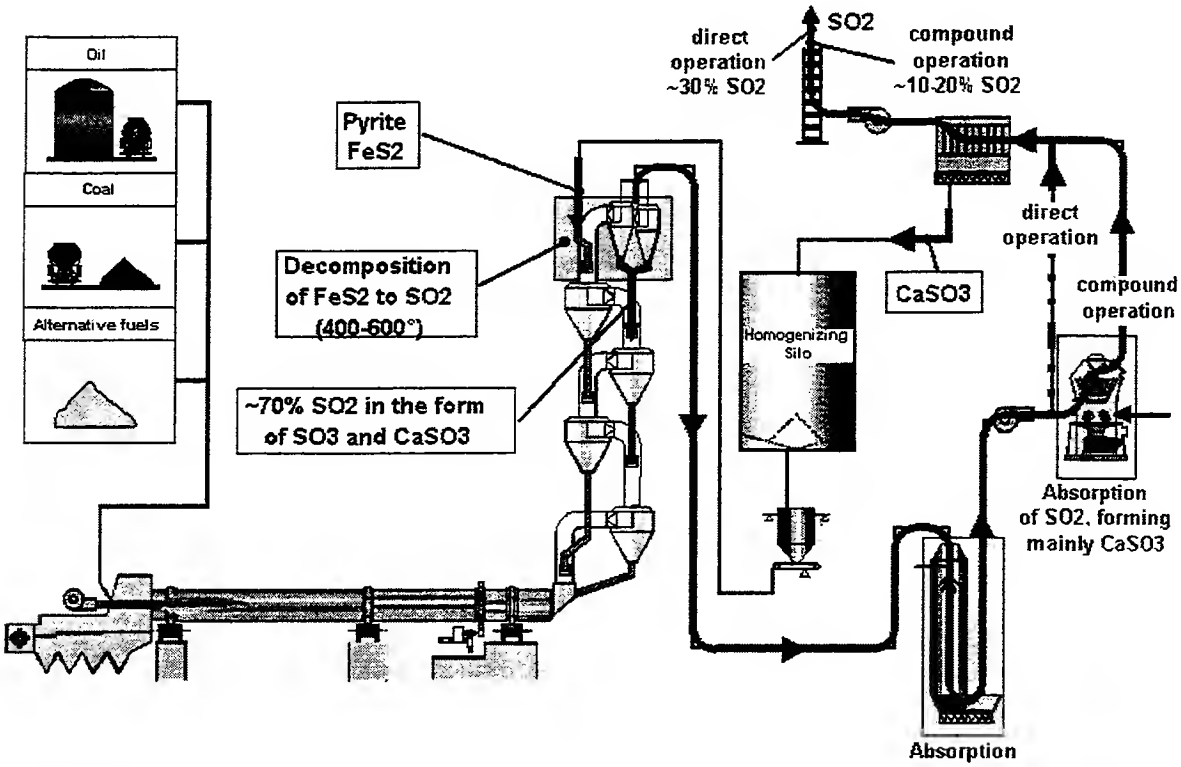
In normal cases a negligible amount of circulating elements is emitted by the stack into the atmosphere. However, the emission of SO_2 is not negligible anymore, when the sulfur in the raw material is present in form of sulfide (FeS_2, PbS, ZnS) or organic compounds. In this latter case the sulfides are volatilized in the temperature range of 400 to 600°C and leave the kiln system partly as gaseous SO_2 emission (Fig. 6 and section 3.3.4).

Fig. 6 SO_2 Emission in case of Sulfides



3. CIRCULATING ELEMENTS IN THE KILN SYSTEM

3.1 Input of Circulating Elements

In the following the typical inputs of circulating elements by the raw materials and the fuel(s) are indicated. It is differentiated between input ranges where usually no encrustation and build-up problems arise and input ranges that usually lead to severe clogging problems.

Important: All figures may serve as rough guidelines only.
The real limits for the build-up and clogging formation depend on a lot of individual parameters like the **degree of volatilization**, the **temperature profile**, the **completeness of combustion** and the **excess air factor** as well as the **kiln system** itself. The indicated values are guidelines for suspension preheater kiln only.

3.1.1 Input by the Raw Materials (loss free basis)

- **Alkalis (K, Na):**
  Generally appear as interlayer cations in the clay minerals and the feldspars.
- **Sulfur:**
  The sulfur is introduced in several mineralogical forms:
  - as **sulfate**: gypsum $CaSO_4 \cdot 2H_2O$; anhydrate $CaSO_4$
  - as **sulfide**: pyrite FeS_2, organic compounds

  In the following only the sulfates are considered.
- **Chlorine:**
  The chlorides are mainly introduced as NaCl (from seawater) or KCl.

Note: The„kiln feed" normally includes already external cycle (see Fig. 2) so that its concentration on circulating elements is higher than the one of the pure raw mix.

Typical limits for 4 to 5 stage SP kilns are listed in section 6.2.2.

3.1.2 Input by the Fuel(s)

Mainly sulfur is introduced by the fuels such as Coal, Coke, fuel oil and so on. In Addition alternative fuels often contain a reasonable amount of sulfur.

Typical limits for 4 to 5 stage SP kilns are shown in section 6.2.2.

3.2 Enrichment of Circulating Elements / Endangered Zones for Encrustation Formation

Depending on the individual circulating element and its compounds, the condensing point lies in the temperature range of 650 to 1000°C.

Condensation: 650 – 800°C : Chlorides and its compounds
800 – 1000°C : Sulfates

Therefore the endangered zones for the formation of build-ups by the condensed circulating elements depend on one hand on the circulating element and its compounds and on the other hand on the kiln system itself (see also Fig. 3).

In the following the build-up zones for the various kiln systems are indicated. At the colder end usually the encrustations by the chlorides are found whereas at the hotter end the ones of the sulfates are met.

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Kiln System	Zones of Build-ups
Precalciner kiln / 4 or 5 stage SP kiln	The two lowest cyclones stage and their riser ducts, kiln inlet area (first section of rotary part)
Lepol kiln	Second preheating chamber, kiln inlet (first section of rotary part)
3 stage SP kiln	Lowest cyclone and its riser duct, kiln inlet, first section of rotary part
2 stage SP kiln	Lowest riser duct, kiln inlet, first section of rotary part
Long dry kiln / wet	Calcining zone of rotary part (transition zone)

3.3 Output of Circulating Elements

3.3.1 Clinker

- The chlorides are so volatile that they hardly leave the kiln via clinker. In exceptional cases, when the sintering zone has largely cooled down or when embedded in big material lumps, the chlorides may leave the kiln in major quantities. However, normally the chlorides form a large cycle within the kiln system and they need to be extracted by a kiln gas bypass.
- The sulfur and the alkalis leave the kiln system normally via clinker either as definite compounds:

K_2SO_4, $K_3Na(SO_4)$, Na_2SO_4
$Ca_2K_2(SO_4)_3$
$CaSO_4$ (rare!)

or in solid solution in clinker minerals:

K	with the Belite, Aluminate
Na	with the Aluminate
SO_3	with the Belite

The calciumanhydrate $CaSO_4$ is rather volatile (decomposes at temperatures > 1000°C) and forms therefore a large sulfur cycle in the kiln system. Therefore it is very important that there is sufficient alkalis to combine with the sulfur and to leave the kiln system as alkali/sulfur compound.

3.3.2 Kiln Gas Bypass

Kiln gas bypasses (see Fig. 4) are mainly used to extract the very volatile chlorides from SP preheater kilns. Chloride bypasses withdraw typically 5 to 15% of the kiln gases. The hot gases from the kiln inlet are quenched down by fresh air, sometimes supported by injection of water into the quench chamber, to a temperature below 600°C. The gaseous chlorides condense onto the withdrawn dust particles and are separated finally in an electrostatic precipitator or a bagfilter (see Fig. 5a, 5b, 5c).

In rare cases kiln gas bypasses are also applied in case of too much $CaSO_4$ and not sufficient alkalis.

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Also for the production of low alkali clinker large kiln gas bypasses are used (20 to 50% of the kiln gases) in order to withdraw the alkalis from the kiln charge and to produce a clinker with an alkali content of < 0.6% (i.e. Na_2O_{eq} < 0.6%). Thereby the alkalis are volatilized by hard burning and by injection of chlorides into the kiln (see also section 0).

The bypass dust must be discarded e.g. into the quarry or is partly reutilized as additive to the cement, or in rare cases may be sold as filler material.
A ballpark figure for the amount of bypass dust: **15 g/kg cli per 10% bypass**

Due to the extraction of the bypass gases additional heat loss arise. The specific value per % bypass depends on the kiln system:

- Precalciner kilns: 8 – 11 kJ/kg cli
- Preheater kilns: 20 kJ/kg cli

Typical chemical concentrations of bypass dusts are found in Fig. 7 and 8.

3.3.3 Kiln Dust

Another possibility to withdraw circulating elements from the kiln system is via external cycle (see Fig. 5) which is interrupted and partly discarded. The enrichment of this kiln dust by circulating elements depends on the kiln system. Whereas kiln dusts from long wet kilns are highly enriched, the ones from SP kilns show nearly the same chemical composition than the raw meal. Typical chemical concentrations of dusts from the different kiln systems are shown on Fig. 7 and 8. Thereby the following legend applies:

WL	=	long wet kiln		
DL	=	long dry kiln		
DG	=	dry kiln with grate preheater (LEPOL)		
DS	=	SP kilns:	F	Filter dust from kiln exhaust gas
			BP	Bypass dust

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Fig. 7 Kiln Dust Analyses

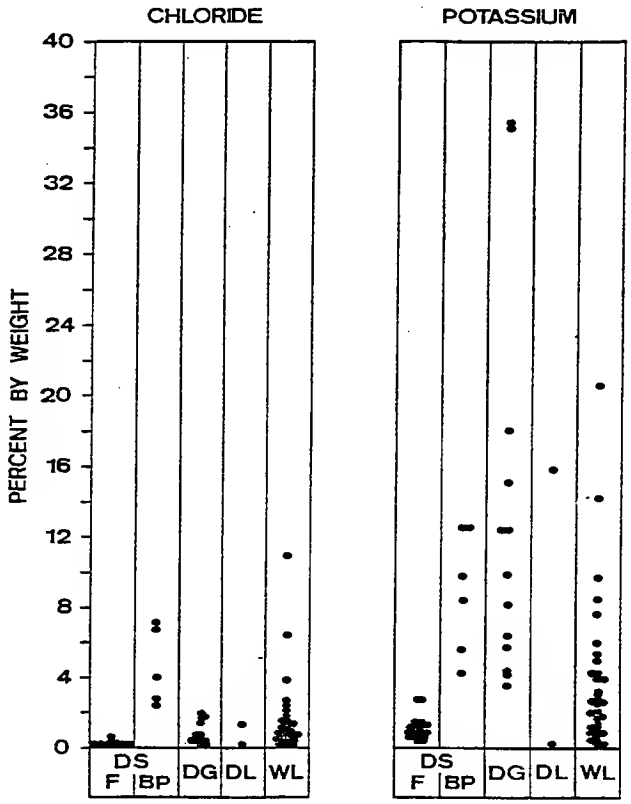
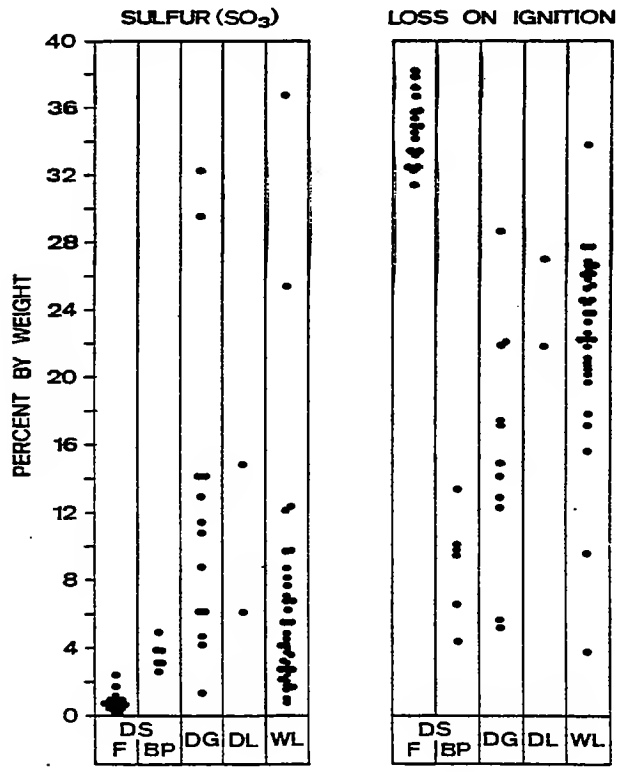


Fig. 8 Kiln Dust Analyses



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3.3.4 Emission by Exhaust Gas

Normally there is no emission of circulating elements by the exhaust gas. The only gaseous component, the SO_2 which does not condense at low temperatures is effectively absorbed by the free lime at the lowest cyclone stage.

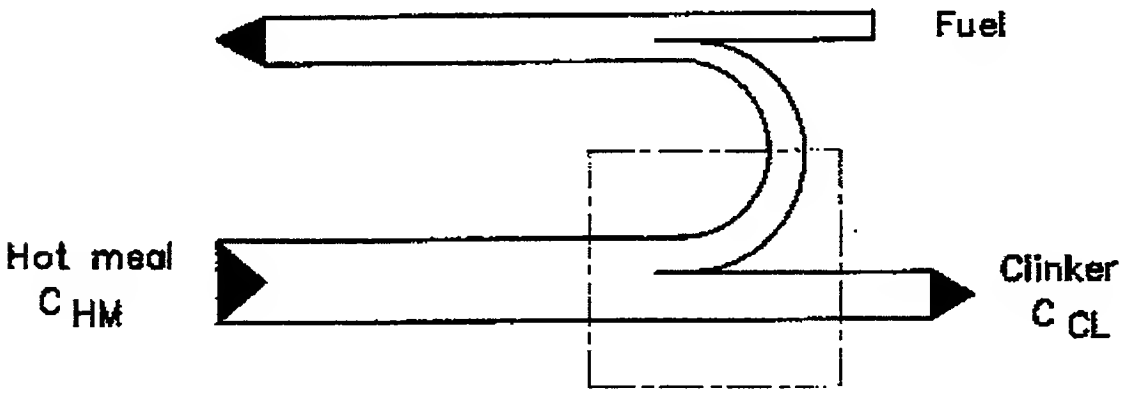
However, a source of SO_2 emission is the raw material sulfur in form of sulfide (e.g. pyrite). The sulfides decompose at temperatures between 400 to 600°C, forming SO_2. Approximately 70% of the SO_2 immediately reacts with the $CaCO_3$ of the kiln feed to $CaSO_3$ and finally at higher temperatures to $CaSO_4$. The residual 30% of the SO_2 are partly absorbed on their way with the flue gases to the stack, mainly in the raw mill and in the conditioning tower. However, a certain portion of the SO_2 is finally emitted by the stack, depending on the kiln and raw mill system (see Fig. 6).

3.4 Volatility of Circulating Elements

3.4.1 Definition of Total Volatility

The total volatility of a circulating element or a compound of it is defined as shown in Fig. 9.

Fig. 9 The Volatility of a Circulating Element



The total volatility of a circulating element or of a compound of it indicates which portion of it is volatilized in the kiln and does not leave the kiln straight with the clinker.

The total volatility is:

$$\varphi = 1 - \frac{c_{cli}}{c_{HM}}$$

whereas φ = factor for total volatility
c_{HM} = concentration of the circulating element at the kiln inlet
c_{cli} = concentration of the circulating element in the clinker

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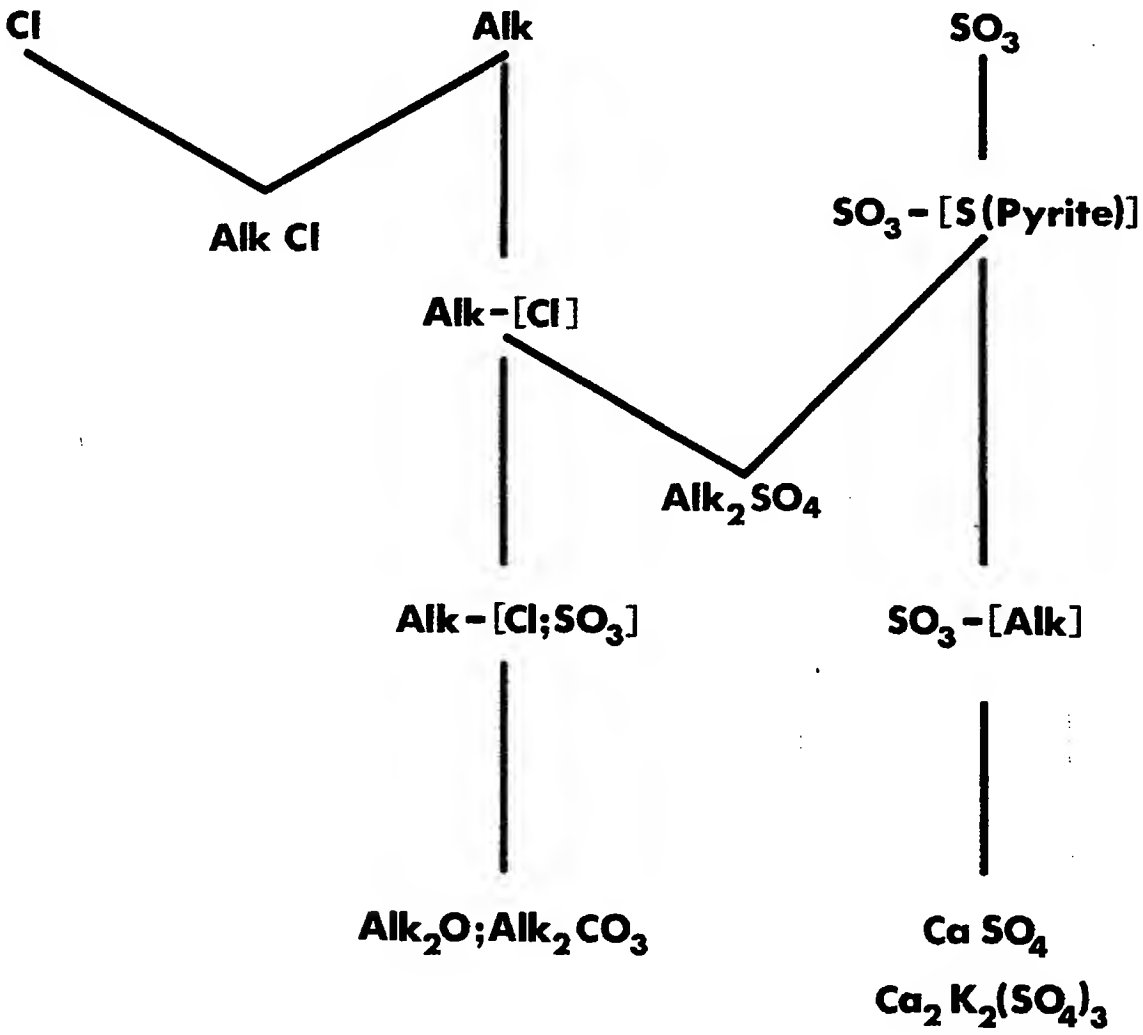
3.4.2 Affinity of the Circulating Elements

Circulating elements have a strong affinity to other elements and form chemical compounds. The following affinity order has been observed (see also Fig. 10):

1) The chlorine combines first with the alkalis, forming KCl, NaCl.
   The residual chlorine, if any, combines with the calcium, forming $CaCl_2$.
2) The residual alkalis combine with the sulfur, forming K_2SO_4, Na_2SO_4. They form also double salts $Ca_2K_2(SO_4)_2$, $K_3Na(SO_4)_2$.
   The residual alkalis, if any, combine with CO, forming K_2CO_3, Na_2CO_3.
3) The residual sulfur (SO_3 or SO_2) combines with CaO, forming $CaSO_4$.

Fig. 10 Affinity

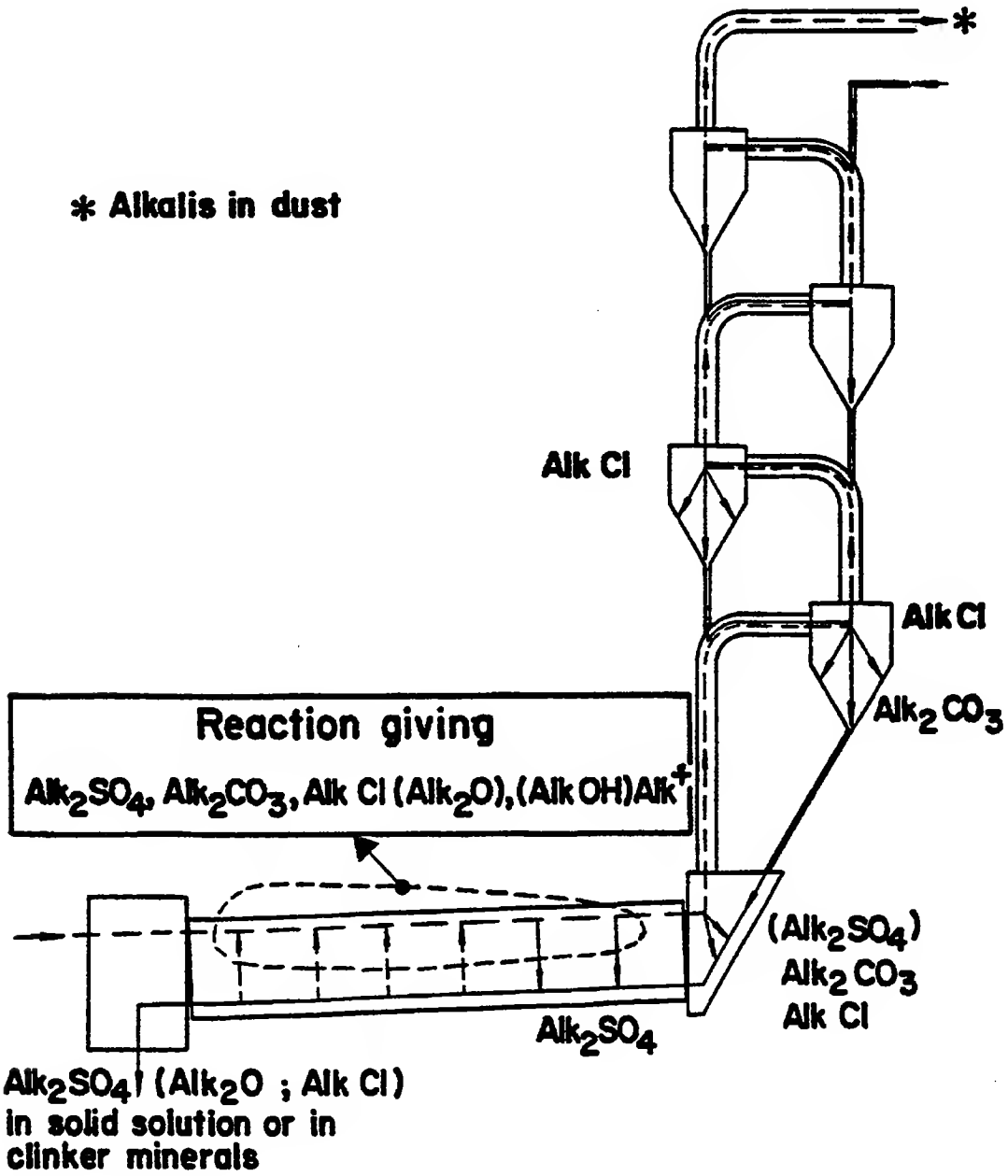


3.4.3 Volatility of the Circulating Element Compounds

3.4.3.1 General

- In the kiln the alkalis are liberated from the clay mineral lattice. They partly dissociate into the gas phase and recombine with other elements according to the affinity order as described in paragraph 3.4.2. The rest recombines directly in the charge material with chlorine or sulfur or is integrated into the lattice of the clinker minerals (belite and aluminate, see paragraph 3.3.1). Especially the sodium is little volatile and goes preferably direct into the clinker (Fig. 11).

Fig. 11 Circulation of Alkalis



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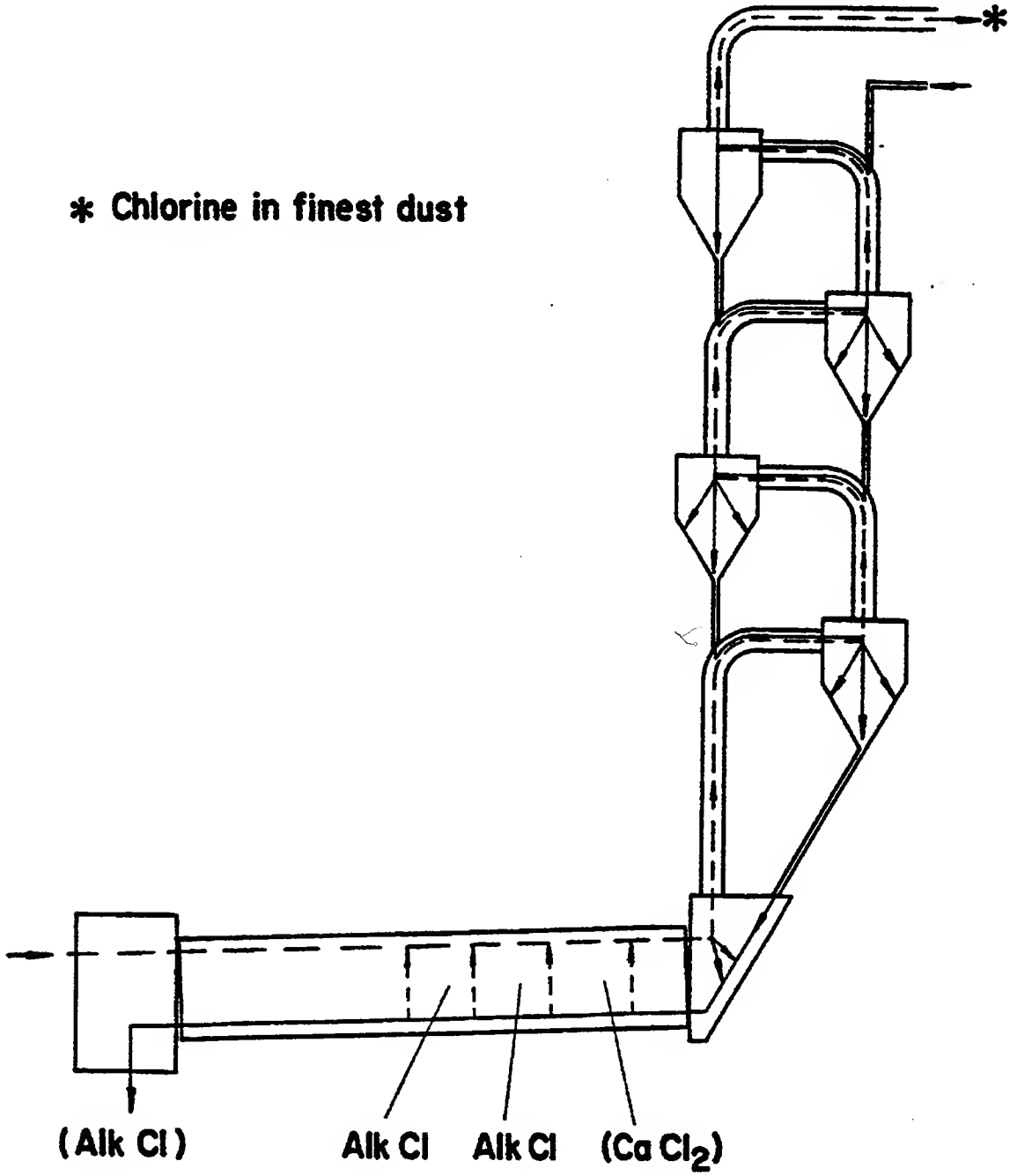
The chlorides liberated during heating of the material and combustion of the fuel react with the alkalis to form alkali chloride. This reaction takes place either in the charge material or, after vaporization, in the kiln gas (Fig. 12).

Fig. 12 Circulation of Chlorine



- Upon volatilization of sulfur at burning zone temperatures in the cement kiln, sulfur dioxide is the predominant component. The gaseous SO_2 derives either from the dissociated sulfur of the combustibles or the decomposed $CaSO_4$ and the partly volatilized Alk_2SO_4 from the charge material.

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Subsequently if there is an excess of alkalis, the internal sulfur cycle is caused primarily by the reaction with these, which takes place almost entirely in the rotary kiln. The alkali sulfates formed in this reaction are, in so far as they are present in vapor form, precipitated on the material. This occurs mainly in the rotary kiln itself, but partly also in the preheater.

The precipitated alkali sulfates thus travel through the kiln again, pass through the sintering zone, and are partly discharged from the kiln with the clinker, while some of the alkali sulfates remain in the cycle.

The calcium sulfate ($CaSO_4$) is formed as a result of the reaction between SO_2 and CaO. It is partly present already in the raw materials or is formed preferably in the temperature range of 800 to 900°C as illustrated in Fig. 13. Above a temperature level of 1000°C the calcium sulfate starts decomposing, first rather slowly and beyond a temperature of 1300°C rapidly. Sulfur dioxide again is formed. If the alkali input is not high enough to combine the whole incoming sulfur as alkali sulfate, high sulfur dioxide concentrations in the kiln gas arise. In such circumstances calcium sulfate may pass un-decomposed through the sintering zone leaving the kiln embedded in the belite mineral or even as $CaSO_4$. The sulfur cycle is illustrated in Fig. 14.

Fig. 13 SO_2 Absorption by CaO resp. $CaCO_3$

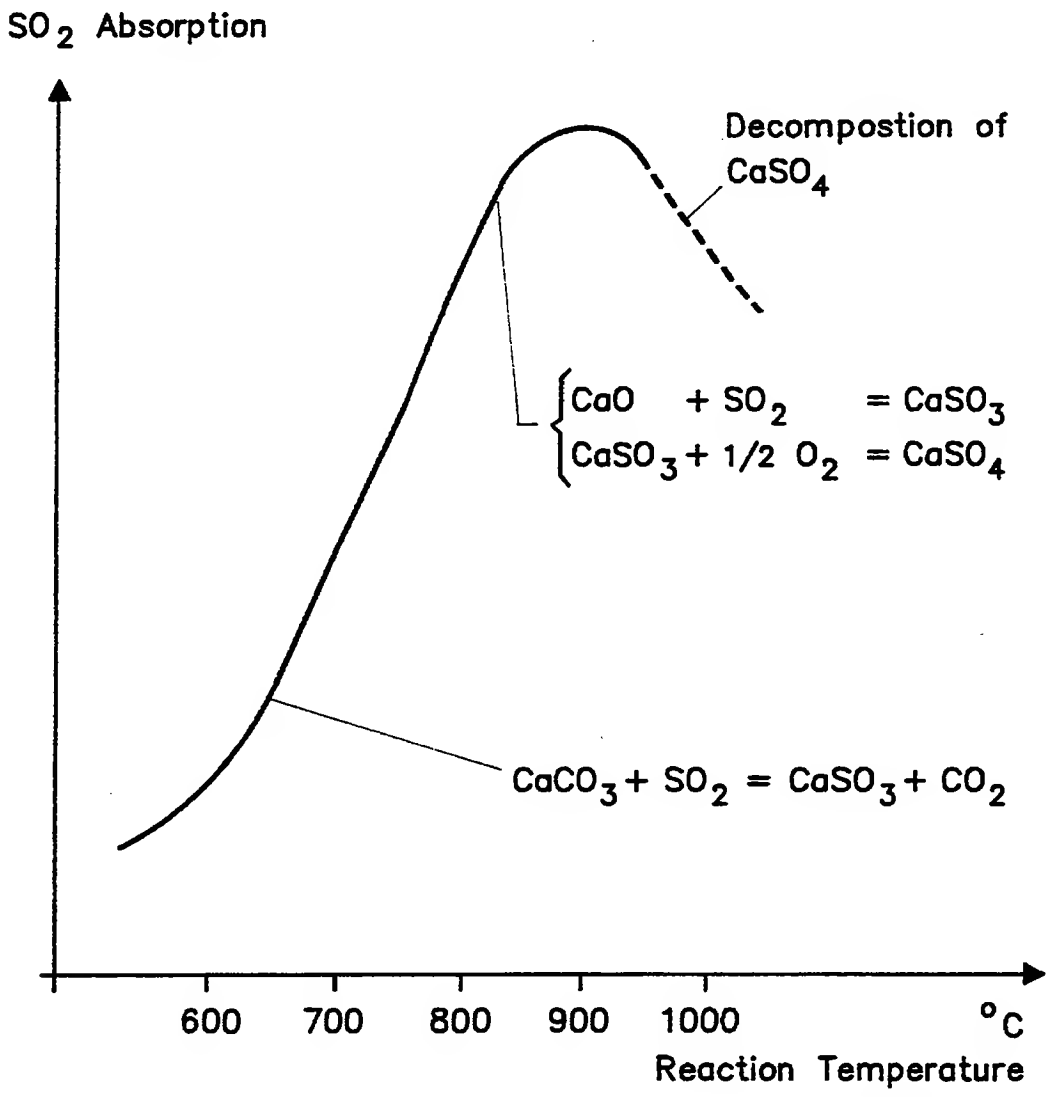

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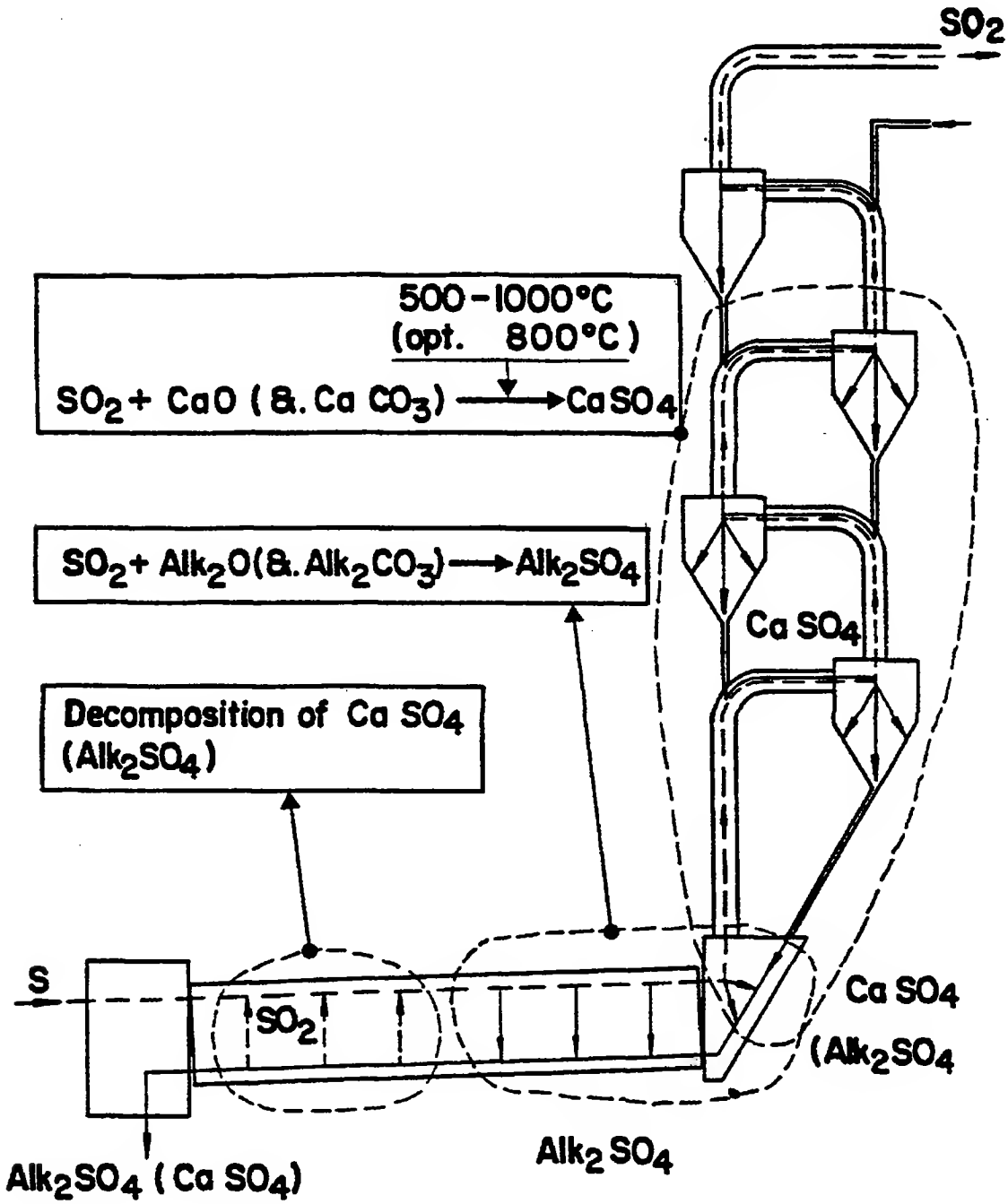
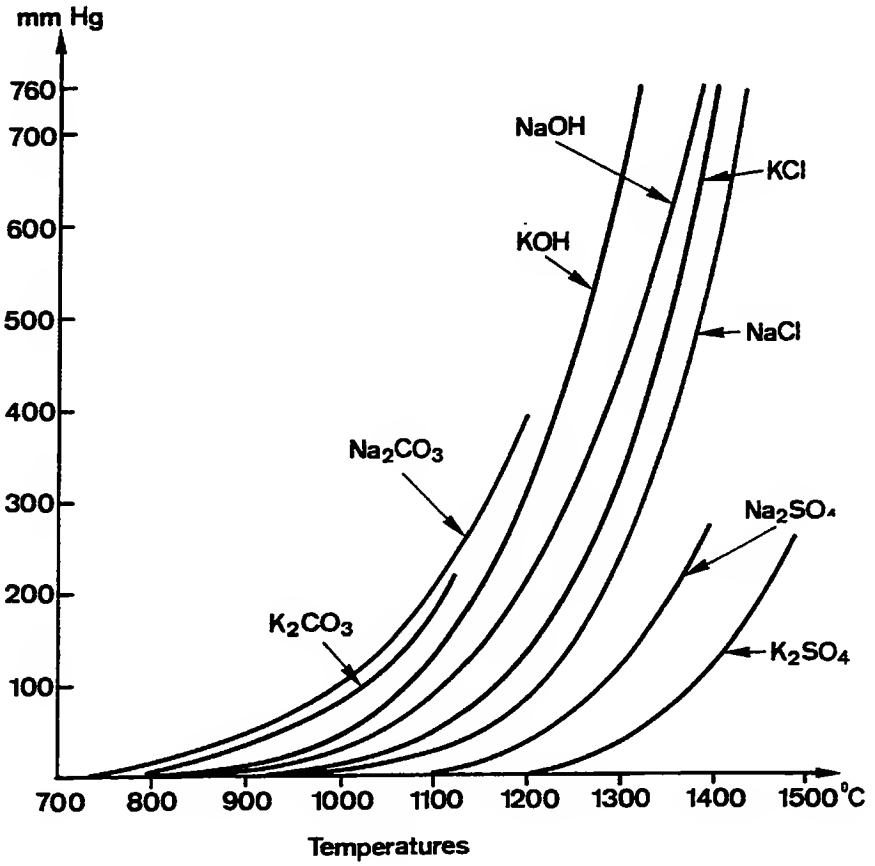
Fig. 14 Circulation of Sulfur



3.4.3.2 Volatility of the Compounds of Circulating Elements

Fig. 15 indicates the relation between vapor pressure of various alkali compounds at different temperatures. This demonstrates that the volatilization mechanism is dependent upon the form of combination of the particular volatile element.

Fig. 15 Vapor Pressure



Chlorides: KCl, NaCl, $CaCl_2$

At a temperature level of 1200 to 1300°C the chlorides are already volatilized to a great extent. At sintering zone temperatures they are almost entirely volatilized so that the total volatility factor φ is approx. 0.97 to 1.

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Sulfates: Alk_2SO_4, $CaSO_4$

Basically, the Alk_2SO_4 are little volatile, whereas the $CaSO_4$ is highly volatile.

Therefore the most important criteria for the total sulfur volatility is the molar ratio between the alkalis and the sulfur itself, corrected by the chlorine:

Molar alkali / sulfur ratio:

$$\frac{Alk}{SO_3} = \frac{\frac{K_2O}{94} + \frac{Na_2O}{62} - \frac{Cl}{71}}{\frac{SO_3}{80}}$$

$$\text{Desirable value: } \frac{Alk}{SO_3} \approx 1.2$$

If there are sufficient alkalis available to combine with the total sulfur input (from raw materials as well as from the fuel), the total volatility for the sulfur is $\varphi = 0.3$ to 0.5.

However, if there is a surplus of sulfur over the alkalis, the highly volatile $CaSO_4$ is formed. Its volatility is approximately $\varphi = 0.9$, but can increase up to $\varphi = 1$, depending on the operating conditions of the burning process.

In general the sulfur volatility is very much depending on the operating conditions of the burning process, such as

- the maximum temperature in the sintering zone
- the retention time of the kiln charge material at high sinter zone temperatures
- the granulometry of the kiln charge material (diffusion to the surface of the granules)
- the partial pressure of O_2 in the kiln atmosphere
- the partial pressure of SO_2 in the kiln atmosphere

For more details see next paragraph 3.4.4.

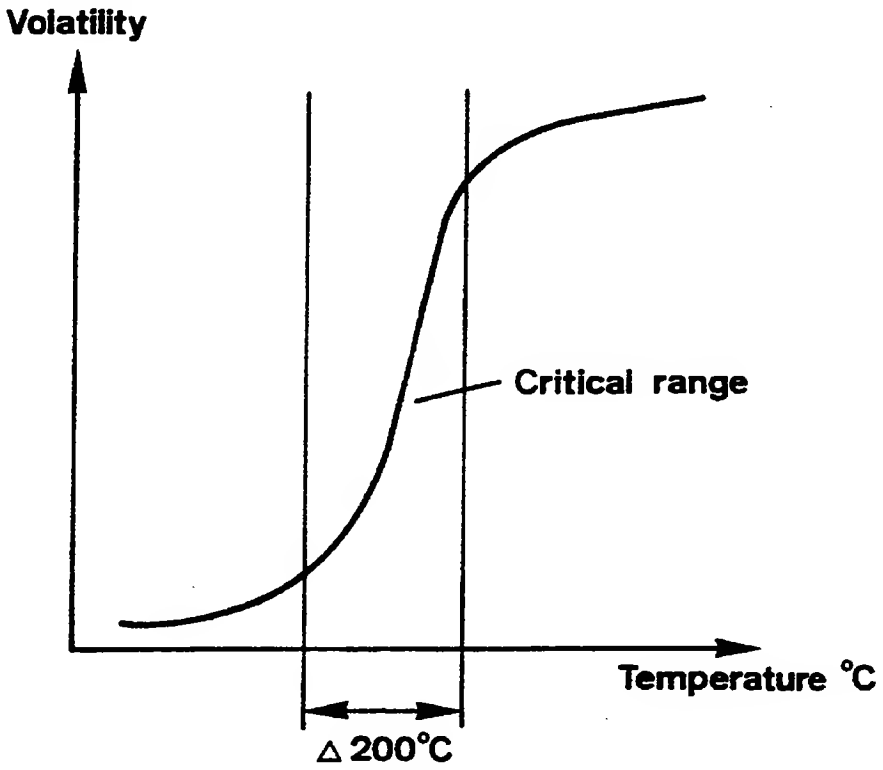
$1.5 \leq \frac{Alk}{SO_3} \leq 0.8$ → pb top S
Low Alkali

3.4.4 Parameters Influencing the Volatility of the Sulfur Compounds

3.4.4.1 Maximum Temperature in the Sintering Zone

The essential feature of the volatility - temperature curve is that above a certain critical temperature the volatility increases first exponentially and then slowly reaches a maximum (Fig. 16).

Fig. 16 Volatility-Temperature Curve



For sulfur this critical temperature is in the order of the burning temperature. Consequently hard burnable raw mixes or overheating of the kiln charge material (low free limes in the clinker) have a very important bearing on sulfur volatility.

3.4.4.2 Retention Time of the kiln Charge Material at High Burning Temperatures

Volatilization of the sulfur compounds is a dynamic process. The longer the kiln charge material is exposed to the high burning temperatures, the higher the total amount of volatilized sulfur.

3.4.4.3 Granulometry of the Kiln Charge Material

If the kiln charge material is well granulated, the sulfur takes more time to diffuse to the surface of the granule. Consequently less sulfur volatilizes.

On the other hand poor granulometry conditions result in higher dust formation which negatively influences the temperature profile in the kiln and shifts it towards the kiln inlet.

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3.4.4.4 Kiln Atmosphere

The composition of the kiln atmosphere is an essential factor for the sulfur volatility:

$$Me_n(SO_4)_m \Leftrightarrow n \cdot MeO + m \cdot SO_2 + \frac{m}{2} O_2$$

The equilibrium of the dissociation is shifted with increasing O_2 partial pressure and SO_2 partial pressure in favor of the sulfates.

Too little excess air or also local reducing kiln atmosphere increases the volatility of the sulfur.

This behavior is illustrated for the K_2SO_4:

$K_2SO_4 \leftrightarrow K_2O + SO_2 + ½O_2$ → **O_2 excess** in burning zone:

$K_2SO_4 \xleftarrow{\leftarrow} K_2O + SO_2 + ½O_2$

→ **O_2 deficiency** in burning zone:

$K_2SO_4 \xrightarrow{\leftarrow} K_2O + SO_2 + ½O_2$

Laboratory tests have been executed for the primary volatility ϵ_1, i.e. the volatility of the sulfur when exposed the first time to high temperatures. It is remarkable that already at a temperature level of 1000°C, the sulfur volatility goes up to 1 (100% volatility) if there is no oxygen in the kiln atmosphere. This is the case if the combustible produces a local reducing atmosphere in the kiln charge material e.g. when burning whole tires at the kiln inlet. From these tests it can be derived that the main influence of the oxygen content on an increase of the sulfur volatility is produced between 0 to 2% O_2. Beyond 2% O_2 the influence is negligible (see Fig 17).

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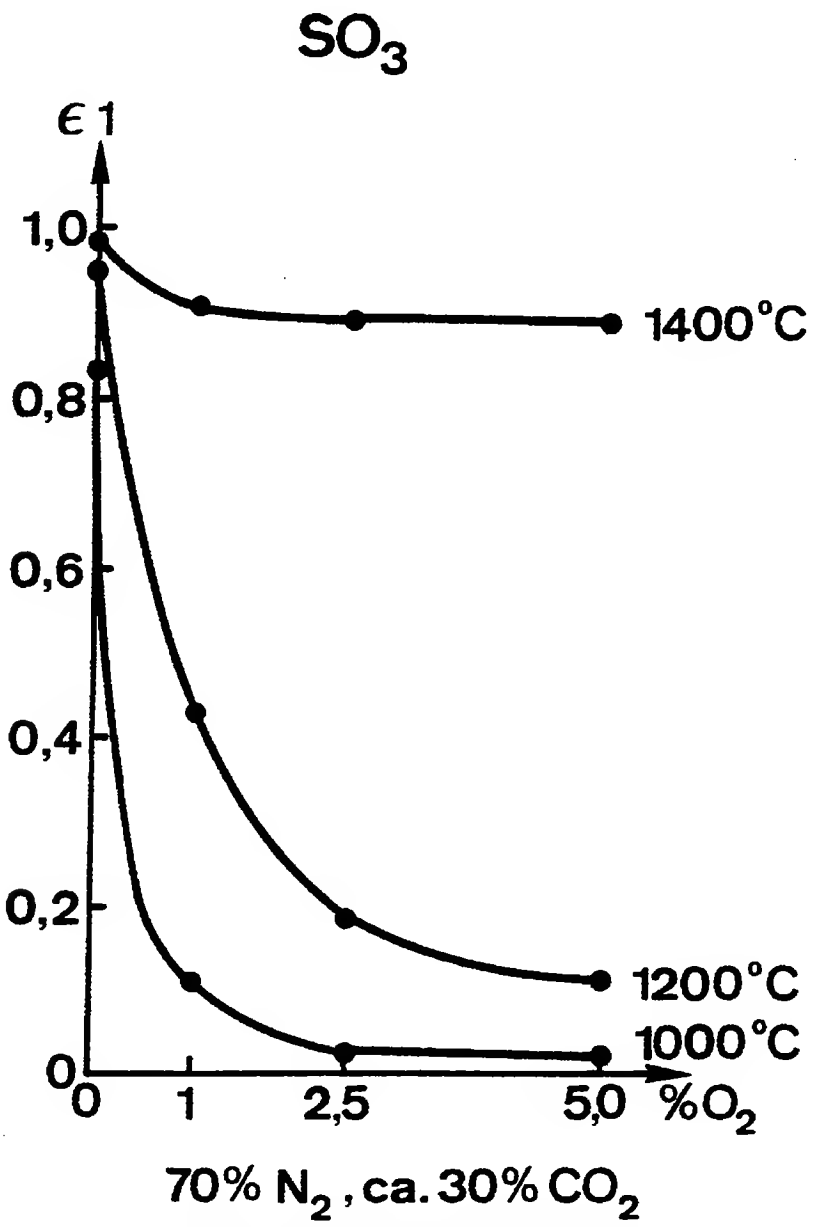
Fig. 17 Sulfur Volatility depending on Oxygen content



The partial pressure of the sulfur dioxide (SO_2) also decreases the sulfur volatility. That means that if the sulfur cycle in the kiln is large enough, the dissociation of the sulfates decreases and the sulfates can gradually leave the kiln with the clinker. However, this is not much of a help, as the required high sulfur cycles already cause encrustation problems!

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3.5 Condensation of Circulating Elements

As previously stated, the circulating elements volatilize in the hot zones of the kiln and condense at the colder areas of the kiln system (internal cycle).

Melting point and boiling point (at 1 bar) of some chlorides, sulfates and carbonates are listed below. Generally speaking it can be stated that the chlorides condense at lower temperatures than the sulfates.

		Melting Point [°C]	Boiling Point [°C] at 1 bar
Chlorides	NaCl	801	1413
	KCl	776	1500 (sublim.)
	$CaCl_2$	772	> 1600
Sulfates	Na_2SO_4	884	?
	K_2SO_4	1069	1689
	$CaSO_4$	1280 (d)	-
Carbonates	Na_2CO_3	851	(decomp.)
	K_2CO_3	891	(decomp.)

In the melting phase secondary compounds are formed which are more complex than the original volatile elements:

Secondary Compounds:

Chlorides	Sulfates	Carbonates	Spurrites
KCl	K_2SO_4	K_2CO_3,	$2\ C_2S \cdot CaCO_3$
(NaCl)	$K_3Na\ (SO_4)_2$	(Na_2CO_3)	$2\ C_2S \cdot CaSO_4$
	Na_2SO_4		
	$Ca_2K_2\ (SO_4)_3$		
	$CaSO_4$		

Furthermore, investigations have revealed that calcium sulfate, calcium oxide and alkali sulfate form eutectic melts which can have much lower melting temperatures than the single compounds (Fig. 18). Under presence of chlorides these melting temperatures are even more lowered to temperatures below 700°C. Very low melting temperatures show also the alkali carbonates. However, in such a case the alkalis can be sulfatizated by the addition of gypsum so that it presents normally not a major problem.

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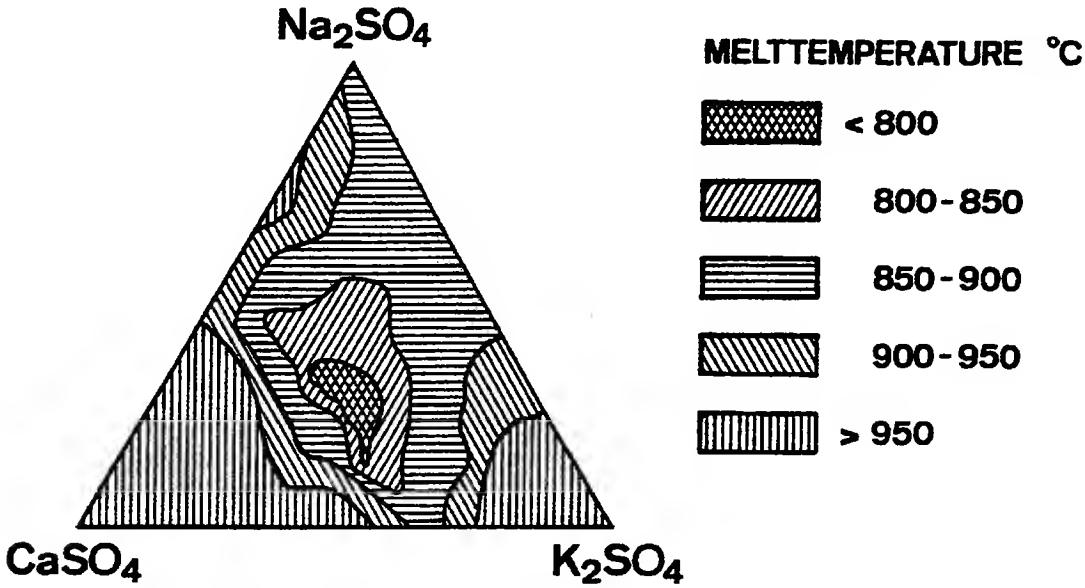
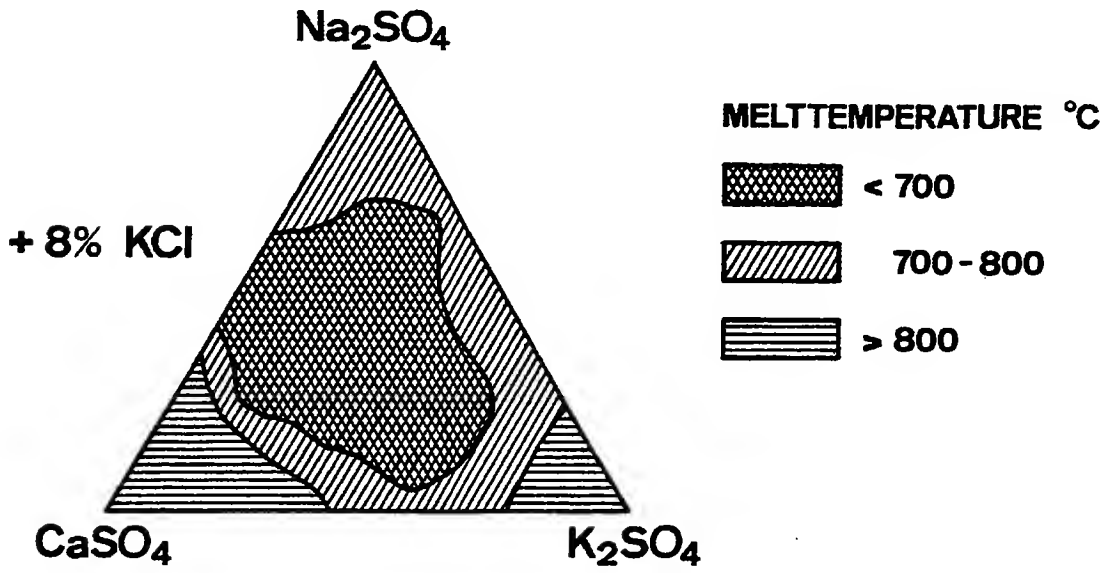
Fig. 18 **Melt Intervals in the System $CaSO_4$ - K_2SO_4 - Na_2SO_4**





All these melts, when present in sufficient quantities, lead to severe encrustations and build-up formation in the preheater and kiln inlet area.

A further problem is given by the fact that the volatilization process goes along with an endothermic reaction whereas the condensation is exothermic. By this way an important amount of heat is drawn from the sintering zone to the kiln inlet so that the area of condensation is shifted even more towards the cold end of the kiln system.

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4. TYPICAL APPEARANCE OF BUILD-UPS

Typical appearance of build-ups and deposits are shown in Fig. 19, 20 and 21.

Fig. 19 Macrographs of Deposit Types

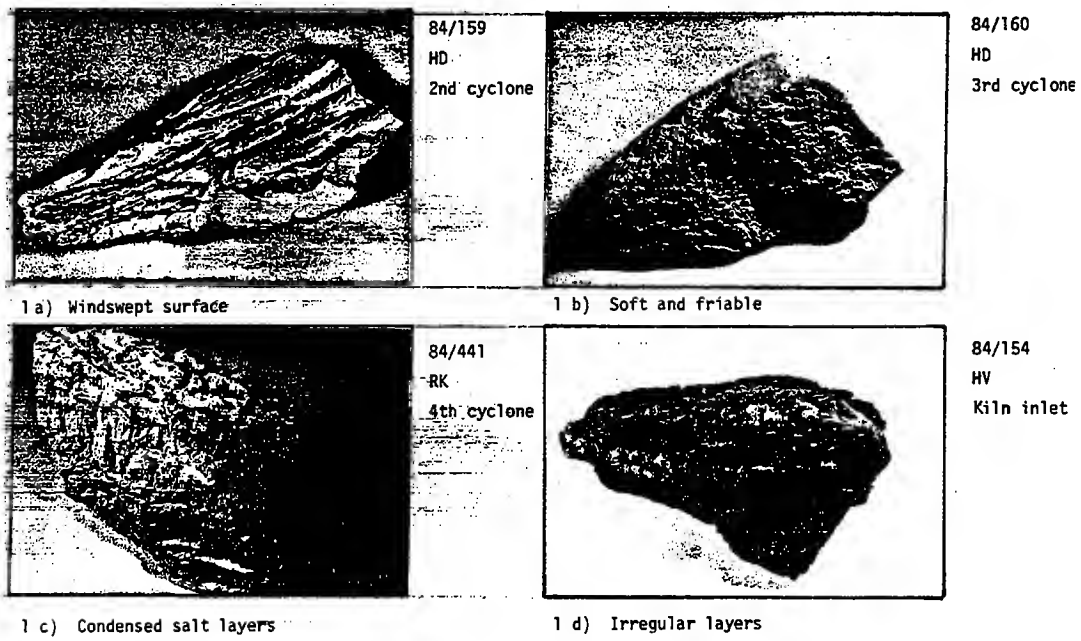

Fig. 20 a) .. d) S.E.M. Micrographs of Deposits

Fig. 20 a) REM 84/506
HD - 1st cyclone: KCl blocks embedded in fine matrix

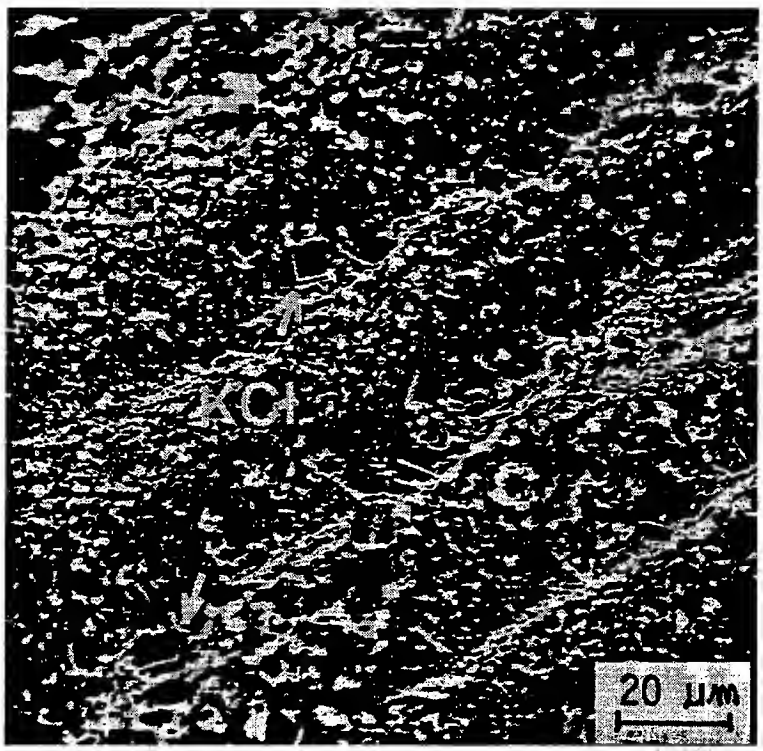

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Fig. 20 b) **REM 84/525**
HD - 2nd cyclone: $CaSO_4$ and C_2S

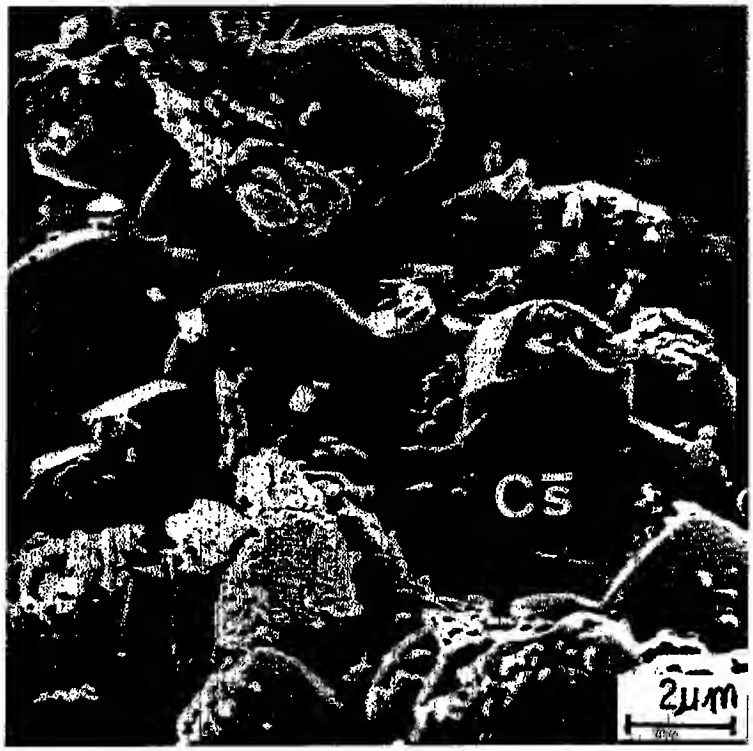

Fig. 20 c) **REM 84/502**
HV - 4th cyclone: Binding nature of glassy KCl matrix

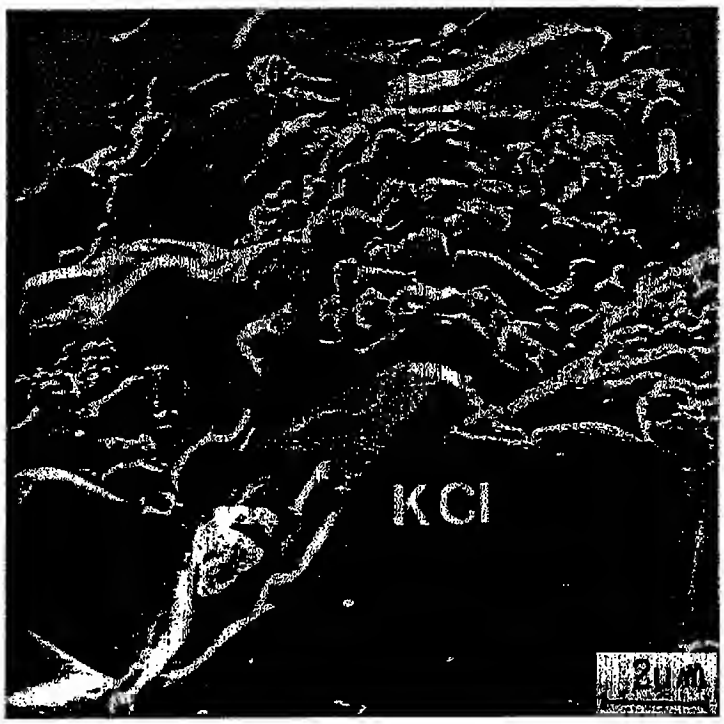

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Fig. 20 d) **REM 84/550**
GM - 4th cyclone: KCl crystal structure filling pore

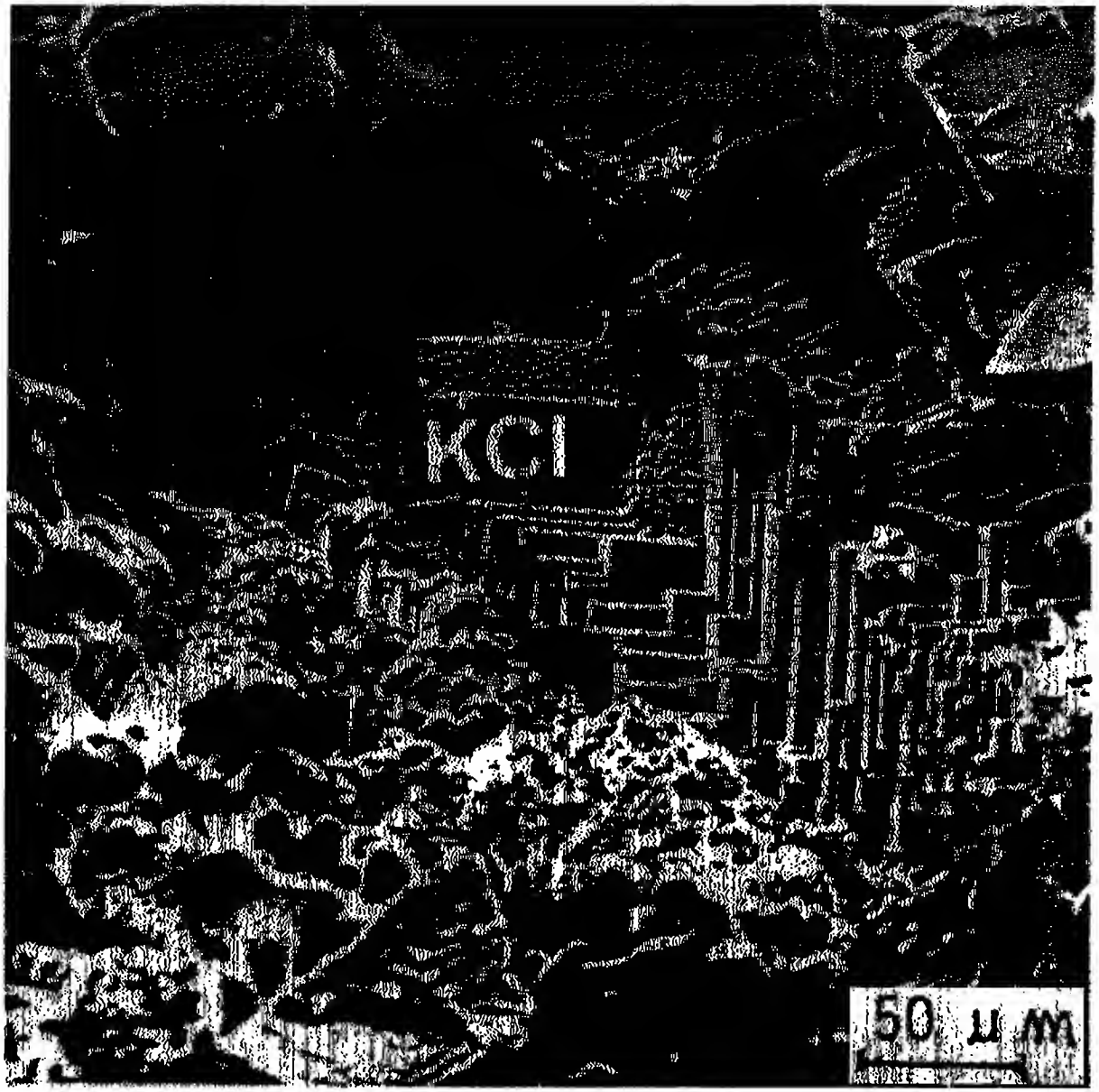

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Fig. 21 a) .. d) S.E.M. Micrographs of Deposits

Fig. 21 a) REM 84/113
TU - Riser pipe: $CaSO_4$ and C_2S

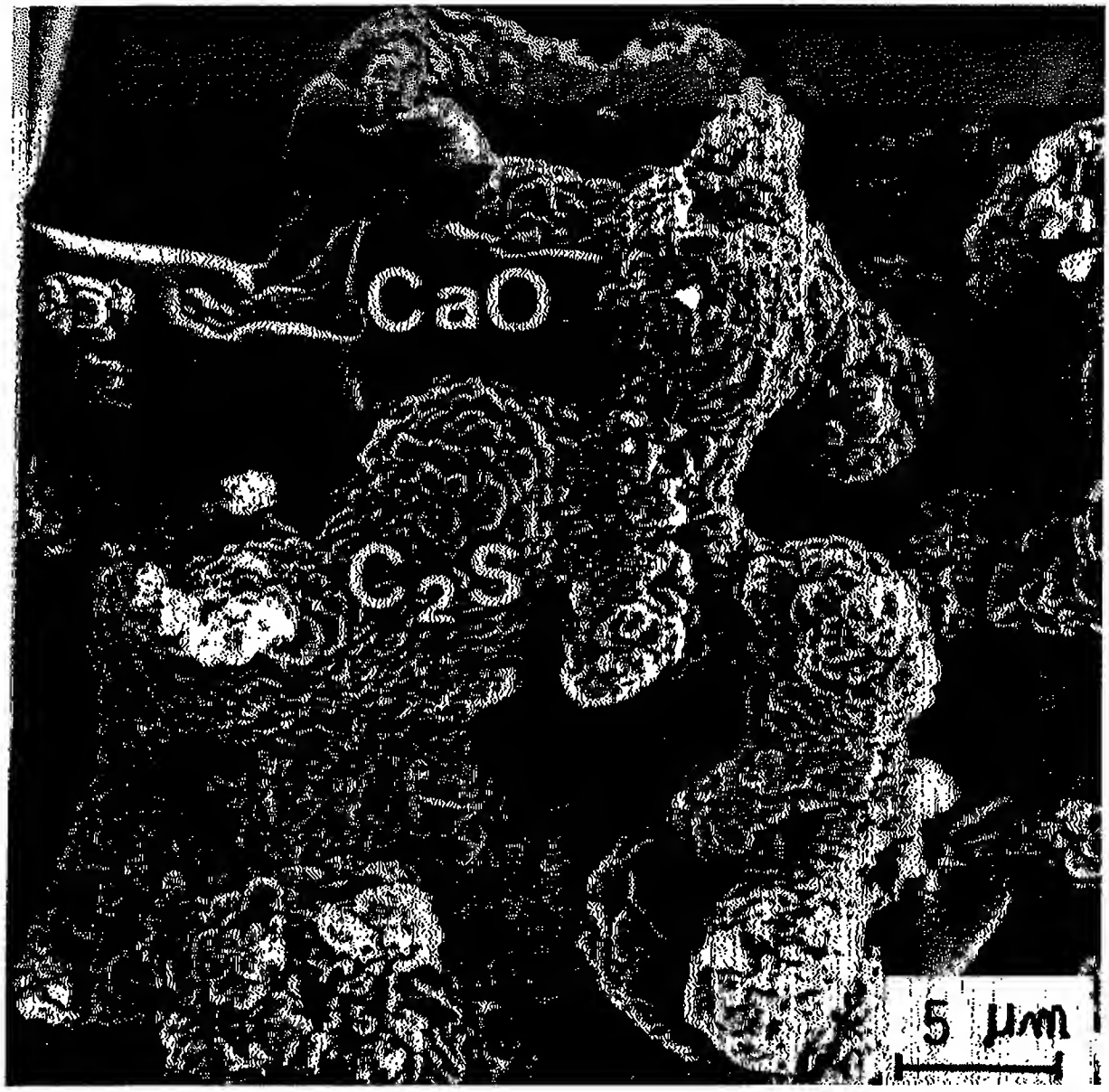

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Fig. 21 b) **REM 84/117**
TU - Riser pipe: K_2SO_4 and Ca SO_4 border

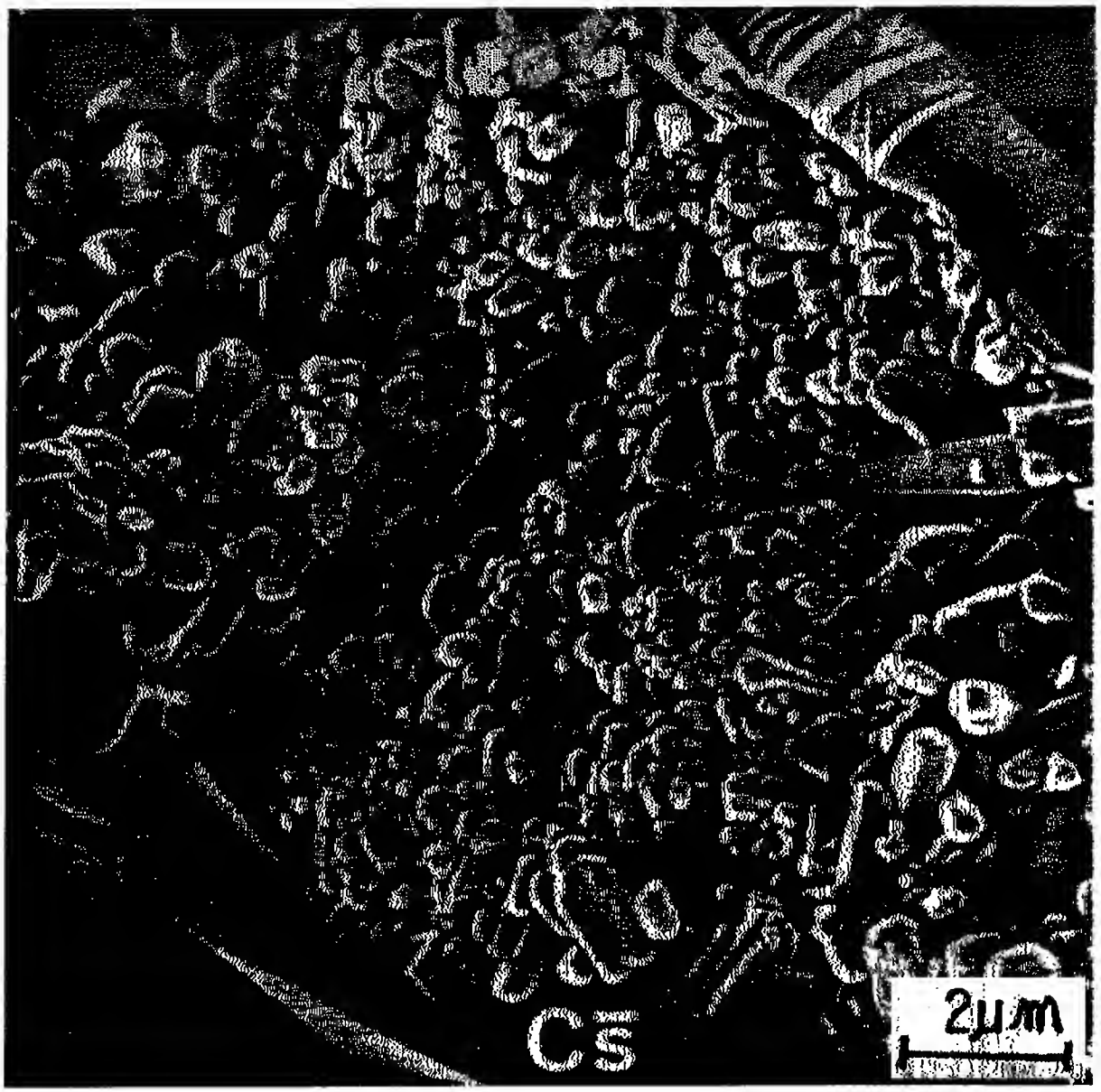

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Fig. 21 c) **REM 84/72**
WU - Kiln inlet: Fly ash balls

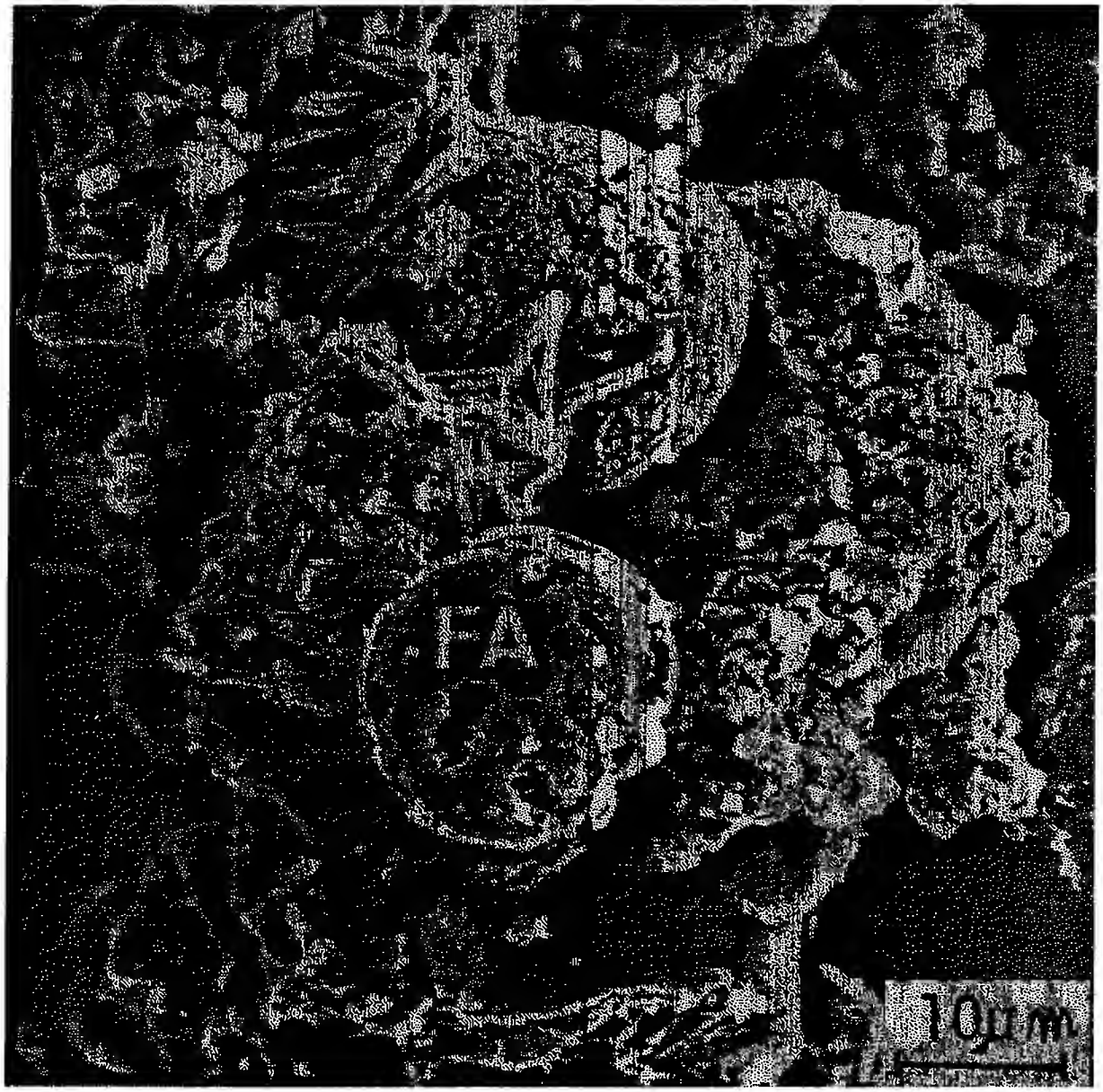

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Fig. 21 d) **REM 84/69**
WU - Kiln inlet: Primary spurrite

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5. KILN OPERATION PROBLEMS DUE TO CIRCULATING ELEMENTS

The consequences due to high internal cycles of the circulating elements are rather severe.

First it starts by an increased encrustation and build-up formation at the kiln inlet area and the lower part of the preheater. As a consequence the pressure loss across the system increases and at the same time also the inbleeding false air quantity increases. This reduces the maximum kiln draft and thus the maximum clinker production. When the kiln operator tries to compensate the lower available kiln draft by a lower excess air rate, the situation becomes even worse as the sulfur cycle further increases.

Studies have revealed that a high chloride cycle impedes also complete combustion. This in turn further increases the sulfur cycle.

High sulfur cycles lead to a poorly granulated clinker and therefore to dust formation. The dust entrains the heat from the burning zone to the kiln inlet, so that the cycles of the volatile elements further increase due to a longer residence time at high temperatures.

Furthermore, the high cycles of volatile elements transport the heat of the sintering zone to the area of condensation by the endothermic - exothermic reactions, causing the same effect as the dust cycles described above.

The result of these mechanisms is always the same:

- Frequent kiln stops due to encrustation and clogging problems, i.e. reduced kiln utilization factor (→ reduction of OEE)
- Reduction of the maximum kiln production
- Higher heat consumption
- Formation of unstable coating at the transition zone and thus high refractory consumption

6. IDENTIFICATION OF PROBLEMS WITH ENCRUSTATIONS AND BUILD-UPS

Chlorides, sulfates, alkalis or any combination of them can cause encrustation and build-up problems. Therefore a detailed analysis is a prerequisite to find an appropriate solution.

The analysis can consist of eight steps as follows:

① Systematic analysis of the hot meal (cp section 6.1)
(every shift: LOI, SO_3, Cl, K_2O, Na_2O)

② Comparison with the defined limits of circulating elements in the hot meal!
→ Chlorine problem, sulfur problem, alkali problem of combination?

③ Performing of a material balance with the inputs and outputs of the circulating elements (cp section 6.1).
→ Where do the circulating elements come from?

④ Comparison with typical tolerable inputs of circulating elements by raw materials and fuels.
→ Which amount is critical?

⑤ Calculation of the molar alkali / sulfur ratio and comparison with standard.
→ Are there enough alkali present to withdraw the sulfur within the clinker?

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⑥ Calculation of the sulfur volatility and comparison with the standard.
→ Why sulfur problem?: → too high inputs?
→ unfavorable alkali / sulfur ratio?
→ because of kiln operation?

⑦ Detailed recording where the build-ups occur and possible a chemical analysis of a typical build-up peace.

⑧ Specific measures against build-up formation (see section 7).

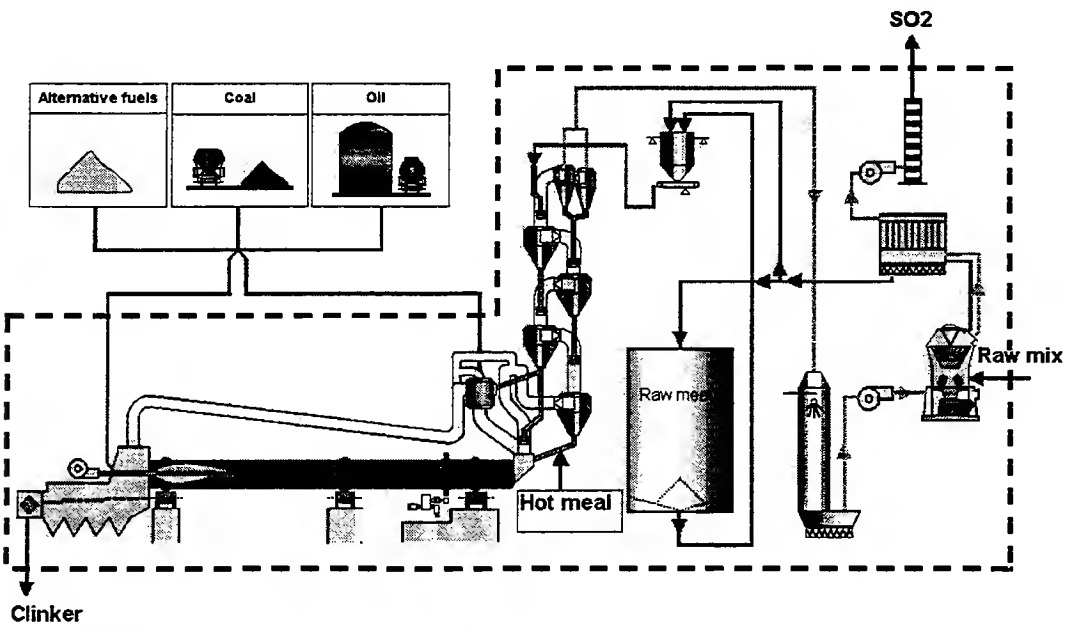
6.1 Material Balance

- Definition of the balance boundary (Fig. 22):
  - Shall the external cycle be included or excluded?
  - Please note that the kiln feed includes the external cycle so that the concentration on volatile elements is higher than the real inputs by the raw material. In this case the kiln dust must be taken into account as further output!
- Taking material samples over a sufficiently large period
  - The sampling period should be at least 8 hours of representative kiln operation.
  - From every input and output an hourly spot sample shall be taken. The spot samples of the individual inputs and outputs are finally combined to one integrated sample.
  - The following quantities for the individual spot samples shall be taken:

raw mix (before grinding):	~ 20.0 kg
meal or dust:	~ 0.5 kg
clinker:	~ 5.0 kg
fuel:	~ 0.5 kg

- Besides the samples for the material balance also samples of the hot meal at kiln inlet (outlet of the lower most cyclone stage) shall be taken, observing the same frequency and sample quantity as stated above.

Fig. 22 Material Balance Boundaries



6.2 Criteria and Indicators to Assess the Build-up Problem

6.2.1 Process Variables

- Position of the build-ups
  - If the build-ups are found up to the second lowest cyclone stage or even to the third lowest cyclone stage the problems could origin from excessive chloride cycles or in rare cases from too high alkali cycles.
  - If the build-ups are found mainly at the lowest cyclone stage, the lowest riser dust and the kiln inlet, the problems normally are caused by excessive sulfur cycles.
- Completeness of combustion
  - CO at kiln inlet > 0.1% and/or O_2 < 1.5 to 2% cause high sulfur cycles.
  - If no secondary firing or precalcination is applied also the CO and O_2 concentrations after preheater can be taken to judge the completeness of combustion (CO must be < 0.1% and O_2 usually 3 to 4%, depending on the inbleeding amount of false air).
- Temperature profile
  - Excessive temperatures within and after the preheater indicate a disturbed temperature profile in the rotary kiln and thus a too high sulfur volatilization.
- Pressure profile
  - The pressure profile over the preheater indicates where the build-ups are located.

6.2.2 Material Balance

First a material balance must be performed as described in paragraph 6.1. Afterwards, the following evaluation shall be done:

- Input of circulating elements
  - It has to be determined in which form the sulfur is present in the raw materials, as sulfate or sulfide. When present as sulfide, part of it will leave the kiln system as emission via exhaust gas stack! Therefore the sulfur emission by the stack has to be deducted from the overall sulfur when judging a potential sulfur problem in the kiln.
  - The inputs of circulating elements (without emitted sulfur through the stack!) should be compared to the ones given below in order to check whether they are in a normal range where usually no build-ups are formed or whether they are too high. This, of course, is only a rough guideline.

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Typical Input Limits for Circulating Elements (for 4 to 5 stage SP kiln only)

- **Raw Material**

(loss free basis)

Cl	< 0.02%	Normal case, no problems
	> 0.05%	Heavy clogging problems, depending on the sulfur cycle
SO_3 [1)]	< 0.5%	Normal case, no problems
	> 1.25%	Heavy clogging problems
K_2O	< 1.0%	Normal case, no problems
	> 1.5%	Problems with encrustations, depending on degree of sulfatization (molar alkali/sulfur ratio)

Na_2O Little volatile, thus no problems due to recirculation of Na_2O

[1)] Sulfates: Sulfur in form of sulfides behaves in a different way, see section 3.3.4 Emission by Exhaust Gas

- **Fuel(s)**

Coal:	S < 1.5%	No clogging problems, depending on sulfur and alkalis input by the raw materials and the completeness of combustion	(corresponding to approx. < 5 g SO_3/kg cli)
Coke:	S < 2%		
Fuel oil:	S < 2.5%		
Coal:	S > 3%	Severe clogging problems, depending on the alkalis and sulfur input by the raw materials and the completeness of combustion	(corresponding to approx. > 10 g SO_3/kg cli)
Coke:	S > 4%		
Fuel oil:	S > 5%		

Gas: Sulfur content is normally zero!

- Alkali / sulfur ratio (A/S)
  - The A/S ratio should be preferably at 1.2 or in a range between 0.8 to 1.5. Compare it to the A/S ratio of the investigated case, deducting first the emitted sulfur from the total sulfur input by the raw mix and the fuel.

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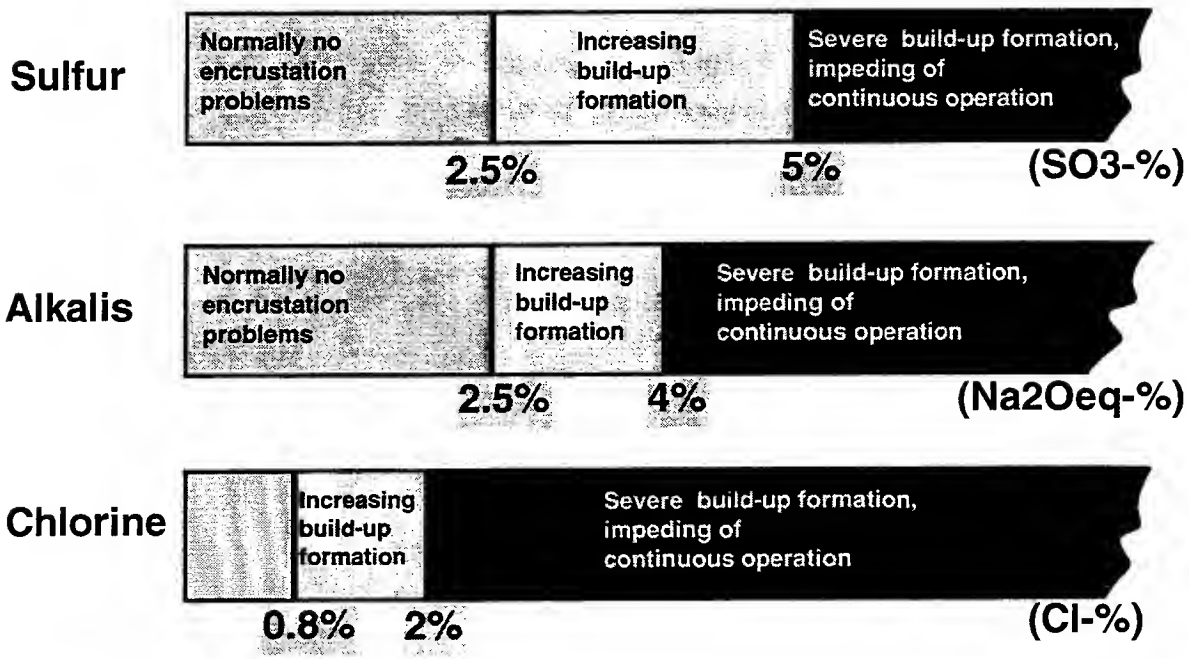
6.2.3 Enrichment of the Circulating Elements in the Hot Meal

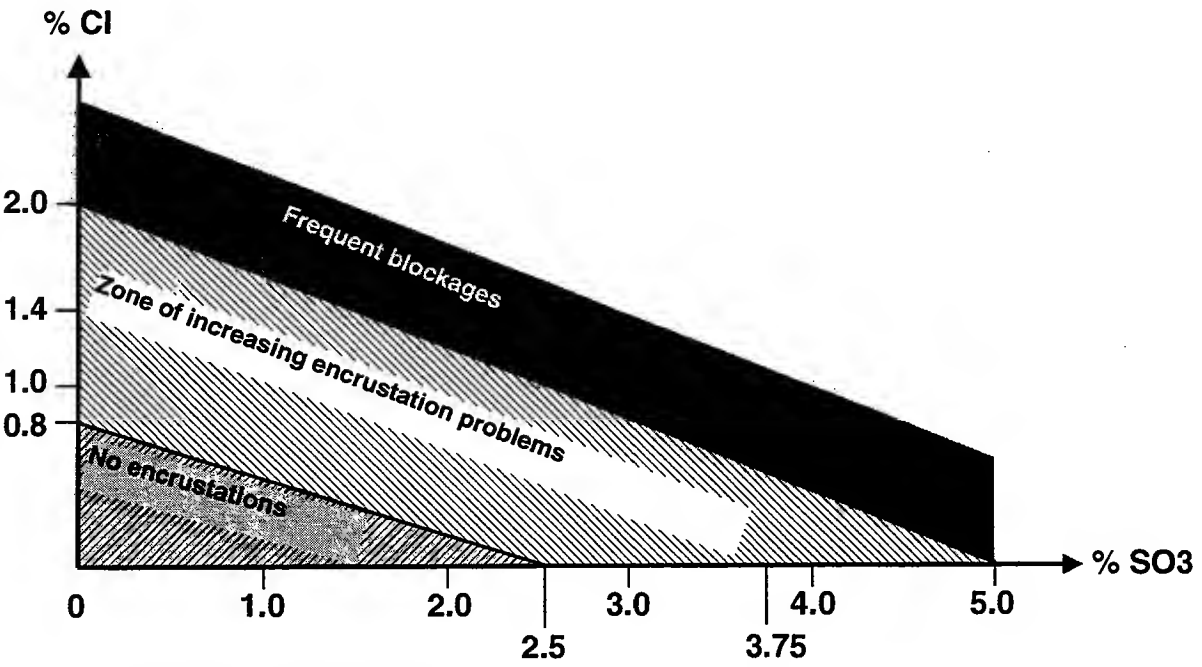
- Fig. 23 illustrates the tolerable concentrations of circulating elements in the „hot meal", i.e. in the meal ex lowest cyclone stage.

Fig. 23 Concentrations in the Hot Meal



- Fig. 24 considers the fact that normally both chlorides and sulfates are present in the hot meal and that both together determine the tolerable limits of concentrations. The actual case shall also be compared with this diagram.

Fig. 24 Maximum Tolerable Concentrations in the Hot Meal



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6.2.4 Total Sulfur Volatility

- With the formula for the total volatility (see section 3.4.1) the individual total volatilities of the sulfates shall be calculated.
  - If φ of the sulfur is > 0.7 and if at the same time the A/S ratio is within the range of 0.8 to 1.2, the sulfur cycle is definitely too high and needs improvements by measures as stated in section 7.4.

6.3 Example of a Circulation Phenomena Problem

6.3.1 Material Balance and Hot Meal Analysis

Material Balance			
Inputs			
Raw mix:	SO_3	=	0.5% cli
(loss free basis)	K_2O	=	0.5% cli
	Na_2O	=	0.2% cli
	Cl	=	0.05% cli
Coal:	S	=	2%, SO_3 = 5%
	Hu	=	26'800 kJ/kg, 6400 kcal/kg
Specific heat consumption:	q	=	3350 kJ/kg, 800 kcal/kg
Outputs			
Clinker:	SO_3	=	0.85%
	K_2O	=	0.5%
	Na_2O	=	0.2%
	Cl	=	0.03%
SO2 emission:	SO_2	=	400 mg/Nm^3 (= 0.1% SO_3)
Hot Meal Analysis			
(loss free basis)	SO_3	=	4.5% cli
	K_2O	=	4% cli
	Na_2O	=	0.3% cli
	Cl	=	2% cli

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6.3.2 Analysis of the Example

6.3.2.1 Material Balance

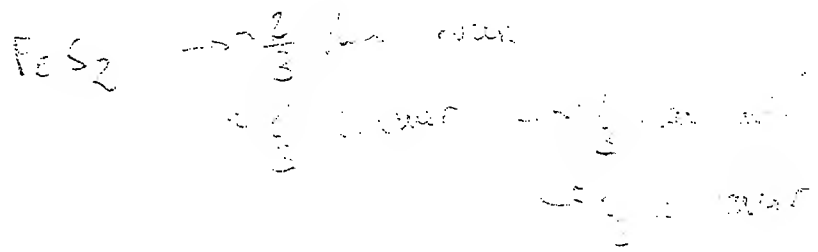
Material Balance		SO_3 [% cli]	K_2O [% cli]	Na_2O [% cli]	Cl [% cli]
Input	Raw Mix	0.5	0.5	0.2	0.05
	Coal	0.6			
	Total	**1.1**	**0.5**	**0.2**	**0.05**
Output	Clinker	0.85	0.5	0.2	0.03
	SO_2 emission	0.1			
	Total	**0.95**	**0.5**	**0.2**	**0.03**
Balance Error		**0.15**	**0.0**	**0.0**	**0.02**

- The total sulfur output is smaller than the total input. This often is the case when performing a material balance. There are two possible explanations:
  - The samples are not representative.
  - The process was not in a steady state. Sulfur was kept back in the system, forming encrustations. During cleaning, the sulfur containing deposits left the kiln with the clinker but were not catched representatively by the sampling procedure.
- The total chloride output is smaller than the total input. Here the same problems arise as described above. Normally the clinker does not contain more than 0.01% Cl due to the high chloride volatility. However, if the chloride cycle becomes very large, values up to 0.05% Cl in the clinker have already been measured.

6.3.2.2 Form of Sulfur in the Raw Materials

Note that the raw material contains sulfidic sulfur (FeS_2) due to the SO_2 emission. The real sulfate input to the kiln is therefore:

from raw mix to kiln	0.5 to 0.1 =	0.4% SO_3
coal	+	0.6% SO_3
total input to the kiln		1.0% SO_3



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6.3.2.3 Comparison with Typical Inputs of Circulating Elements (see section 6.2.2)

Inputs by the raw materials:

SO_3 = 0.4% (sulfates only!)	→ o.k.
K_2O and Na_2O	→ o.k.
Cl = 0.05%	→ very high, bypass required

Sulfur inputs by the coal:

S = 2%	→ high, but within the normal range

6.3.2.4 Alkali / Sulfur Ratio

$$A/S = \frac{\frac{\%K_2O}{94} + \frac{\%Na_2O}{62} - \frac{\%Cl}{71}}{\frac{\%SO_3}{80}} = \frac{\frac{0.5}{94} + \frac{0.2}{62} + \frac{0.05}{71}}{\frac{1.0^{*)}}{80}} = 0.63$$

*) total input to the kiln (see section 6.3.2.2)

⇒ The A/S ratio is outside of the desirable range of 0.8 to 1.5.

6.3.2.5 Enrichment of Circulating Elements in the Hot Meal

- Comparison with tolerable concentrations of circulating elements (see section 6.2.2)

SO_3 = 4.5%	→ High, far beyond normal limit, problems with encrustation
Alkalis: 0.66 · 4 + 0.3 =2.9%	→ Slightly above normal, but highly desirable due to the large sulfur input!
Cl = 2%	→ High, far beyond normal limit, problems with encrustations

- Comparison with diagram for maximum concentrations (see section 6.2.2)

SO_3 = 4.5%
Cl = 2%
→ frequent blockages to be expected

6.3.2.6 Total Sulfur Volatility (SO_3)

$$\varphi = 1 - \frac{c_{cli}}{c_{HM}} = 1 - \frac{0.85}{4.5} = 0.81$$

$\varphi > 0.7$: → Because of the low A/S ratio, a substantial portion of the sulfur is in form of $CaSO_4$ which leads to the high volatility of > 0.7.

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7. MEASURES AGAINST BUILD-UP FORMATION

7.1 General Measures

- Reduction of the Inputs of Circulating Elements
  - The most obvious measure against build-up formation is to reduce the input of circulating elements. It is normally not possible to change the main raw materials. However, sometimes a minor component that contains a substantial amount of circulating elements can be replaced. In most cases the sulfur content of the fuel is more easy to change. Please note that the fuel may also contain considerable amounts of chlorides.
  - The circulating elements of the raw materials in the quarry deposits are often not homogeneously distributed. In such cases prehomogenization of the raw materials combined with selective quarrying helps to reduce peak inputs of circulating elements into the kiln system.
- Smooth Kiln Operation
  - In many cases the cycles of circulating elements are frequently increased by an inadequate kiln operation applying to drastic changes in fuel feed and draft.
  - Also the best kiln operator cannot apply during his daily work of 8 h such a super constant kiln operation. Therefore the author is of the strong opinion that only a fully automatic kiln control will provide optimal results (cp. LINKman).

7.2 "Intelligent" Cleaning

Important: The more circulating elements are introduced into the kiln system the better and more efficient cleaning methods have to be applied!

- Preheaters have to be controlled and cleaned at least once a shift to remove immediately possible build-ups! Therefore a experienced tower guard is needed.
- On each platform enough cleaning slots are required to manually clean the riser duct and the inlet chamber whenever it is necessary. At particularly critical spots additional slots should be installed.
- Critical locations where always material deposits are built shall be armed with air blasters. Please note that mostly several blasters for one location must be installed since their cleaning radius is rather small (< 0.5 m). The blasters should always shoot in the direction of material flow (see Fig. 25).

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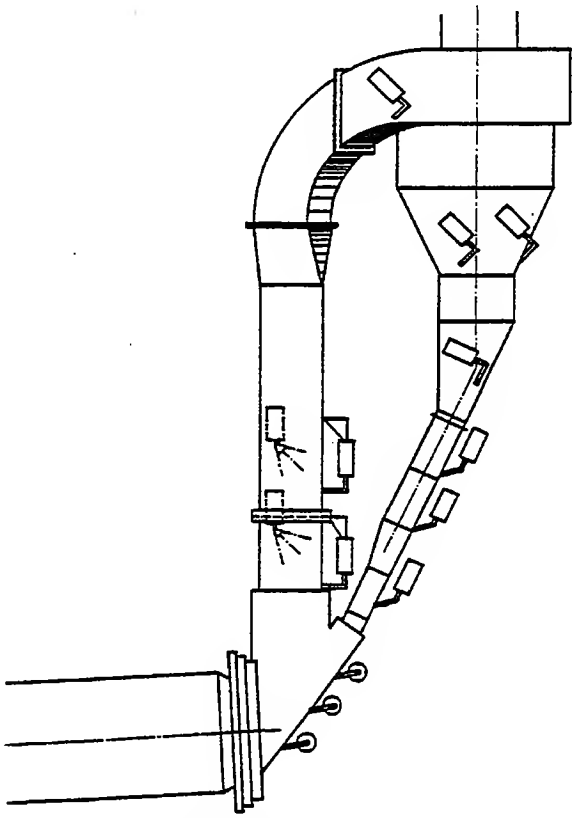
Fig. 25 Application of Air Cannons



Typical clogging areas of SP kiln :

- **inlet chamber**
- **riser duct**
- **cyclone inlet**
- **cyclone meal duct**

- The riser ducts shall be protected by introducing the raw meal from the upper stage as low as possible (Fig. 26). That way the circulating elements are able to condense at the cooler meal.

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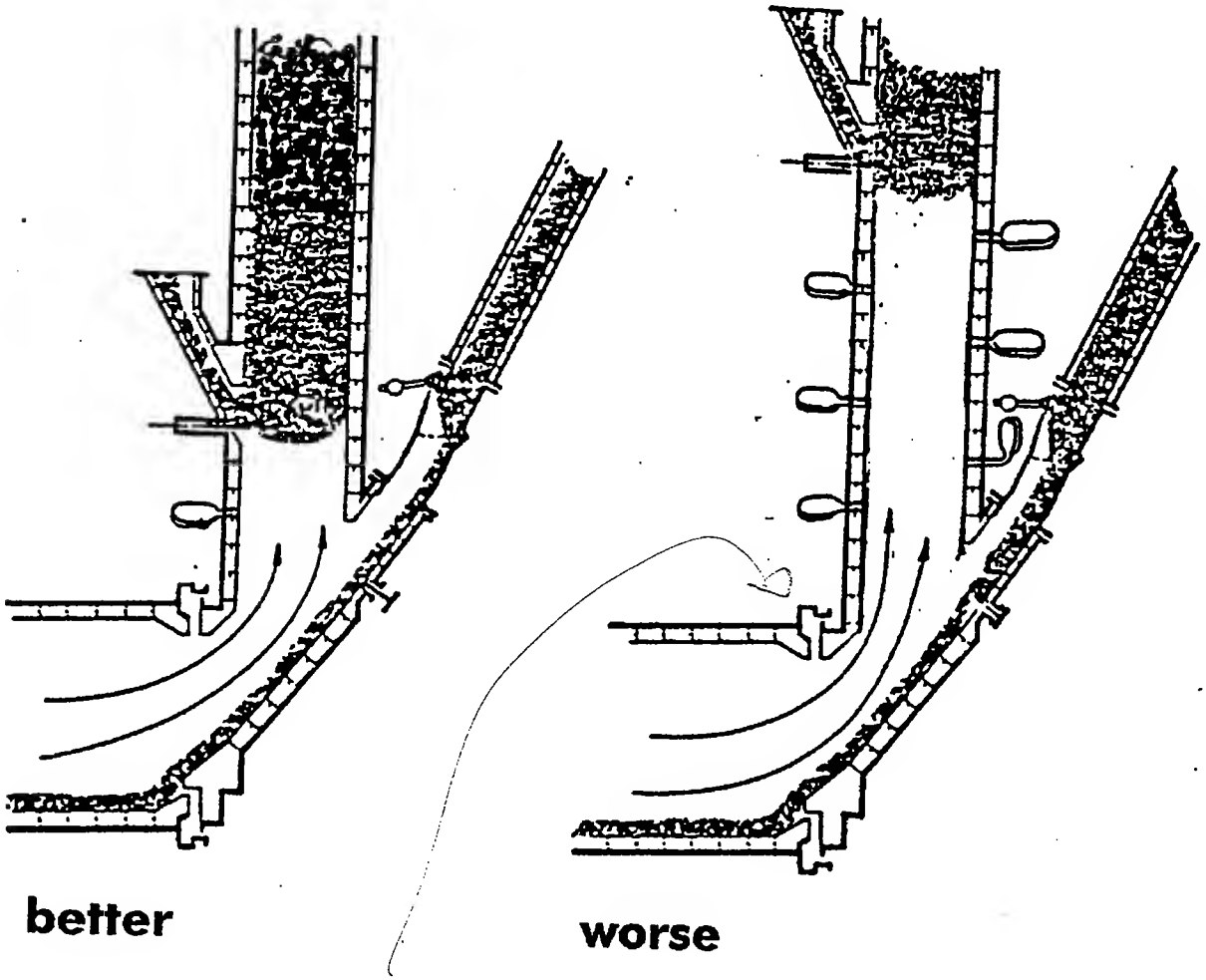
Fig. 26 **Protection of the Riser Duct Walls against Condensation of Circulating Elements**



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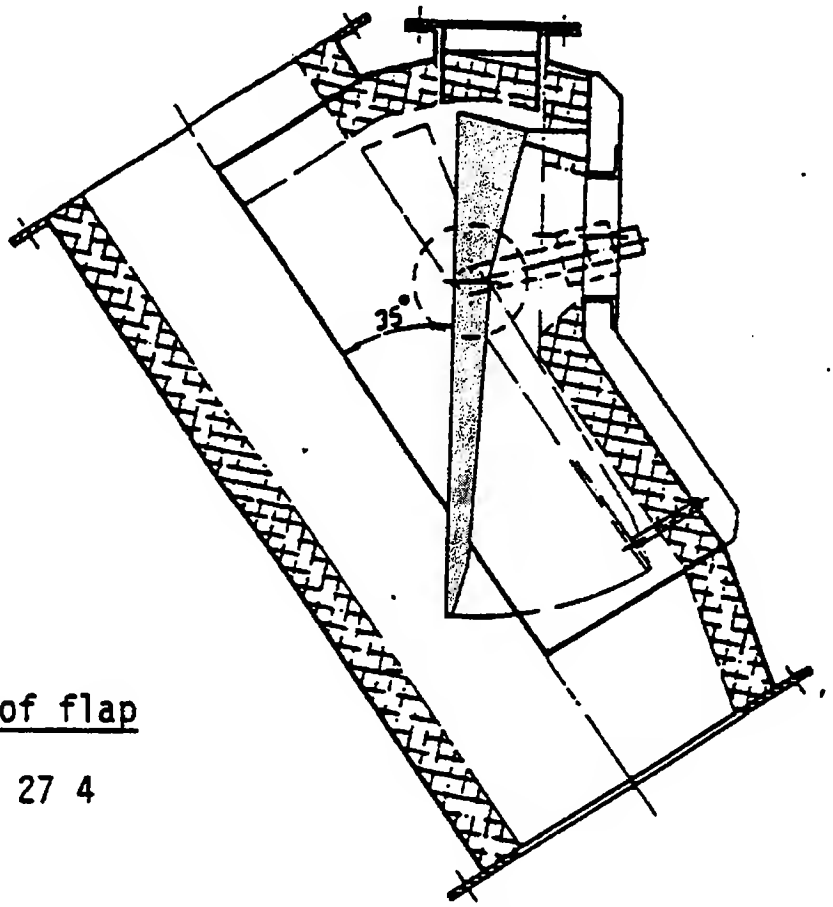
The meal ducts should be equipped with pendulum flaps in order to avoid a hot gas bypass through the ducts into the cyclones (Fig. 27).

Fig. 27 Example of a Pendulum Flape for the hot meal duct of the lowest cyclone in a suspension preheater (Polysius)



- Meal ducts have to have a sufficient inclination (> 55°), a sufficient large diameter and must not have any sharp bends.
- The cyclone outlets for the hot meal should be sufficiently large. If frequent bridging of the outflowing raw meal occurs, the outlet section shall be first equipped with air blasters and, if this measure does not help, it must be enlarged.
- The installation of coating neutral refractory at the transition chamber and the lowest riser duct has produced good results in many applications and is state of the art.
- Any false air entrance at the kiln inlet area and the lower preheater part must be avoided (continuous supervision and maintenance). Otherwise false air is likely to increase the formation of build-ups.
- For efficient cleaning special high-pressure water pumps (e.g. type WOMA) shall be applied. The application of this pump is dangerous and needs special training of the operators. There is also a danger of destroying the refractory lining!

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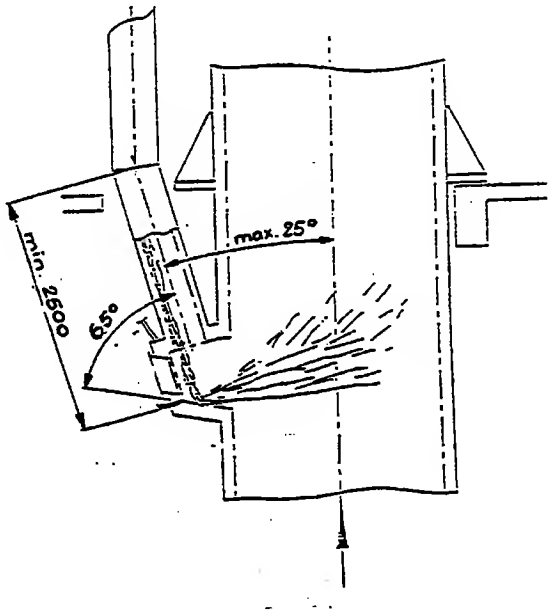
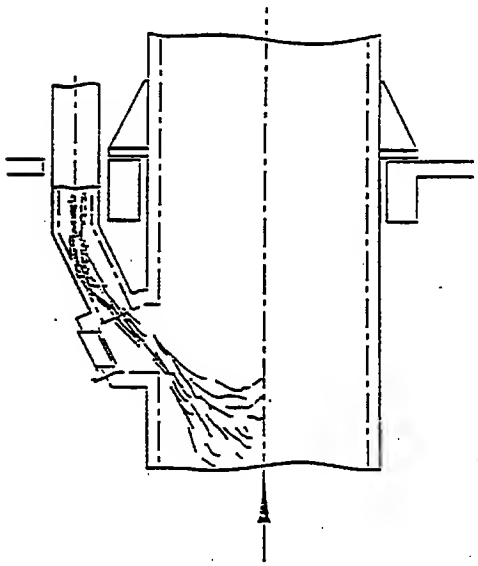
- Adequate meal distribution boxes shall be installed, which distribute the raw meal over the whole cross section. An example is given in Fig. 28.

Fig. 28 Meal Distribution Boxes by Polysius



CORRECT



WRONG

- Dead corners in riser ducts, cyclones and the transition chamber shall be avoided.
- Cleaning should only be done if the pressure profile or a quick inspection indicate a need for. Opening of the big access doors disturbs kiln operation and let enter cold air, which favors build-up formation.

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7.3 Measures against Chloride Problems

Important: With the clinker chlorine can only be withdrawn to a very limited amount (0.01 to maximum 0.02% Cl). Therefore the measures against chlorine build-ups are limited.

- Reduction of the Volatility
  - There is little chance to decrease the chlorine and chlorine volatility as evaporation takes place at a low temperature range (800 to 1000°C).
- Discarding of Dust (external cycle)
  - For long dry and wet kilns as well as for Lepol kilns the chlorides are effectively withdrawn from the kiln by discarding the finest fraction of the total of the kiln dust.
  - For preheater kiln dust discarding is normally not very efficient as the accumulation of chlorides in the dust is small. However, in case of a large internal chloride cycle this measure could help to keep it at a lower level. The effect of the measure can be calculated by a material balance.
- Reduction of the Sulfur Cycle
  - Build-up formation is provoked by all circulating elements together. As the chloride cycle cannot be effectively influenced, all possible measures should be taken to reduce the sulfur cycle as it increases the tolerance threshold for chlorine.
- Installation of a Kiln Gas Bypass
  - If the chloride input of a SP kiln is larger than 0.02 to 0.03% (loss free basis) a chloride bypass must be installed. Whether the higher limit of 0.03% can be tolerated depends on the expected sulfur cycle.
  - A rule of thumb to calculate the quantity of required bypass gases is:

% Bypass = % Cl input by the raw material (loss free) x 100

Example: % Cl input by the raw material = 0.05%
→ Required bypass size = 0.05 100 = 5%

  - The Bypass has to be installed at point of highest concentration in the gas of chlorine and alkalis.

7.4 Measures against Sulfur Problems

Important: In contrast to chlorine sulfur can be withdrawn in high quantities with the clinker. All measure aim to reduce the volatility in order to introduce the sulfur into the clinker.

- Reduction of the sulfur input (raw material and fuel)
- Adjusting of the molar alkali / sulfur ratio between 0.8 and 1.2; best to a value beyond 1. K_2SO_4 is little volatile, $CaSO_4$ is highly volatile and can be withdrawn only in form of double-salts or within the belite.
- Keep the lime saturation factor as low as possible so that the sulfur can dissolve in the belite.
- Possibly modification of the raw mix in order to improve the burnability and the granulation of the clinker (decrease of the silica ratio). These measures help to reduce the required maximum temperature in the burning zone and to reduce the sulfur volatilization from the clinker granules.
- Minimal fluctuations in the chemistry and the quantity of the kiln feed so that constant burning conditions can be maintained.

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- Avoid overburning! High sulfur content means a porous clinker. In this case not a high litre weight is needed to produce low free lime! For quality control drop litre weight or adjust regularly the rated value for the litre weight to the free lime values.
- High sulfur cycles produce a dusty kiln atmosphere. Do not burn down the dust at all costs in order to avoid overheating.
- Sufficient excess air at the kiln inlet to avoid reducing conditions (2% O_2). Please note that the measurement of O_2 at kiln inlet is a spot sample and does not necessarily represent the whole inlet cross section. The 2% O_2 can therefore be a guide value only.
  Attention: Too much excess air produces high kiln inlet temperatures, which again increase the sulfur volatility. Use carefully excess air! Do not tolerate any CO at the kiln inlet, i.e. CO < 0.05%.
- Introduce automatic kiln control (LINKman) in order to stabalize the kiln operation.
- The burner should be directed parallel to the kiln axis and should not point to the kiln charge in order to avoid local reducing burning conditions.
- The burner itself should be of the latest design, which produces a short and stable flame in order to have the shortest possible retention time of the kiln charge in the high temperature zone and low kiln inlet temperatures are achieved. As a result the sulfur volatility is minimized.
- Apply secondary firing or precalcination to reduce the thermal load in the sintering zone. This possibility is very limited when the kiln inlet temperatures are too high and reducing conditions occur. In this case heavy sulfur build-ups or even build-ups from the first clinker mineralization result.
- Very good dispersion of all liquid fuels in order to achieve a short flame (enough viscosity and atomizing pressure).
- Solid fuels (also alternative fuels) should be enough prepared. The main fuel at the main burner should have the fineness of coal! Use coarser fractions only limited in the main burner (10 to 20% heat). Coarse particles make the flame longer and so increase the sulfur volatility. Introduce additional coarse fuel fractions separate above the flame and not directly into the flame.
- If solid fuels (e.g. used tires) are burnt at the kiln inlet, lifters shall be installed in order to keep the fuel pieces at the surface of the kiln charge and to avoid local reducing burning conditions.
- Permanent control of pressure fluctuations in the pneumatic pipes at the burner head. Pulsation < ± 5 mbar! Avoid long transport pipe and keep the velocity > 30 m/s.
- In case of extremely high sulfur inputs and a alkali deficiency at the same time kiln gas bypasses are applied. But these are by far not as efficient as for chlorine problems.

  Remark: Especially for long dry and wet kilns discarding of dust is a very effective measure to withdraw the sulfur from the kiln. This does not hold true for SP kilns as the accumulation of sulfur in the external cycle is small.
  For SP kilns the installation of a kiln gas bypass may become necessary if there are almost no alkalis to extract the sulfur from the kiln.

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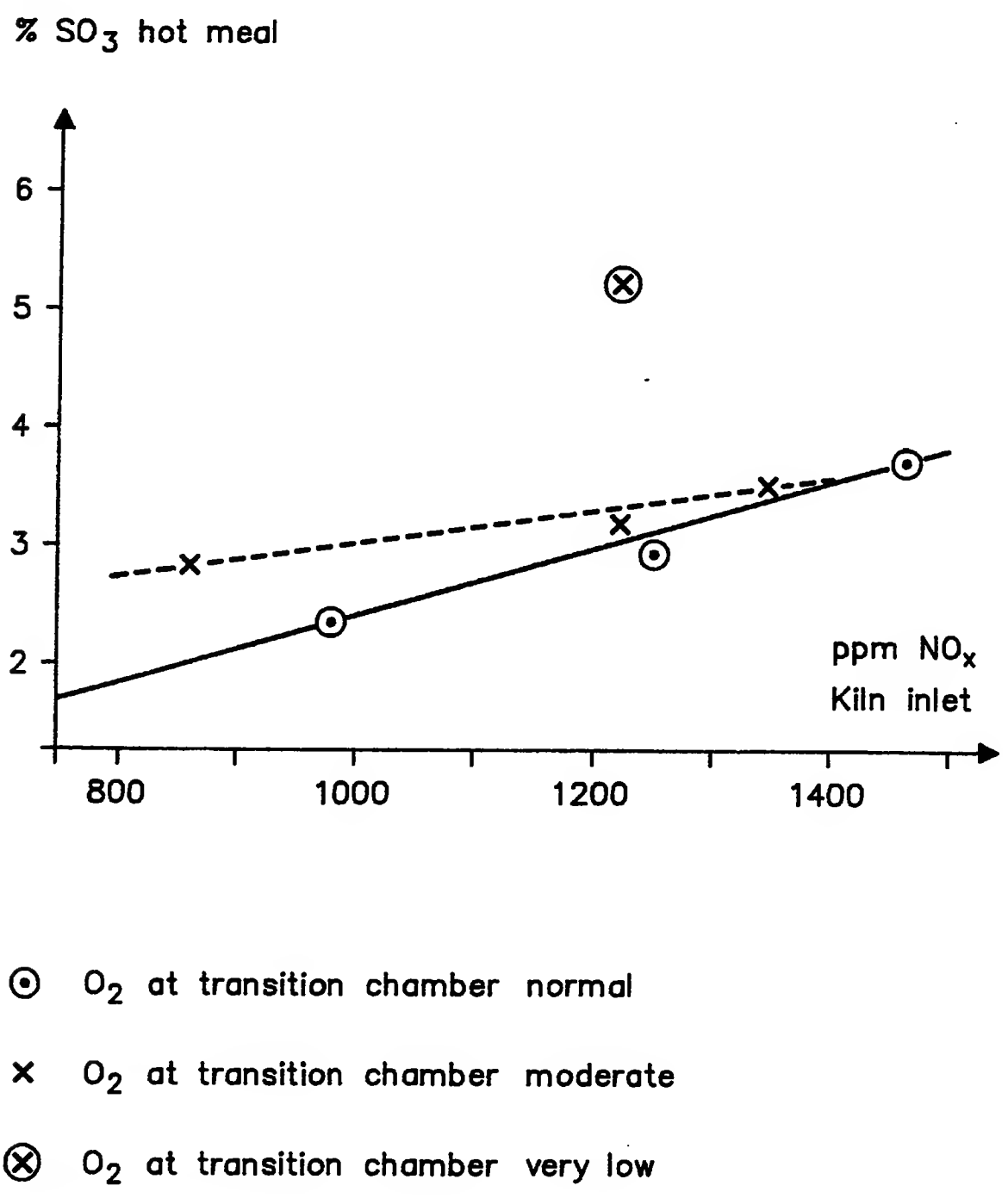
- Shifting of the condensation area of the sulfur towards the kiln inlet and the first zone of the rotary part.
  → This effect is achieved by increasing the A/S ratio and by avoiding high kiln inlet temperatures.
- Avoid overburning of the clinker, Fig. 29 demonstrates the strong influence of burning zone temperature (indicated by the concentration of NO_x on the accumulation of SO_3 and especially $CaSO_4$ in the hot meal. Furthermore also the influence of the excess air is shown.

Fig. 29 Enrichment of SO_3 in the Hot Meal of the Lowest Cyclone Stage



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7.5 Measures against Alkali Problems

- Alkali Volatility
  - In absence of sulfur the alkali volatility is very high and creates problems in the preheater. In such a case the raw mix can be sulfatizated by addition of gypsum.
- Low Alkali Clinker
  - If low alkali clinker must be produced, all measures must be taken to increase the alkali volatility, such as
    - reducing the sulfur input
    - producing a long and stable flame
    - applying hard burning ($CaO_{free} < 1$), if possible reducing the burnability by increasing the silica ration
    - applying a minimum of excess air
    - chlorination of the raw mix either by burning chloride
    - containing solvents or adding $CaCl_2$
- In case of long dry or wet kilns the alkalis are withdrawn by discarding a fraction or the total dust of the external cycle. In case of a SP kiln a (large) kiln gas bypass is required.

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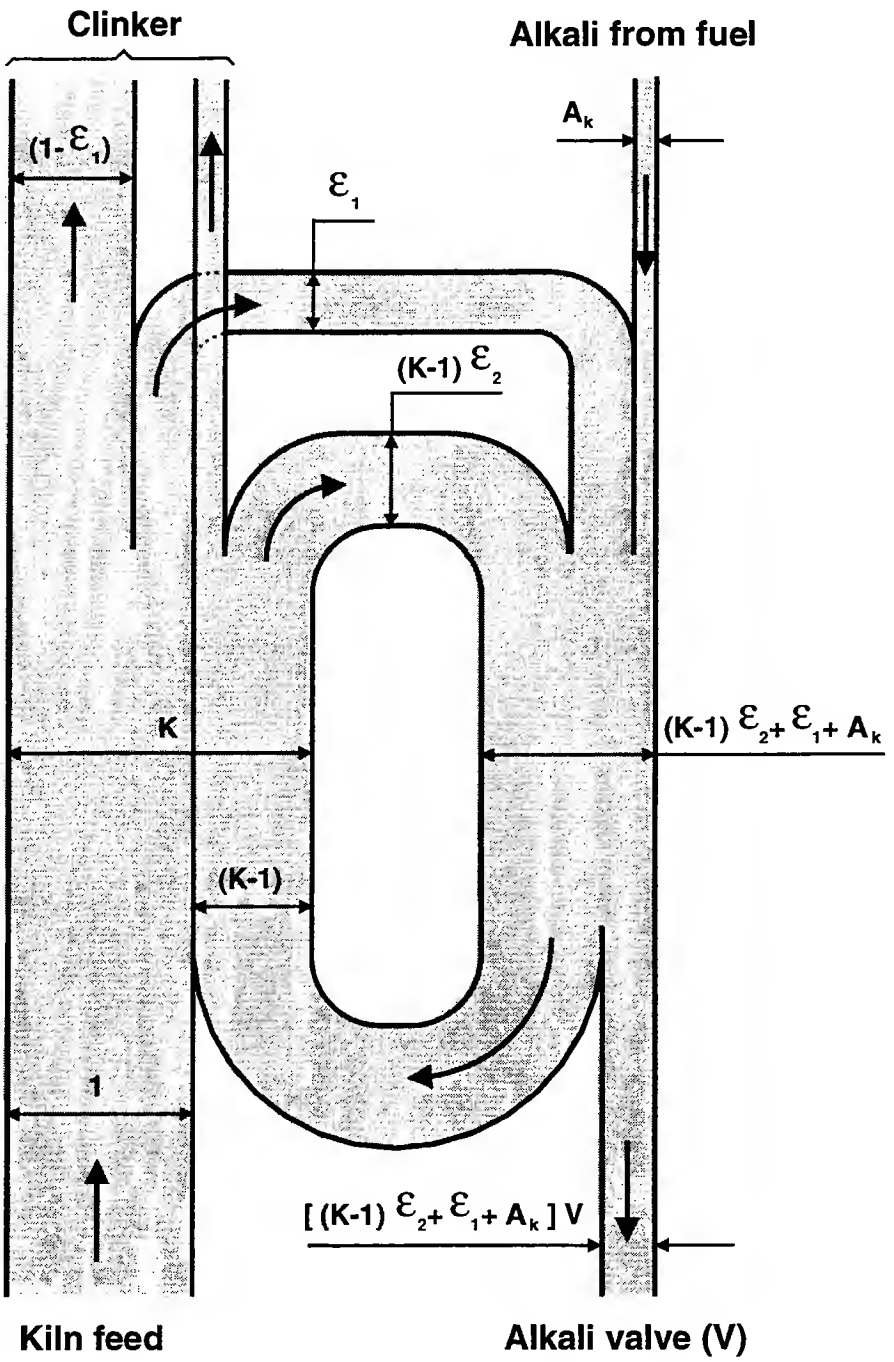
8. MATHEMATICAL MODEL TO SIMULATE THE CYCLES OF THE CIRCULATING ELEMENTS

The cycles of the circulating elements in the kiln system can be simulated by a mathematical model (Fig. 30). This model is used for bypass calculations and for simulating special effects on the behavior of the volatile cycles.

Fig. 30 **Mathematical Model to Simulate the Cycles of the Circulating Elements in the Kiln System (by Weber)**



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Chapter 9

Heat Balance

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Heat Balances of Kilns and Coolers and Related Topics

Th. Richner / A. Obrist
PT 99/14496/E (update of VA 8/5180/E)

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SUMMARY

A heat balance is an efficient tool to measure the actual state of a kiln system. It allows a better assessment of the heat consumption and reveals the potentials for improvements (regarding the thermal aspects).

The principle of a heat balance is rather simple: we select our system limits for the balance and measure all inputs and outputs.

After the necessary measuring campaign we need to calculate the various heat items, e.g.:

- Heat of fuel combustion
- Combustibles in raw meal or exhaust gas
- Heat of formation
- Heat of evaporation
- Sensible heats of all gas and mass flows
- Radiation and convection heat

The heat balance shows clearly how the heat is spent among the individual items. This information is therefore most suitable to detect abnormal operating conditions or potential for improvements. As a next step we can decide which practical measures should be taken in order to achieve improvements in the thermal energy consumption.

When improvements are realized we can often take into account that the saving of fuel is by a factor of 1.3 to 1.5 higher than the primary improvements on the balance item (multiplication factor), as long as we consider the high temperature zone.

Considerable improvements can be realized by reducing shell losses, false air inleaks, heat exchange in certain preheater types and in clinker coolers.

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1. INTRODUCTION

Heat balances on a kiln system can offer extremely useful information on the thermal performance of the system. Heat balances show where or how the fuel heat is consumed, based on the simple principle of:

input = output

Unnecessary energy losses can be easily detected.

The principle aim of this chapter is to serve a practical guide for doing heat balances on cement kilns. It can be used as working paper and does neither require special thermodynamic knowledge nor supplementary literature or tables.

For the more interested reader some special aspects are treated in a separate chapter.

The principle of heat balance may be easily transferred to other systems such as preheaters, coolers and drying systems. Therefore the use of this chapter can be extended to other systems than cement kilns.

In this chapter, only SI units are used, which means that heat is always given in kJ (kilo Joule). Conversion calculations within the chapter will no more be required. Incidentally this may also contribute to the consequent use of SI units.

2. OVERVIEW OF COMPLETE HEAT BALANCE PROCEDURE

2.1 Why to do a Heat Balance?

Various reasons or circumstances may cause a need for a heat balance measurement. The following situations may justify a heat balance:

- Performance test
- Recording of kiln performance before/after a modification
- Unusually high heat consumption or abnormal kiln operational data
- Kiln optimization campaign

It may be self explaining that an extensive heat balance also costs money, especially if a large number of people are involved. Therefore the costs may be put in relation to the obtainable benefits. A potential improvement of say 100 kJ/kg of heat consumption on a 3000 t/d kiln means a savings in the order of US$ 200'000 per year (at current fuel prices). In such a case it may be worthwhile to invest some money for a detailed investigation including a complete heat balance.

Although the specific heat consumption proper could also be determined by measuring nothing but fuel heat and clinker production, a complete heat balance does offer considerably more information and security. The consistency of the measured data is proved much better, and the balance shows clearly where the heat is consumed. A heat balance is obviously a very efficient tool for assessment of thermal efficiency.

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2.2 How to Proceed

A heat balance does not only mean calculation of heat balance items. The complete procedure usually includes the following steps:

2.2.1 1st Step: Preparation

The extent of works to be done depends on the completeness and reliability which is desired. A careful planning and preparation is recommendable. The following basic items must be clarified:

- What has to be measured (kind and location of measuring / sampling points)?
- Duration of test?
- Frequency of measurements (continuous recording, spot measurements, time intervals, etc.)?

Under above preconditions the number of people required and the necessary measuring equipment can be determined. Temporary equipment may usually become necessary, whereas the existing permanent instrumentation should be carefully checked and calibrated.

The following list may be used as checklist for a test preparation:

- People available for test period?
- Does everybody know what to do at what time?
- Necessary logsheets for manual recordings ready?
- Data recording system (electronic, pen recorder, etc.) available?
- Flow of information among test team?
- Calibration or checks of instruments and scales done (flowmeters, orifice plates, venturis, dp-cells, thermocouples, etc.)?
- Temporary measuring equipment available? Complete? Correct span? Functionning properly?
  For example:
  - pitot tube
  - U-tube manometer / electronic manometer
  - mobile thermometer
  - radiation pyrometer
  - gas analyzer / Orsat
  - sampling equipment for gas
- Sampling procedure for solids (e.g. meal, dust, clinker) clarified?
- Analyzing facilities?

2.2.2 2nd Step: Execution

An important precondition for a good test is a steady kiln operation. The test should only be started if the system has reached a constant equilibrium state. During the test, variations of operating parameters should be avoided.

For the measuring techniques reference is made to the corresponding chapter.

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It is recommended to check completeness and reliability of measurements already during the test, afterwards missing or uncertain information may create problems at the final evaluation.

2.2.3 3rd Step: Evaluation

This step is the main scope of this chapter. Evaluation of data means to establish a heat balance calculation according to the principle

"input = output".

2.2.4 4th Step: Discussion

A heat balance as such must bring some practical conclusions otherwise it would be only of academical interest. The following items may be considered:

- Acceptable (normal) heat consumption?
- Are heat balance items normal for given kiln systems?
- Measures in order to improve heat economy of the system?

The last item can become quite an extensive work and the economical feasibility must be considered as well. Such subjects, however, are beyond the scope of this chapter.

Note: Summarizing, it is obvious that the 3rd step „Evaluation" is only a limited part in the whole context.

3. BASIS OF BALANCE AND REMARKS REGARDING EXECUTION

3.1 Balance Limits

In many balance reports the boundary for the balance is not shown expressively. As long as „standard cases" are considered it may sometimes seem evident how the balance limits were selected. Nevertheless it is advantageous to indicate always clearly where the boundary for the balance is. This will avoid misunderstandings and reduce error possibilities. For that purpose a (simplified) flowsheet is required. In this scheme the boundary must be marked by a line which envelopes the system under consideration. When a measuring campaign in a plant is planned such definitions should be made already in the planning phase, i.e before the test.

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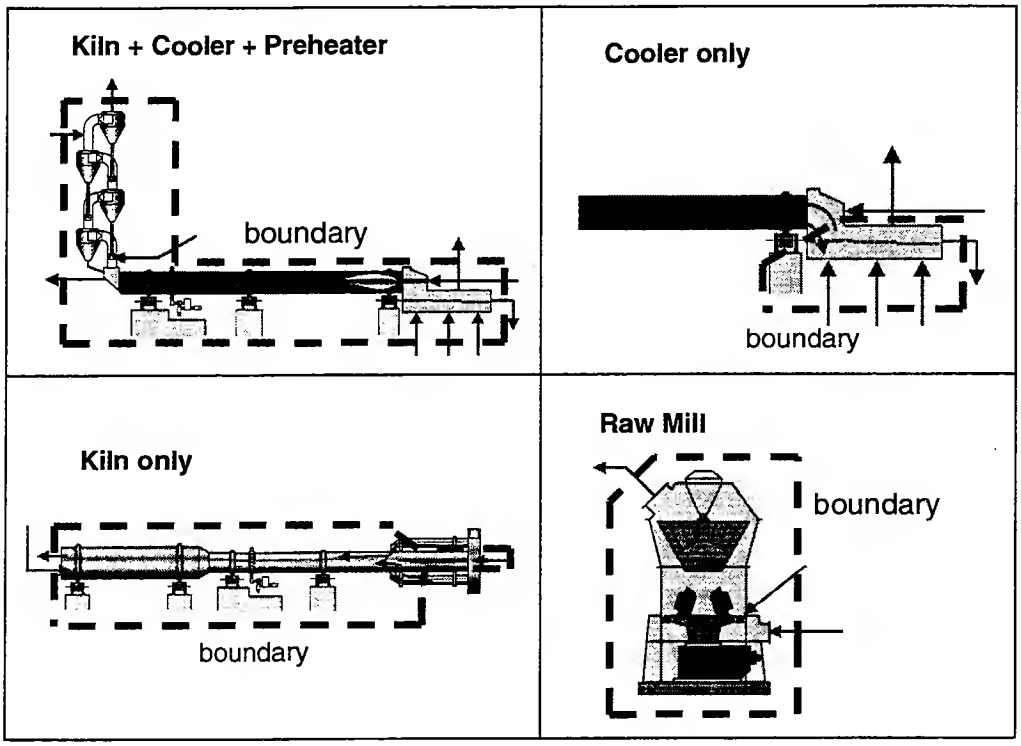
Fig. 1 Examples of Balance Boundaries



The boundary generates various cutpoints with ingoing or outgoing solid- and gas streams. Likewise radiation and convection heat crosses the boundary.

Each cutpoint means a certain item in the heat balance because it represents a heat flow either into or out of the system.

By this measure the system is clearly defined and nothing can be forgotten.

Basically, any shape of the boundary could theoretically be chosen. To give an extreme example: The boundary could even cut a rotary kiln at half length! However, the boundary must be selected according to practical considerations. This means that the cutpoints which are generated must be:

- easily accessible for reliable measurements
- of practical interest in the whole context.

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3.2 Guidelines for Test Duration

A long test duration would allow for a good accuracy but the available time is limited by practical considerations.

As a rough guideline the minimum test duration should be about ten times the material retention time in the complete kiln system:

- test duration = 10 x retention time

Other factors like regularity of kiln operation also influence the test duration. If the process is very unsteady longer times should be envisaged.

On a normal suspension preheater kiln the retention time of the material is in the order of one hour. In contrast, on a lime shaft kiln retention times above 24 hours may occur. Therefore the necessary test durations for these two cases must be completely different.

Although test durations must be set individually the following list may be used as rough guideline:

Type of Kiln	Test Duration (hours) for Heat Balance
Suspension preheater kiln with precalciner	12
Suspension preheater kiln without precalciner	12 to 24
Long dry / wet kiln	24

3.3 Kiln Operation

During the test, the kiln must run at constant and steady conditions. Changing of setpoints should be avoided whenever possible.

Interruptions have to be logged. If serious problems occur, the test has to be extended or even postponed. Therefore it is often worthwhile to plan a certain time reserve.

From a theoretical point of view a proper balance can only be made if the system runs at steady conditions. E.g. during heating-up heat is stored in the system and there is no balance between input and output (input > output).

3.4 Kiln Data

In order to facilitate the final discussion it is usually necessary to collect the main data of the system such as:

a) System
   - process
   - type of kiln
   - nominal capacity
   - type of preheater / precalciner
   - type of cooler
   - supplier
   - year of commissioning
   - fuel and firing system
   - type of burner nozzle
   - dust reintroduction system
   - dimensions of main equipment (sizes, inclinations, etc.)
   - data on fans, derives, etc.

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b) Operation
- * various operating data (rpm, kW, temperature and pressure profiles along kiln system, grate speed, undergrate pressures, etc.)
- * electric power readings (before / after test)
- * chemical analysis of raw meal, dust(s) and clinker, LSF, SR, AR, etc.

Above data are not necessarily required for heat balance calculations proper, but they should be included in a complete balance report in order to describe the system and to give more information on its performance.

4. HEAT BALANCE CALCULATIONS

4.1 General Remarks

4.1.1 Symbols and Units

A		m^2	area
C_R		W/m^2K^4	radiation constant
c_P		kJ/kg C	specific heat (at const. pressure),
	or	kJ/Nm^3 C	specific heat capacity
CV		kJ/kg	calorific value
D		m	diameter
g		m/s^2	gravity constant
h		kJ/kg	heat content (specific)
	or	kJ/Nm^3	
	or	kJ/kg cli	
L		m	length
m		kg	mass
	or	kg/kg	specific mass
m_f		kg/h	mass flow
Q_f		kW	heat flow (1 kW = 1 kJ/s)
t		C	temperature (Centigrade)
T		K	temperature (Kelvin)
v		m/s	velocity
w		kg/kg	water content
Greek Letters			
α		W/m^2K	heat transfer coefficient
ε		-	emissivity (for radiation)
λ		W/m C	heat conductivity
ρ		kg/m^3	density

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Dimensionless Numbers

Nu	Nusselt number (for heat transfer)
Pr	Prandtl number
R	kiln feed (raw meal) / clinker-ratio
Re	Reynolds number

Indices

conv	convection
rad	radiation
tot	total
o	ambient condition or zero condition

Conversion Factors

Length	1 inch	0.0254 m
	1 ft	0.3048 m
Area	1 sq. ft	0.092903 m^2
Volume, Volume Flow	1 cu.ft	0.028316 m^3
	1 cu.ft/min	1.699 m^3/h (actual m^3)
Mass	1 lb.	0.45359 kg
	1 short ton (USA)	907.185 kg
Pressure	1 bar	10^5 N/m^2
	1 mm H_2O-Col.	9.806 N/m^2
	1 atm.	1.013 bar
Energy	1 kJ	1000 J
	1 MJ	1000 kJ
	1 kWh	3600 kJ
	1 kcal	4.187 kJ
	1 BTU	1.055 kJ
Temperature Conversion	C =	5/9(F - 32)
	K =	273.15 + C
Heat Flow	1 kW	1000 W = 1 kJ/s
	1 kcal/h	1.163 W
	1 BTU/h	0.29307 W
Specific Heat	1 kcal/kg C	4.187 kJ/kg C = 4187 J/kg C
	1 BTU/lb F	1 kcal/kg C = 4.187 kJ/kg C
Heat Transfer Coeffic.	1 kcal/m^2h C	1.163 W/m^2 C
	1 BTU/ft^2h F	5.678 W/m^2C
Standard Conditions for Gases	Standard Conditions	0°C and 1 atm (1.013 bar)

$$Nm^3 = act.m^3 \times \frac{273.15}{273.16 + t(C)} \times \frac{p(bar)}{1.013bar}$$

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4.1.2 Reference Temperature

We will set the usual reference temperature to 20°C i.e. the sensible heat of mass flows at 20°C becomes zero.

All „heats of transformation" such as combustion, evaporation, formation are also based on 20°C reference.

4.1.3 Input / Output

Whether a heat item represents an input or output is determined by the direction of mass flow, according to:

- into boundary = input
- out of boundary = output

This is a simple and useful convention (not a natural law).

If „heats of transformation" occur within the boundary the net heat effect is normally used as criterion:

- heat producing process = input
- heat consuming process = output

A heat producing process is e.g. the fuel combustion (exothermic).

4.1.4 Reference Quantity

The heat balance is referred to 1 kg of clinker produced. This requires a general calculation step (division by clinker production) which is not shown in the following formulas in order to maintain a more simple presentation.

4.2 Determination of Clinker Production

4.2.1 General

The clinker production during a test is a key figure for all the following calculations. We will refer all flows and heats to 1 kg of clinker produced.

Determination of clinker production can be made by direct weighing of the production which is by far the best method. If this can be done the following chapter 4.2.2 can be ignored. But in some cases no direct measurement is possible. Therefore, an indirect determination may be necessary as shown in the following.

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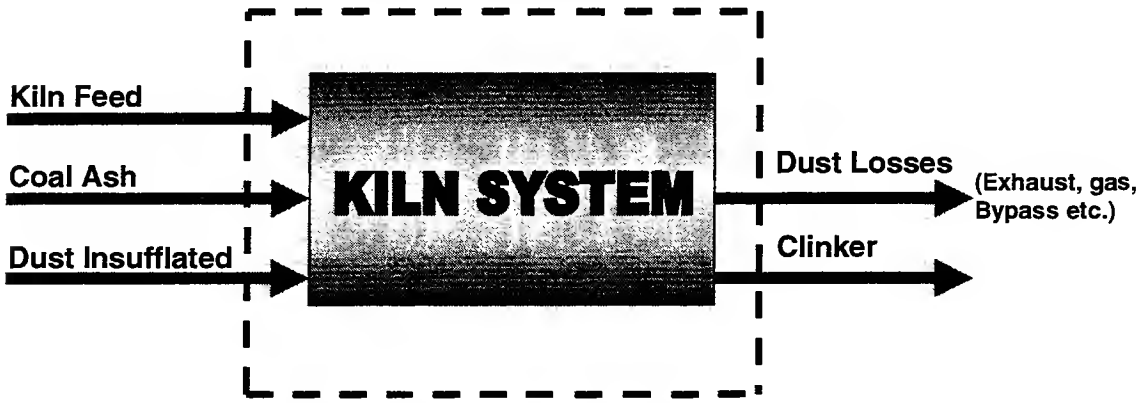
4.2.2 Indirect Determination

It is well known, that from approx. 1.56 kg raw material (not kiln feed!) 1 kg clinker can be produced. Starting from this fact it seems to be easily possible to calculate the clinker production. However, the above factor of 1.56 if of limited practical help because it can be superimposed by dust return and depends on some other influences.

The best method in such a case is to establish a mass balance for the system. Similar to a heat balance a boundary for a mass balance can be defined. Thus the clinker production becomes:



Boundary for mass balance

$$\text{Clinker} = \left\{ \begin{array}{l} + \text{ kiln feed} \\ + \text{ coal ash} \\ + \text{ dust insufflated} \\ - \text{ dust losses} \end{array} \right\} \quad \text{all calculated on L.o.I. free basis!}$$

This principle is quite simple. The following additional remarks can be given:

- The boundary for the mass balance does not necessarily have to coincide with the heat balance boundary.
- The mass balance boundary must cut the kiln feed measuring point (because the mass flow is known at this point).
- The balance is always made on a loss-on-ignition-free (L.o.I.-free) base (no balance for the entire quantity including L.o.I. can be made since part of it is converted to gas, which is not included here).

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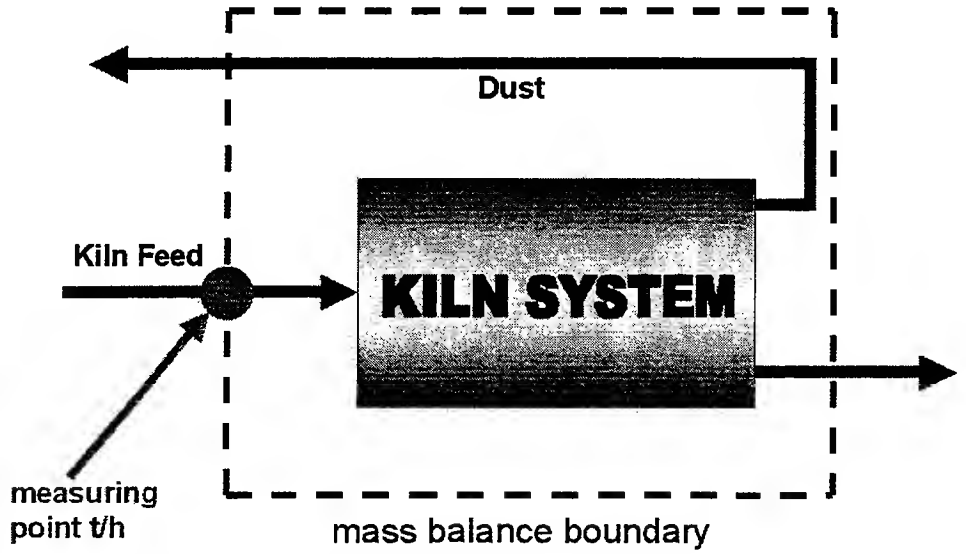
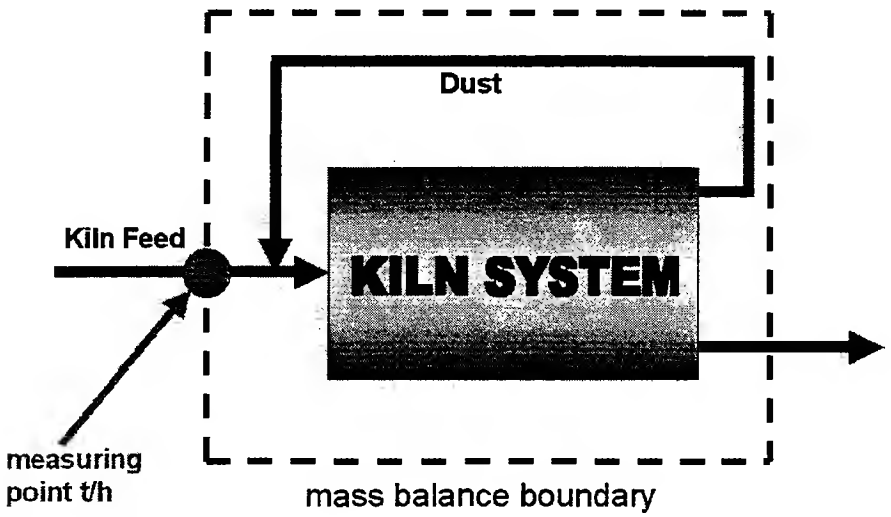
Regarding only the dust loss from the exhaust gas there are two basic possibilities:

a) Dust Flow crosses mass balance boundary



- mass balance is influenced by dust flow
- note that is not significant whether or how the dust is returned (outside the boundary)!

b) Dust flow does not leave mass balance boundary (internal dust return)



- mass balance is not influenced by dust flow
- internal dust return must be steady, i.e. no storage being built up (silos!)

Two practical examples illustrating above two cases are shown in the following.

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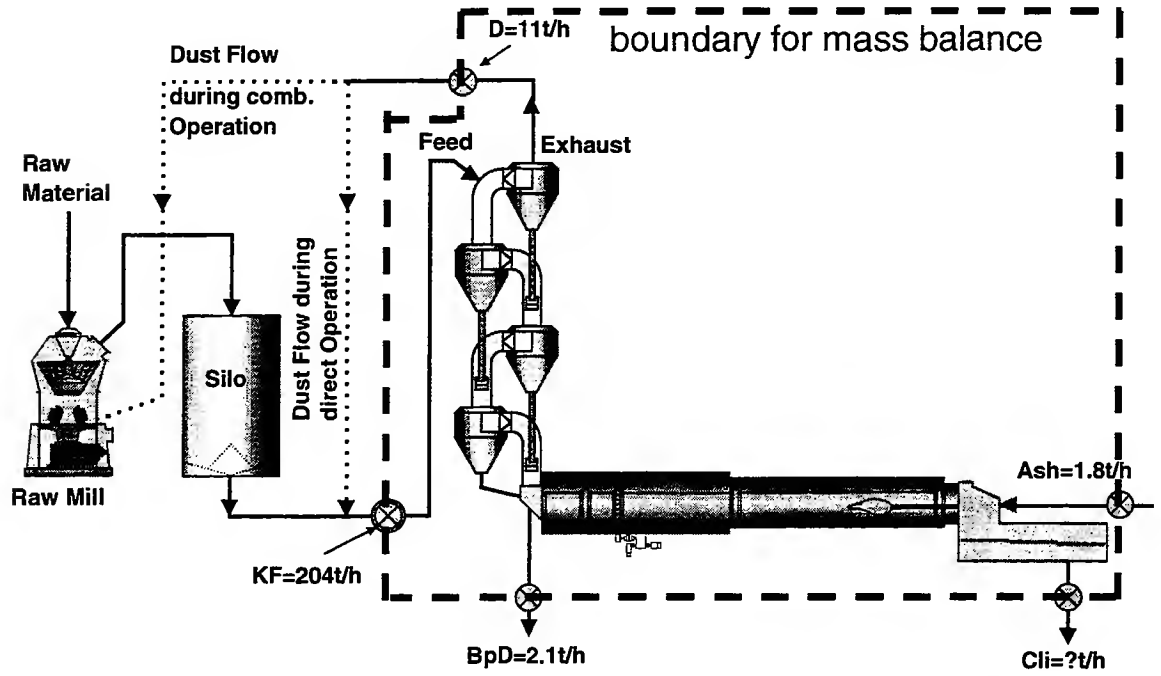
4.2.3 Examples

a) External Dust Recirculation (outside of boundary)

Fig. 2 Mass Balance / External Dust Recirculation



	m_f (dry) [t/h]	L.o.I. [-]	m_f (1-L.o.I.) [t/h]
Kiln Feed (KF)	204.0	0.357	131.17
Coal Ash (Ash)	1.8	~ 0	+ 1.80
Dust in Exhaust (D)	11.0	0.315	- 7.54
Bypass Dust (BpD)	2.1	0.070	- 1.95
Clinker (Cli)		~ 0	**123.48 t/h**

Note that the kiln feed data (L.o.I.) may slightly change when going from combined operation to direct operation or vice versa!

Kiln Feed / Clinker = R = 204.0 / 123.48 = **1.652** (all included)

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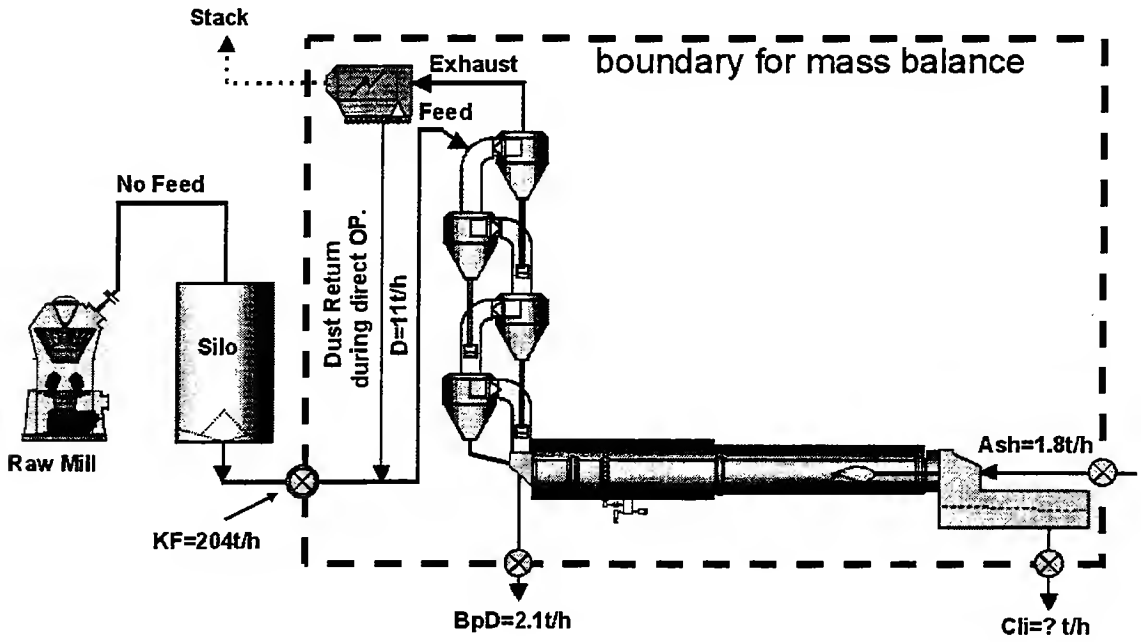
b1) Internal Dust Recirculation (inside of boundary)

Fig. 3 Mass Balance / Internal Dust Recirculation



During direct operation the following calculation method applies:

	m_f (dry) [t/h]	L.o.I. [-]	m_f (1-L.o.I.) [t/h]
Kiln Feed (KF)	204.0	0.357	131.17
Coal Ash (Ash)	1.8	~ 0	+ 1.80
Bypass Dust (BpD)	2.1	0.070	- 1.95
Clinker (Cli)		~ 0	**131.02 t/h**

Kiln Feed/Clinker = R = 204.0/131.02 = **1.557**

Note the difference to example a)!

Incidentally the example b1) is shown for comparison to example a) and does by no means represent a superior solution of the dust return!

The following example b2) shows a very similar way of calculation to example b1).

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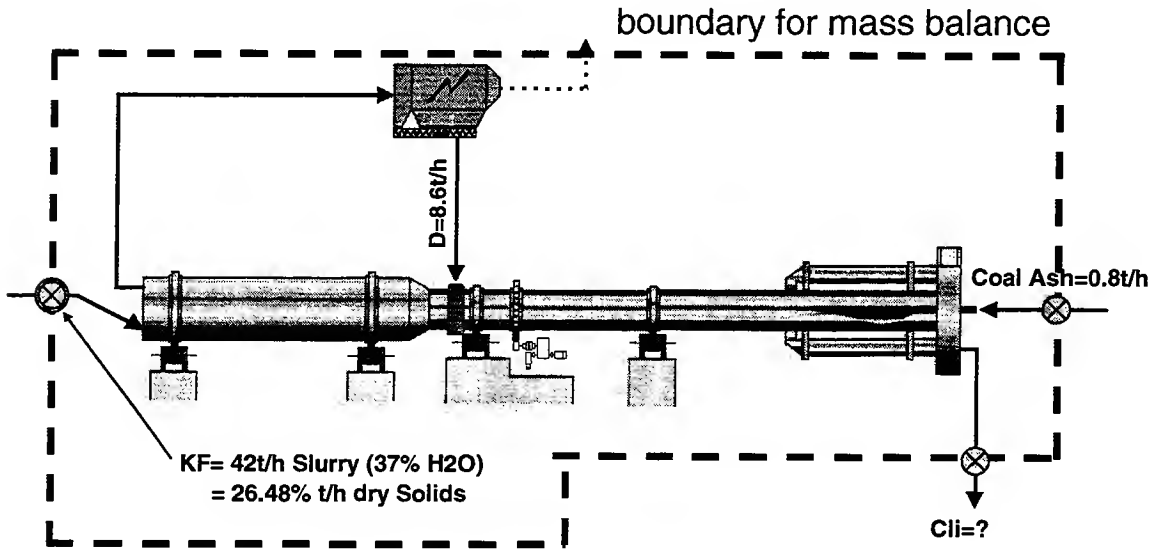
b2) Internal Dust Return (wet kiln)

Fig. 4 Mass Balance / Internal Dust Recirculation



	m_f (wet) [t/h]	m_f (dry) [t/h]	L.o.l. [-]	m_f (1-L.o.l.) [t/h]
Kiln Feed (KF)	42	26.46	0.360	16.93
Coal Ash (Ash)	-	0.8	~ 0	0.8
Clinker			~ 0	**17.73 t/h**

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4.3 Heat from Fuel

4.3.1 Fuel Firing

Fuel can be introduced at the following locations:

- Kiln firing
- Secondary firing or precalciner
- Burnable components in raw meal

Generally the heat from fuel is calculated:

$$h = m \cdot CV \qquad \text{(kJ/kg cli)}$$

h = heat (kJ/kg cli)

m = specific fuel consumption (kg/kg cli or Nm^3/kg cli)

CV = calorific value (kJ/kg fuel or kJ/Nm^3 fuel)

For the calorific value CV only the low (or net) value is used within the „Holderbank" Group (for conversion see chapter „Combustion Engineering").

Heat balance referring to high (or gross) heat value are sometimes encountered. But here an important note must be made: the use of high heat values is based on a quite different method of considering heat losses. Therefore those balances cannot be compared directly to our type of balance.

4.3.2 Coal Firing

For coal firing it is essential that the CV and the consumption (m) refer both to the same conditions. The normal convention is to refer to the state as fed to the kiln burner.

The state as fed to burner may sometimes be different from the state as analyzed. Differences can occur because of dust addition in coal mill, loss of volatile matter or just simply by changed moisture conditions.

If the actual CV at the given coal moisture w is not known it can be calculated from the value CV_o at the moisture w_o:

$$CV = [(1-w) / (1-w_0)] \cdot (CV_0 + w_0 \cdot r) - w \cdot r$$

where: r = 2450 kJ/kg

= heat of water evaporation at reference temperature 20°C

w, w_o = weight fractions of water

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Examples:

a) Coal, CV at 5% moisture = 26'500 kJ/kg
CV at 1% moisture (as fed to burner) = ?

$CV = [(1-0.01) / (1-0.05)] \cdot (26500 + 0.05 \cdot 2450) - 0.01 \cdot 2.450$

CV = 27'464 kJ/kg

If above coal (1% moisture) is fired at a specific rate of 0.1230 kg/kg cli:

$h = 0.1230 \cdot 27'464 =$ 3'378 kJ/kg cli

b) If the CV for dry matter is known (CV_{dry}) the general formula can be simplified (for $w_o = 0$):

$CV = (1 - w)\ CV_{dry} - w \cdot r$

CV_{dry} = 28 MJ/kg, moisture 2%

$CV = 28 \cdot (1 - 0.02) - 0.02 \cdot 2.45 =$ 27.39 MJ/kg

4.4 Burnable Components in Raw Material

4.4.1 Organic Matter

Organic matter can be present in the raw meal e.g. in form of oil shale or exceptionally even in form of free crude oil.

Such material is partly volatilized in the preheating zone and leaves the system partly as unburnt light hydrocarbons. If the latter are actually measured and considered in the total heat balance it is recommended to determine the calorific value of the raw meal (not directly, but by extracting its organic matter). Thus the heat input becomes:

$h = CV_{low} \cdot R$ (kJ/kg cli)

where:

CV_{low} = calorific value due to organic matter, referred to 1 kg raw meal

R = kiln feed / clinker - factor

If light hydrocarbons in the exhaust gas are not measured nor considered in the total heat balance anyway the following approach will produce better results:

Determine the organic content only (org. C, measured by burning in pure O_2 to CO_2). Then consider exclusively the organic C (the rest can be neglected). The approximate heat input becomes:

$h = C \cdot R \cdot 33'000$ kJ/kg (kJ/kg cli)

where

C = organic carbon content in raw meal

Above mentioned is only an approximation but often used due to its simplicity.

Example:

R = 1.6 kg/kg cli, C = 0.2% carbon

q = $0.002 \cdot 1.6 \cdot 33'000 =$ 106 kJ/kg cli

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4.4.2 Inorganic Matter

In certain cases residues of non-oxidized pyrite (FeS_2) can occur in the raw meal. The heat input becomes:

$h = S \cdot R \cdot 12'930$ kJ/kg (kJ/kg cli)

where

S = weight fraction of sulfur (expressed as S!) from pyrite in raw meal

Example:

R = 1.6 kg/kg cli, 0.05% S (from pyrite) in raw meal

h = $0.0005 \cdot 1.6 \cdot 12'930$ = 10 kJ/kg cli

The practical heat effect in the above case is only marginal.

4.5 Loss due to Incomplete Combustion

If unburnt gases such as CO, H_2, CH_4 occur in the exhaust gas an additional heat output occurs.

The loss can be calculated to:

$h = m \cdot (CO \cdot 12'640 + H_2 \cdot 10'800 + CH_4 \cdot 35'840)$ (kJ/kg cli)

m = specific gas quantity (Nm^3/kg cli)

CO, H_2, CH_4 = volume fractions in exhaust (referred to wet gas)

In most cases only CO is measured and then the calculation becomes:

$h = m \cdot CO \cdot 12'640$ (kJ/kg cli)

Example

m = 1.50 Nm^3/kg cli (SP kiln)

CO = 0.2% (ref to wet)

q = $1.50 \cdot 0.002 \cdot 12'640$ = 38 kJ/kg cli

4.6 Heat of Formation

The heat of formation takes into account all main reactions which occur when raw meal is transformed to clinker, as follows:

RAW MEAL

- Heat of dehydration of clay (endothermic)
- Heat of decarbonation of $CaCO_3$ + $MgCO_3$ (endothermic)
- Heat of formation of clinker (exothermic!)

CLINKER

In most of the practical cases it is sufficient to assume a constant value of

h = 1750 kJ/kg cli

The value mentioned before represents a heat output since the overall reaction is endothermic (heat consuming). The main contribution to the overall value comes from the decarbonation of $CaCO_3$ (approx. 2100 kJ/kg cli).

The heat of formation may naturally have some variations from one raw meal to the other. But due to the narrow range which is specified for the cement clinker composition no major deviations (of say more than +/- 50 kJ/kg cli) have to be expected in normal cases.

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If the CaO in clinker does not originate from $CaCO_3$, but from raw material sources containing less CO_2 than the CaO balance (such as coal ash or partly decarbonized raw mix) the heat of formation would theoretically change. However, this problem can be easily avoided by the following definition:

- All mass flows (as kiln feed, dust losses, coal ash) which carry a certain amount of CaO not coming from $CaCO_3$ are considered as „heat flows" according to their „non-carbonatic" CaO content (see chapter 4.7).

By following the above definition it is not necessary to make alterations at the heat of formation. An „abnormal" situation can simply be treated by creating additional inputs or outputs, but not by changing the „standard" heat of formation.

The procedure is based on the idea of an „ideal" clinker burning from pure carbonates. If those ideal conditions do not apply we set corrections in the corresponding heat inputs or outputs. Such principles are well known from calculation of bypass CaO-losses.

Additional information on the heat of formation is given in chapter 6.2.

4.7 Heat due to Partly Decarbonized Material

Decomposition of carbonates in an essential heat effect in a cement kiln. Usually one thinks of the two extreme cases of either complete presence of carbonates ($CaCO_3$, $MgCO_3$) or complete absence of carbonates (e.g. clinker). But there can also be cases in between which influence the heat balance.

Material streams can transport non-sensible heat due to partly decarbonized material (CaO) or non-carbonatic CaO. The following possibilities exist:

1) raw meal (containing e.g. CaO free or non carbonatic, CaO-bearing minerals)
2) exhaust gas dust (partly decarbonized)
3) bypass dust (largely decarbonized)
4) coal ash

The heat can either be negative or positive according to the direction of flow (ingoing or outgoing).

If item 2) (exhaust gas) is considered in the balance it is essential to consider also item 1) (raw meal)! In many cases kiln dust (which contains some free CaO) is returned to the kiln but not necessarily at exactly the same rate as the outcoming dust (e.g. on a system combined with raw mill, alternating between combined and direct operation). Even if dust is returned the net heat effect of item 1) minus item 2) may not automatically be zero!

If the CaO and CO_2 is known from chemical analysis the non carbonatic part CaO_{nc} can be calculated:

$$CaO_{nc} = CaO - (56 / 44)\ CO_2$$

Strictly speaking, above formula is only correct if no other carbonates than $CaCO_3$ occur. If $MgCO_3$ is present in form of carbonates the „carbonatic MgO" has to be taken into account:

$$CaO_{nc} = CaO - 56\ [(CO_2/44) - (MgO/40)]$$

Thus the heat is calculated to

$$h = m \cdot CaO_{nc} \cdot 3150 \qquad \text{(kJ/kg cli)}$$

m = spec. mass (kg/kg cli)

CaO_{nc} = „non-carbonatic" CaO, expresses as weight fraction

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Above formula does not consider the small possible influence of $MgCO_3$ decarbonation heat. If the assumption can be made that all MgO occurs in non-carbonatic form (e.g. after a heat treatment in the 700°C range) the following improved formula can be applied:

$h = m\ (CaO_{nc} \cdot 3150 + MgO \cdot 2710)$ (kJ/kg cli)

Examples

1) Raw Meal and Kiln Dust

(dust returned, measurements refer to balance limits, normal raw mix)

Raw Meal		**Kiln Dust**	
R	= 1.65 kg/kg cli	m	= 0.09 kg/kg cli
CaO	= 42.3%	CaO	= 43.5%
MgO	= 0.9%	MgO	= 1.1%
CO_2	= 34.0%	CO_2	= 30.5%
CaO_{nc}	= 42.3-56 [(34/44) - (0.9/40) = 0.3%	CaO_{nc}	= 43.5-56 [(30.5/44) - (1.1/40) = 6.2%
h	= 1.65 · 0.003 · 3150 = **16 kJ/kg cli (input)**	h	= 0.09 · 0.062 · 3150 = **18 kJ/kg cli (output)**

In the above example the net heat effect is virtually zero and may be completely neglected. But this may not be used as a general rule as shown by the next example.

2) Raw Meal and Kiln Dust

(similar to example 1), but raw mix containing a major proportion of non-carbonatic CaO)

Raw Meal		**Kiln Dust**	
R	= 1.60 kg/kg cli	m	= 0.09 kg/kg cli
CaO	= 43.2%	CaO	= 43.5%
MgO	= 0.9%	MgO	= 1.1%
CO_2	= 30.0%	CO_2	= 30.5%
CaO_{nc}	= 43.2-56 [(30/44) - (0.9/40) = 6.28%	CaO_{nc}	= 43.5-56 [(30.5/44) - (1.1/40) = 6.2%
h	= 1.60 · 0.0628 · 3150 = **317 kJ/kg cli (input)**	h	= 0.09 · 0.062 · 3150 = **18 kJ/kg cli (output)**

Although non-carbonatic CaO occurs seldom, it may largely contribute to the heat balance (in the latter example a net effect of 299 kJ/kg cli!). The practical problem, however, is to determine the CaO_{nc} with sufficient precision.

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3) Bypass Dust

m	=	0.04 kg/kg cli (quantity of bypass dust)		
CaO	=	56.2%		
MgO	=	1.0%		
CO_2	=	1.8%		
CaO_{nc}	=	56.2 - (56/44) * 1.8	=	53.9%
q	=	0.04 * 0.539 * 3150 + 0.01 * 2710	=	95 kJ/kg cli (output)

4) Coal Ash

m	=	0.02 kg/kg cli (quantity of ash)		
CaO	=	21%		
MgO	=	2%		
q	=	0.02 (0.21 * 3150 + 0.02 * 2710)	=	14 kJ/kg cli (input)

4.8 Heat of Evaporation

If water is evaporated within the balance limits the heat of evaporation becomes

$h = m \cdot 2450$ (kJ/kg cli)

m = evaporated water (kg/kg cli)

Above items means an output. A large source of water evaporation is usually the slurry feed to a wet kiln.

Although the calculation of heat of evaporation is obviously simple a few notes are given:

- Only the free water of the kiln feed is considered (the hydrate water is already included in the heat of formation!).
- Water evaporation can also be caused by water injection into preheater, kiln, cooler, etc.
- The water evaporated from the fuel must not be included (this effect is already included in the net heat value, provided it has been determined as described in paragraph 4.3).

Examples

a) Wet Kiln: slurry water content = 35%

R	=	1.56 kg/kg cli (dry)		
m	=	0.35 / (1-0.35) · 1.56	=	0.84 kg/kg cli
h	=	0.84 · 2450	=	2058 kJ/kg cli

b) Water Spray into Planetary Cooler

m	=	0.05 kg/kg cli
h	=	0.05 * 2450 = 123 kJ/kg cli

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4.9 Sensible Heat

4.9.1 General

Generally the sensible heats are calculated as follows:

$h = m \cdot c_p \cdot (t - 20°C)$ (kJ/kg cli)

m = specific mass (kg/kg cli or Nm^3/kg cli)

c_p = average specific heat (kJ/kg C or kJ/Nm^3 C)

t = temperature of m (C)

Above formula uses a reference temperature of 20°C, i.e. sensible heats of material and gas flows at 20°C are zero.

The value h (kJ/kg cli) can either be positive (if t > 20°C) or negative (if t < 20°C).

Whether h represents an input or an output (see 4.1) depends only on the direction of the flow „m":

- if m = entering boundary → h = input
- if m = leaving boundary → h = output

Basically the value m can be expressed as kg or Nm^3. It is then logical that the c_p values used must also refer to same unit (per kg or per Nm^3).

For convenience we use the following convention:

- for solid flows → unit = kg
- for gaseous flows → unit = Nm^3 [1)]

[1)] The application of the unit „kg" for gases would also be thinkable and would even facilitate the mass balance calculations. But since Nm^3 is well established in the cement industry and since the cp-diagrams refer to Nm^3 only, Nm^3 is used in this chapter.

The necessary c_p values are to be found in the diagrams Fig. 5 and 6 (for solids and fuels) and Fig. 7 (for gases).

The same basic mode of calculation of sensible heat can be adapted for all material and gas flows crossing the boundary such as:

- Raw meal
- Clinker
- Dust(s)
- Exhaust gas, bypass gas, waste gas, etc.
- Cooling air, false air, etc.

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Fig. 5 Cp of Solids

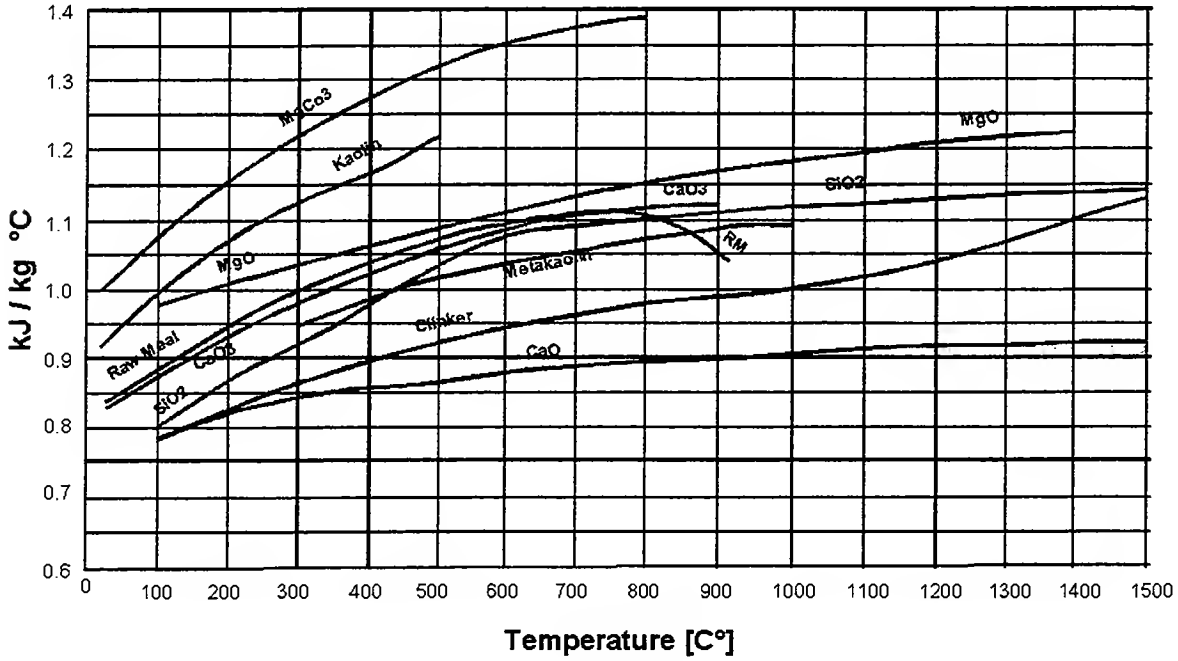

Fig. 6 **c_p of Liquids and Fuels**

Temp.	c_p of Liquid Water
0°C	4.22 kJ/kg C
50°C	4.18 kJ/kg C
100°C	4.22 kJ/kg C

Cp of Fuels

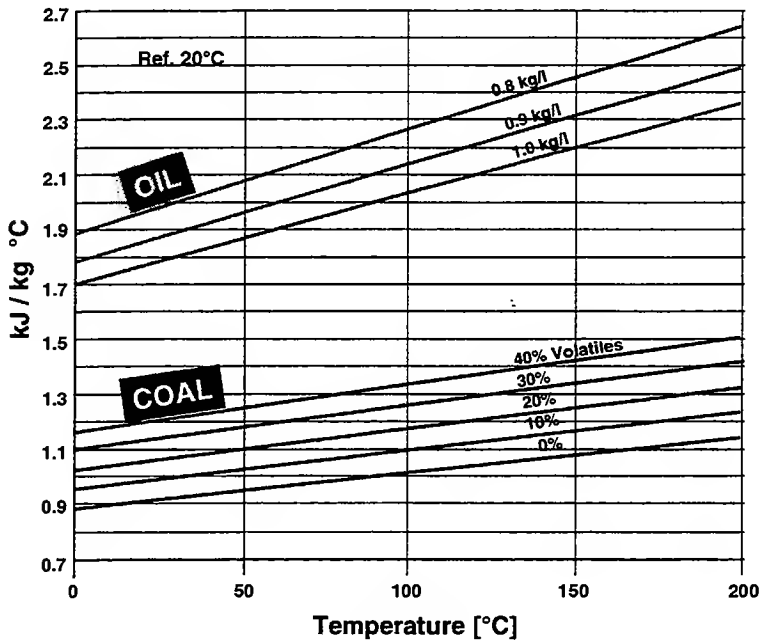

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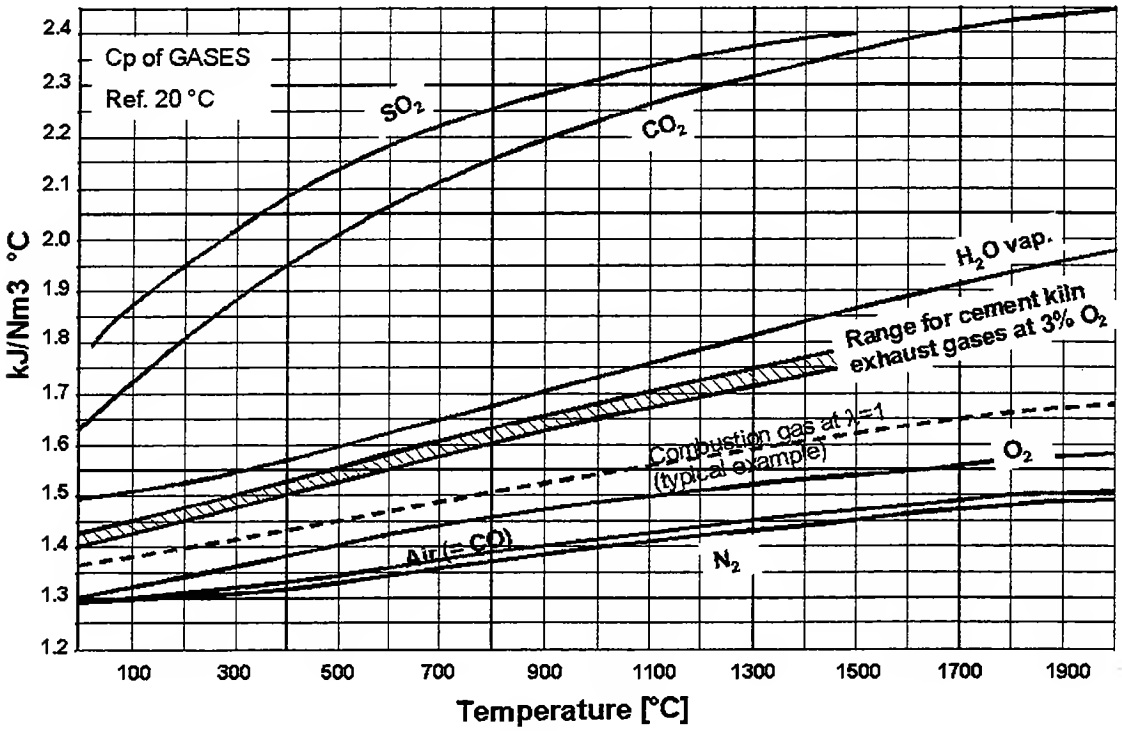
Fig. 7 c_p of Gases



4.9.2 Calculation of c_p of Mixtures

The c_p values are usually tabulated for pure substances. If a mixture is present it may become necessary to calculate the c_p starting from the given composition. This is better shown in a practical example (instead of a general formula):

Example:

Exhaust gas of suspension preheater kiln at 360 °C

- Measured by Orsat (dry):
  - CO_2 = 27.2% (CO = O)
  - O_2 = 4.3%
  - N_2 = 100 - 27.2 - 4.3 = 68.5%
- From moisture determination:
  - H_2O = 0.08 Nm^3/Nm^3 wet

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What is the c_p value of the mixture (wet gas)?

	Dry Gas (Vol %)	Wet Gas (Vol %)	c_p (pure (kJ/Nm^{3}C)	c_p x Vol. Frac. (kJ/Nm^{3}C)
CO_2	27.2	25.0	1.92	0.480
O_2	4.3	4.0	1.37	0.055
N_2	68.5	63.0	1.31	0.825
H_2O	0	8.0	1.55	0.124
Total	100%	100%	-	1.484 ≅ 1.48

Result: c_p of mixture = 1.48 kJ/Nm^{3}C (at 360°C)

The above value is typical for a normal SP kiln. We will always find values in the 1.5 kJ/Nm^{3}C range.

4.10 Heat Loss due to Radiation and Convection

4.10.1 General

In practice it is quite convenient to treat both, radiation and convection heat transfer together. Although the physical laws of these two heat transfer phenomenas are different they are usually given as a total.

The determination of total heat transfer coefficients by using simplified working diagrams will be accurate enough. Therefore the theory how to calculate the values is treated in separate chapters (6.3 and 6.4). The user of this chapter does usually not need to go into details of radiation and convection heat transfer theories. In addition, the calculation from the complete theory would hardly increase the practical accuracy, since all these theories contain a certain inaccuracy.

4.10.2 Radiation Heat Transfer

Radiation heat transfer depends on the surface temperature and the emissivity ε (0...1).

Typical values for ε:

Type of Surface	Temperature °C	ε
Rough oxidized steel	100	1
Rough oxidized steel	400	0.9
White lime paint (on steel)	400	0.8
Aluminium paint	100	0.2 to 0.4
Aluminium, rolled sheet metal	100 to 500	0.08
Aluminium, polished or bright foil	100 to 500	0.04

In most cases an ε value has to be set at the pyrometer used for the measurement (depending on the type).

For pyrometers giving readings for a constant emissivity ε = 1. Fig. 8 can be used for correction.

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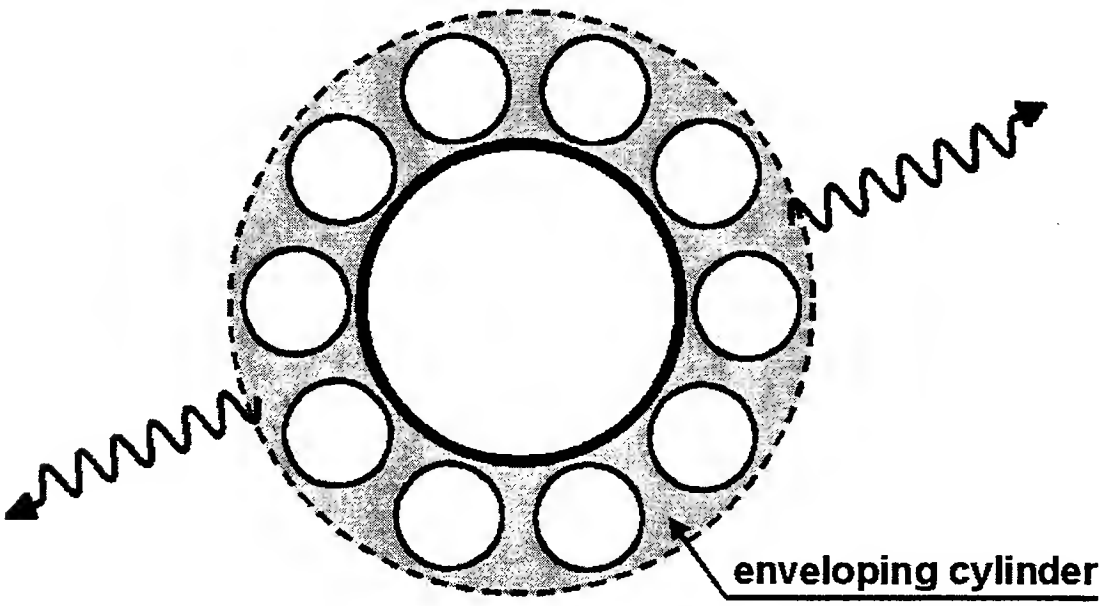
The pure radiation heat transfer coefficient α_{rad} (W/m^2C) is shown in Fig. 9.

Note that the temperature axis in Fig. 9 means temperature difference to ambient (same convention as for the following Fig. 10 and 11 - see paragraph 4.10.3 and 4.10.4).

Radiation heat transfer calculation does usually not create special problems as long as the radiating surfaces face freely towards ambient. A particular situation occurs on a planetary cooler where certain directions of radiation are shielded. That means not all areas are able for radiation towards the ambient.



Here, it would be obviously wrong to consider the total of the tube surfaces as radiation area. As a first approach the outer enveloping cylinder area (enveloping all tubes, see sketch) can be taken as reference area.

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Fig. 8 **Relation between Emissivity ε and True and Apparent Surface Temperature**

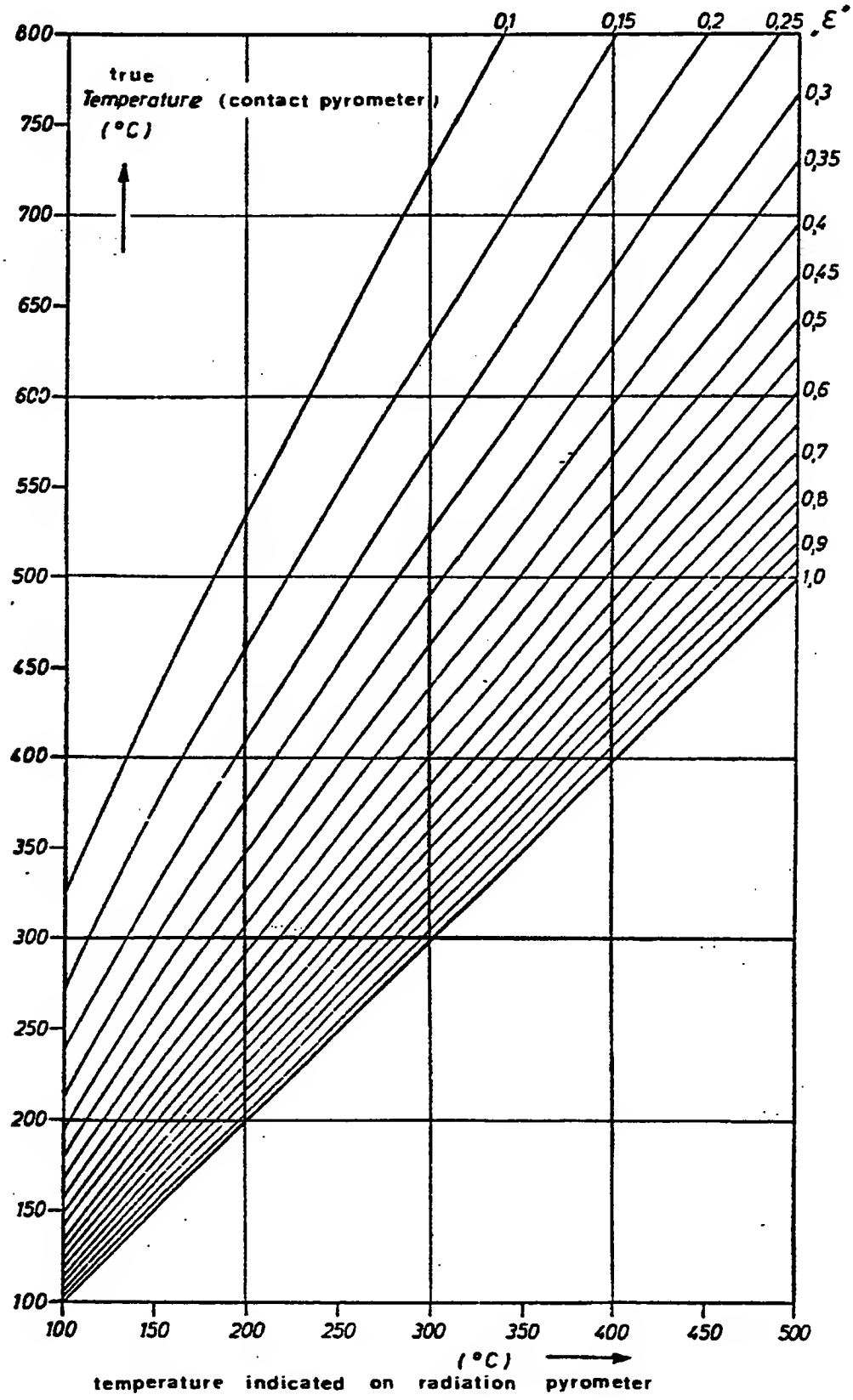

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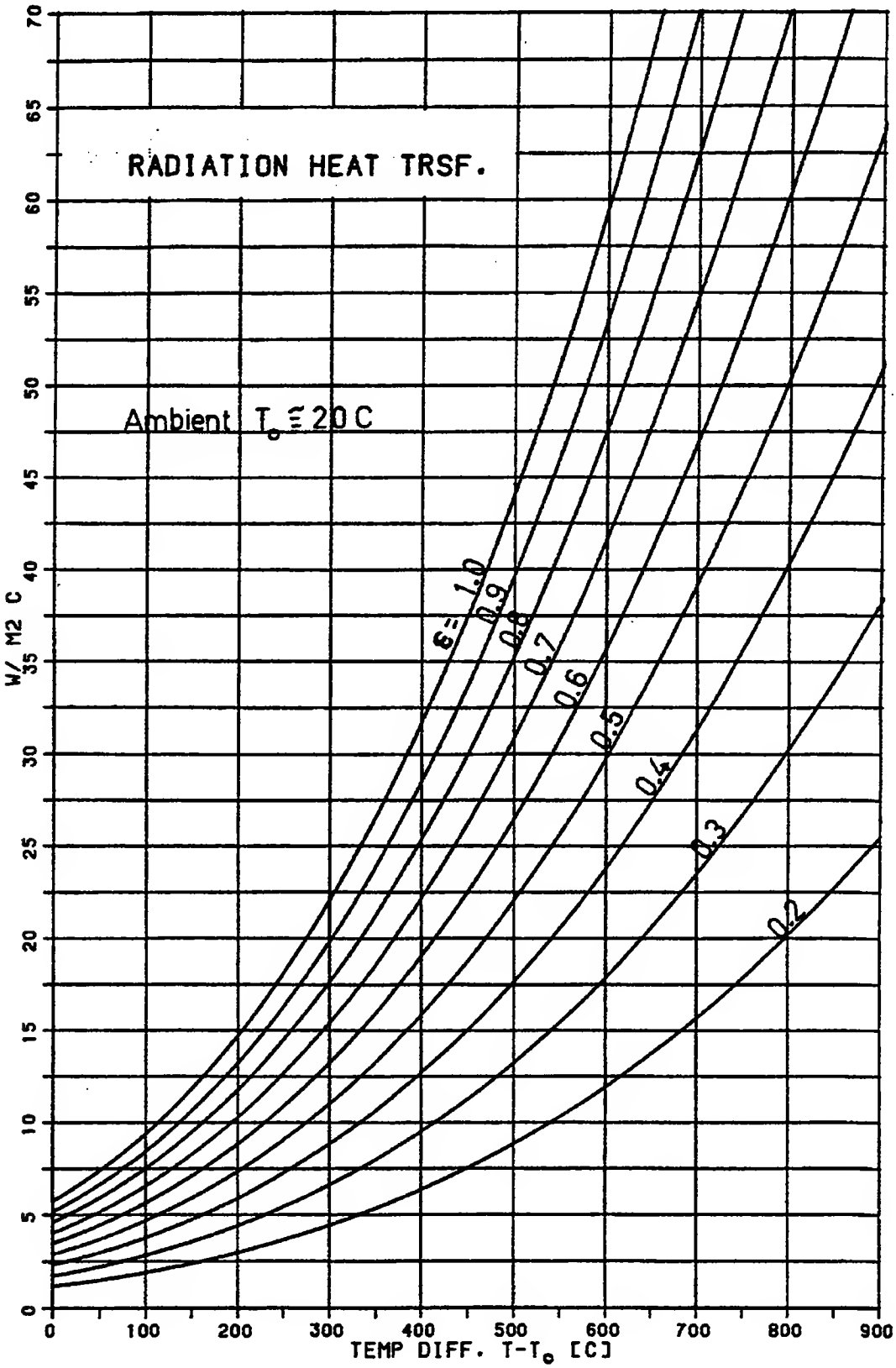
Fig. 9 Radiation Heat Transfer Coefficient



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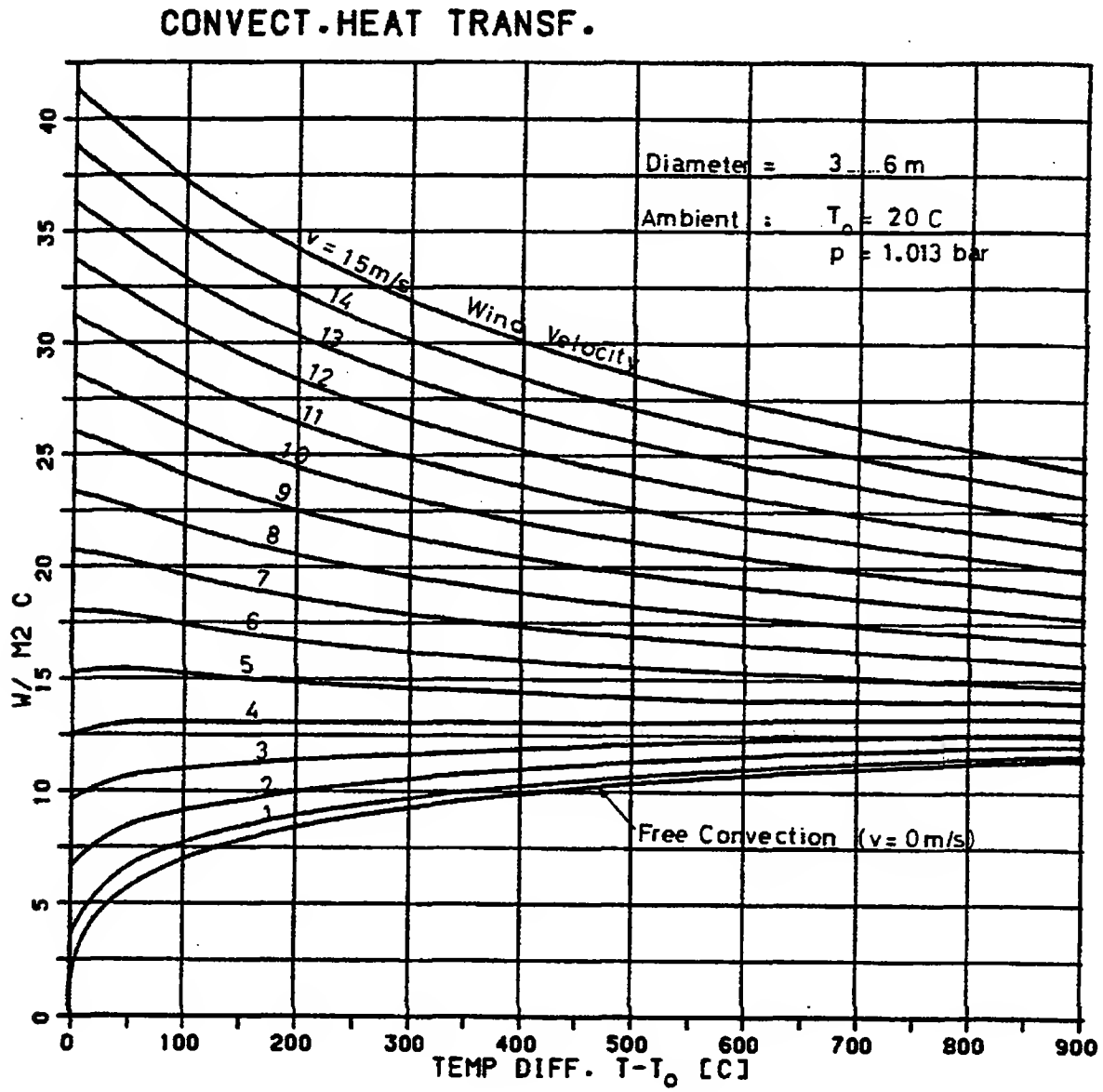
4.10.3 Convection Heat Transfer

Two different types of convection can be distinguished:

- Free convection (occurs by natural thermal draft, at low wind velocities)
- Forced convection (occurs at high wind velocities)

In practice an intermediate region between these two extreme cases can be encountered, depending on the wind velocity v (m/s). The resulting α_{conv} can be taken from Fig. 10.

Fig. 10 Convection Heat Transfer Coefficient



The diagram Fig. 10 refers to a normal kiln diameter range from 3 m to 6 m.

Note that the temperature axis in Fig. 10 means temperature difference to ambient, not temperature itself.

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- Influence of diameter
  - Free convection (v = 0 m/s):
    - The free convection coefficient is not depending on the diameter (or height) of equipment. If there is no wind Fig. 10 can be applied for any dimension.
  - Forced convection:
    - In contrast to the free convection the forced convection does depend on the diameter. For this reason the application range of Fig. 10 is limited to 3 m...6 m, but only if high wind velocities occur. At low wind velocities no diameter limits exist.
    - For very small tubes (or gas ducts) the following corrections of α_{conv} (increases) can be made:

      For ∅ 2 m + 15%
      For ∅ 1 m + 30% } only at high wind velocities (min. 2 m/s)
      For ∅ 0.5 m + 50%

4.10.4 Total Heat Transfer Coefficient (Radiation and Convection)

For the determination of the total heat transfer coefficient α_{tot} (W/m^2C) two cases are possible:

a) Standard Case

ε = 0.9 (rough oxidized steel surface)

This particular case is rather simple. Direct use of Fig. 11 is possible.

(Go into diagram with temperature difference and read out the total heat transfer coefficient.)

b) Non-standard Case

ε differs clearly from 0.9 and/or correction of convective heat transfer coefficient is necessary.

Make use of Fig. 9 and 10.

$\alpha_{tot} = \alpha_{rad} + \alpha_{conv}$

(Caution: never add free convection + forced convection, Fig. 10 does already include the overlapping of both effects.)

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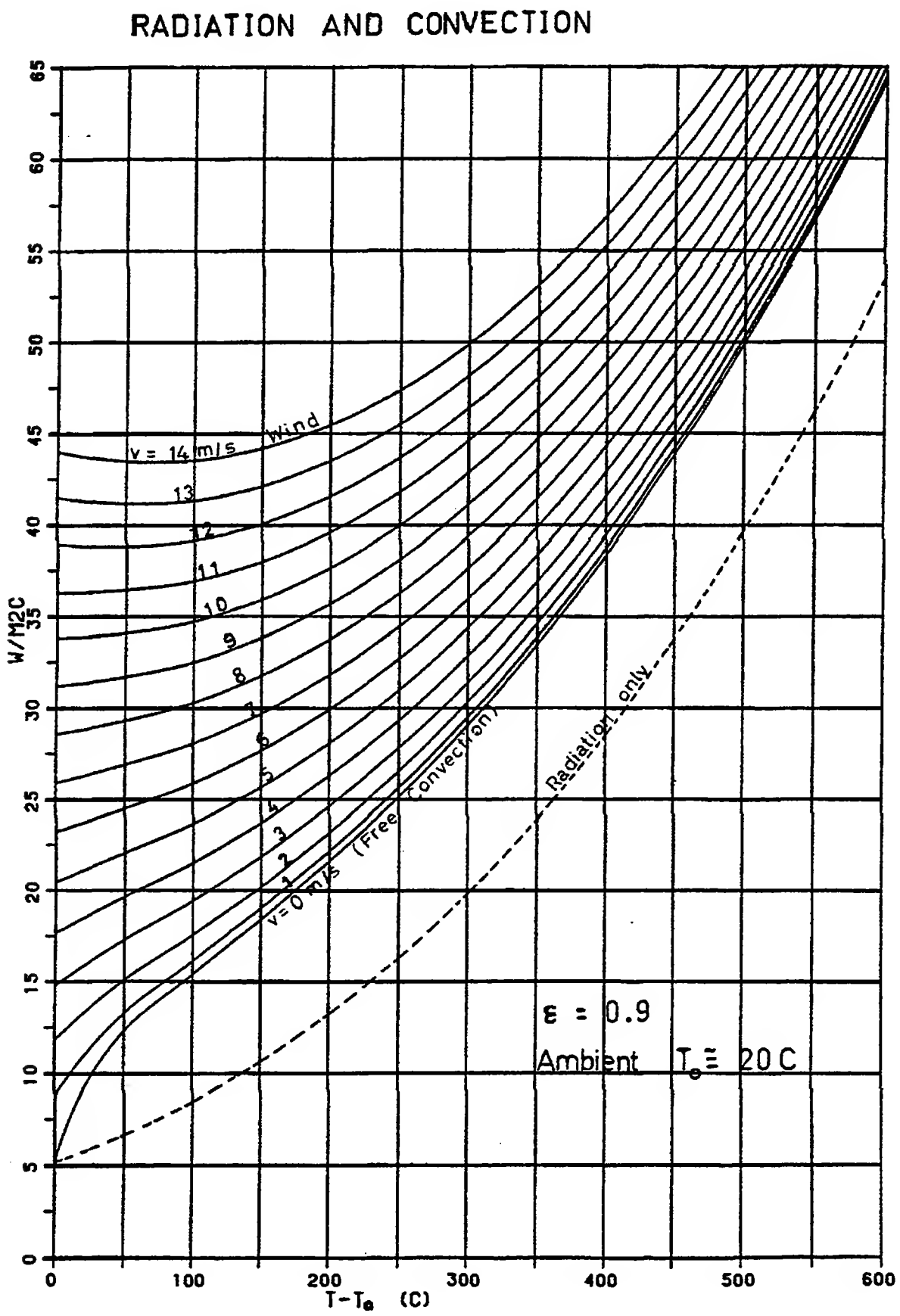
Fig. 11 Radiation and Convection Heat Transfer Coefficient (total)



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4.10.5 Total Heat Flow

The total heat flow from radiation and convection heat transfer is calculated according to:

Q_f = $\alpha_{tot} \cdot A \cdot (t - t_0)$ (W)

Q_f = heat flow (W); 1kW = 1000W

α_{tot} = $\alpha_{rad} + \alpha_{conv}$ = total heat transfer coefficient (W/m^2C)

A = shell area (m^2)

t = shell temperature (C)

t_0 = ambient temperature (C)

From the heat flow Q_f the specific loss can be calculated:

h = Q_f (kW) / m_f (t/h) · 3.6 (kJ/kg cli)

mf = clinker production (t/h)

3.6 = conversion factor, because of unit (t/h) for m_f

If the temperature of a shell area is not approximately constant it is necessary to subdivide the area into individual sections.

On a rotary kiln we may e.g. select about 10 (or more) individual cylindrical areas, each of them having the surface:

$$A_i = \Pi \cdot D_i \cdot L_i \quad (m^2)$$

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Example:

- Rotary kiln
- Clinker production = 1900 t/d = 79.17 t/h
- ∅ 4.4 m x 67 m
- Average emissivity: $\varepsilon = 0.9$
- Ambient temperature = 20°C
- Average wind velocity v = 1 m/s (nearly free convection!)
- Shell temperature profile as indicated in the following table:

Length Position (m)	Element Length (m)	Dia-meter (m)	Element Area [1] (m^2)	Surface Temp. (°C)	ε (-)	∝ total[2] (W/m^2C)	Q_f (heat flow) (W)
0 - 5	5	4.40	69.1	230	0.9	22.7	331'000
5 - 10	5	4.40	69.1	360	0.9	32.9	774'000
10 - 15	5	4.40	69.1	310	0.9	28.6	574'000
15 - 25	10	4.40	138.2	220	0.9	22.1	611'000
25 - 35	10	4.40	138.2	330	0.9	30.3	1'299'000
35 - 45	10	4.40	138.2	260	0.9	24.9	825'000
45 - 55	10	4.40	138.2	290	0.9	27.1	1'011'000
55 - 60	5	4.40	69.1	270	0.9	25.6	442'000
60 - 67	7	4.40	96.8	220	0.9	22.1	428'000
Total	**67 m**		**926.1 m^2**				**6'295'000 W**

Total heat loss = 6'295 kW

Specific heat loss = $\frac{6'295 \cdot 3.6}{79.17 t/h}$ = 286 kJ/kg cli

[1] $A = \Pi \cdot D \cdot L$ (cylinder)

[2] radiation plus convection (see Fig. 11)

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5. HEAT BALANCES

5.1 General

The results for heat inputs and outputs as calculated according to the preceding paragraph 4 are summarized in a balance sheet.

Table 1 (complete kiln system) and Table 2 (cooler only) are given as examples. They may also be used as checklist for completeness of own measurements and calculations.

- „Rest"-item:
  - A real balance (from complete date) will always include an item „rest", where all measuring errors (no measurement is 100% accurate) and non-considered items are included. The amount of the „rest" item should not exceed 3% of the total heat input.
- Heat consumption:
  - The actual heat consumption is the total of fuel input(s) which is not exactly the same as the total of heat inputs. If there are more than one fuel input, it is advantageous to indicate the actual (true) heat consumption separately.
- Cooler balance:
  - On a cooler, reliable measurement of secondary air heat is virtually impossible. Therefore, this value is determined by balance calculation and no rest item is given. The thermal efficiency of the cooler is usually defined as:

    $$\eta = \frac{\text{heat of secondary (and tertiary) air}}{\text{heat of hot clinker}} \quad (\%)$$

  - The % values in the balance sheet, however, are based on the total of inputs. This can cause a slight difference from the percentage of above η.

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Table 1 Heat Balance of Cement Kiln System

(General case)

- All referred to 1 kg clinker
- Reference temperature = 20°C
- Ambient temperature = ... °C
- Production = ... t/h
- Specific heat cons. = ... kJ/kg cli

	Specifications (kg/kg cli), (Nm3/kg cli) (kW) etc.	Temp. (°C)	Heat (kJ/kg cli)	Heat (%)
INPUT				
Fuel combustion				
- primary firing		-		
- secondary firing		-		
Burnable matter in kiln feed		-		
Raw meal: sensible heat				
Fuel: sensible heat				
Primary air: sensible heat				
Cooler air: sensible heat				
CaO (non-carbonatic) in kiln feed		-		
Total of inputs		-		100%
OUTPUT				
Heat of formation	-	-		
Water evaporation:				
- kiln feed		-		
- water spray(s)		-		
Exhaust gas:				
- sensible heat				
- dust CaO-loss		-		
- unburnt gases (CO, etc.)		-		
Cooler: - waste air sensible heat				
- middle air sensible heat				
- clinker exit sensible heat				
Bypass losses: - sensible heat				
- dust sensible heat				
- dust CaO-loss		-		
- unburnt gases (CO, etc.)		-		
Radiation and Convection:	kW			
- preheater	kW			
- rotary kiln	kW			
- cooler	kW			
- tert. Air duct	kW			
Rest				
Total of outputs		-		100%

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Table 2 Heat Balance of Clinker Grate Cooler

All referred to 1 kg clinker
Reference temperature = 20°C
Ambient temperature = ... °C

	Specifications	Temp. (°C)	Heat (kJ/kg cli)	(%)
INPUT Hot clinker Cooling air				
Total input				100%
OUTPUT Secondary air (incl. dust) Tertiary air Middle air Waste air Clinker outlet Water spray Radiation and convection				
Total output				100%

5.2 Examples of Heat Balances of Various Kiln Systems

Heat balances are given in Table 3 of various kiln systems. While comparing these it has to kept in mind, however, that kiln systems cannot only be judged based on these heat balances. The heat consumption of kiln system may depend on many items, those of major influence are:

- Kiln size, i.e. production rate
- Heat loss due to radiation and convection
- Secondary air quantity and temperature
- Operating conditions of kiln

Furthermore related systems have also to be taken into account. Consider a grate preheater kiln (Lepol kiln):

Its specific heat consumption may almost be as low as that of a 4-stage preheater kiln. However the grate preheater kiln, the semi-dry type, requires that additional heat is spent for material drying, whereas the exhaust gas of the 4-stage preheater kiln may be utilized to dry up to 8% water content of the raw material.

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The table 3 gives heat balances of three different kiln systems.

- Wet kiln:
  - The wet kiln has a production capacity of approximately 3000 t/d. The slurry water content is 38%.
- Lepol kiln:
  - The heat balance of the Lepol kiln is of a comparatively small unit. The nodule moisture content is 12%. In general grate preheater kilns of the semi-dry type have a heat consumption only slightly higher than the 4-stage preheater kilns.
- Dry suspension preheater:
-
  - The heat balance of the 4-stage preheater kiln is typical for a unit in the 2000 to 3000 t/d range.

Table 3 Heat Balances of Wet, Grate Preheater and Preheater Kilns

	WET PROCESS		SEMI-DRY (LEPOL)		DRY PRE-HEATER (4-STAGE)	
	kJ/kg cli	%	kJ/kg cli	%	kJ/kg cli	%
1. INPUT						
FUEL						
from sensible heat	25	0.4	15	0.4	13	0.4
from combustion	5560	96.7	3343	97.6	3150	97.7
RAW MEAL						
from sensible heat	25	0.4	30	0.9	54	1.7
from sensible heat of water	71	0.2	17	0.5	-	-
COMBUSTION AIR						
from sensible heat of all the air supplied (prim. sec.)	67	1.2	20	0.6	6	0.2
TOTAL INPUT	5750	100	3425	100	3223	100
2. OUTPUT						
Heat of formation	1750	30.4	1750	51.1	1750	54.3
Evaporation of water from raw meal	2370	41.2	506	14.8	13	0.4
Exhaust gas sensible heat	754	13.1	314	9.2	636	19.7
Dust sensible heat	25	0.4	21	0.6	18	0.6
Incomplete combustion (CO)	-	-	-	-	-	-
Clinker exit temperature	59	1.0	50	1.5	63	2.0
Cooler exhaust gases	100	1.7	276	8.1	423	13.1
Losses due to radiation and convection	540	9.4	452	13.2	297	9.2
Water cooling (Recupol inlet chute)	-	-	42	1.2	-	-
Rest	152	2.6	14	0.4	23	0.7
TOTAL OUTPUT	5750	100	3425	100	3223	100

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6. SPECIAL PART

6.1 Influence of Reference Temperature

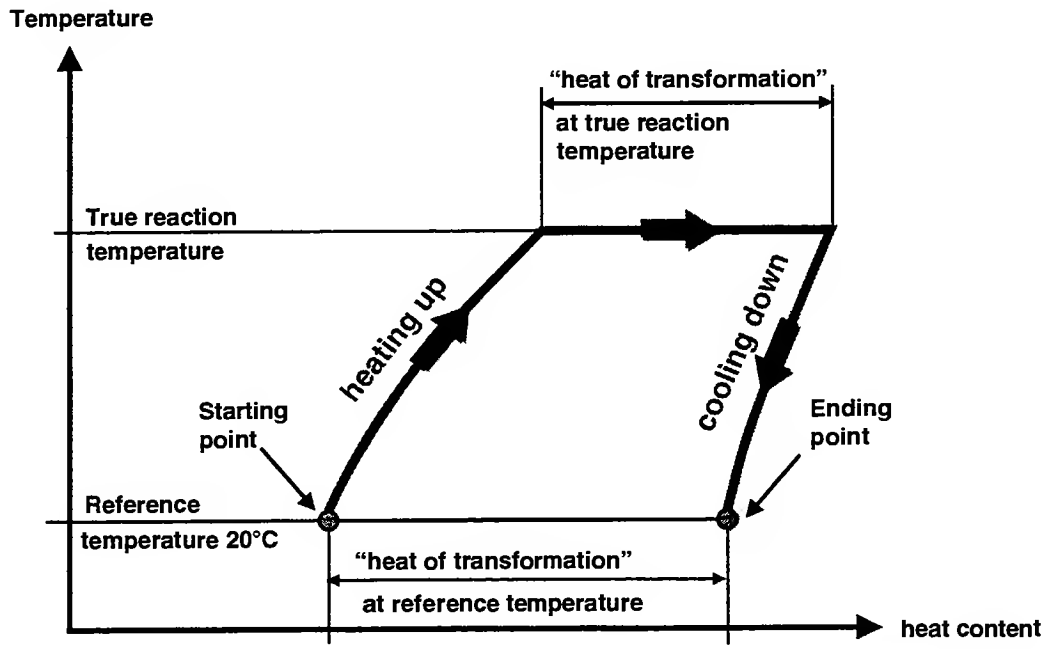
The reference temperature for a heat balance is usually set to 20°C. At this temperature all sensible heats become zero. This is quite practical because the normal ambient temperature is always near 20°C.

In addition the choice of the reference temperature has various other consequences which are normally not mentioned expressively. As long as the calculation procedure according to this chapter is applied we will not run into any practical problems. But if we want to go a bit deeper inside the matter we are soon confronted with some special questions as will be shown in the following.

6.1.1 „Heat of Transformation"

By the term „heat of transformation" we summarize here all heat effects by chemical reactions and changing of the physical state (combustion, heat of formation, evaporation). In order to understand the meaning of the reference temperature on these heat effects the following diagram is shown. It gives an example of an (ideal) isothermal process, including upheating and cooling.

Fig.12



We note from the diagram that the „heat of transformation" at reference temperature and at true reaction temperature are not identical. The reason is the different specific heat content of the starting material and the reaction products. It is also obvious that any other level for the reference temperature would basically change the „heat of transformation".

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The „heat of transformation" at a reference temperature 20°C should therefore be seen in its proper sense: It is the heat effect under the convention, that we start at 20°C and end exactly at those 20°C. The real process (heating up, transformation, cooling) has no influence on the result, regardless of the true temperatures. This is a direct consequence of the principle of energy conservation.

Therefore, we may take into account that all „heats of transformation" which we use in a heat balance are based on reference temperature 20°C. Such values should not be confused with the heat effect at the true temperature of transformation or any other temperature.

The items of interest are:

- Calorific value (combustion heat):
  - The CV must also be based on 20°C reference. The error which occurs when choosing a reference of e.g. 0°C is fortunately so small that it is within the normal accuracy of a CV determination.
- Heat of formation:
  - Regarding the heat of decarbonation only we may sometimes find values for the true reaction temperature (decarbonation, in the 800°C range). Such values shall not be used for a heat balance at 20°C reference, because this would produce an essential error!
- Heat of evaporation of water:
  - Note that we must use the evaporation heat at 20°C (2450 kJ/kg) in the balances and not the value at say 100°C (2260 kJ/kg). The effects of upheating of water or vapor are automatically considered by the normal balance calculation procedure (items for sensible heats).

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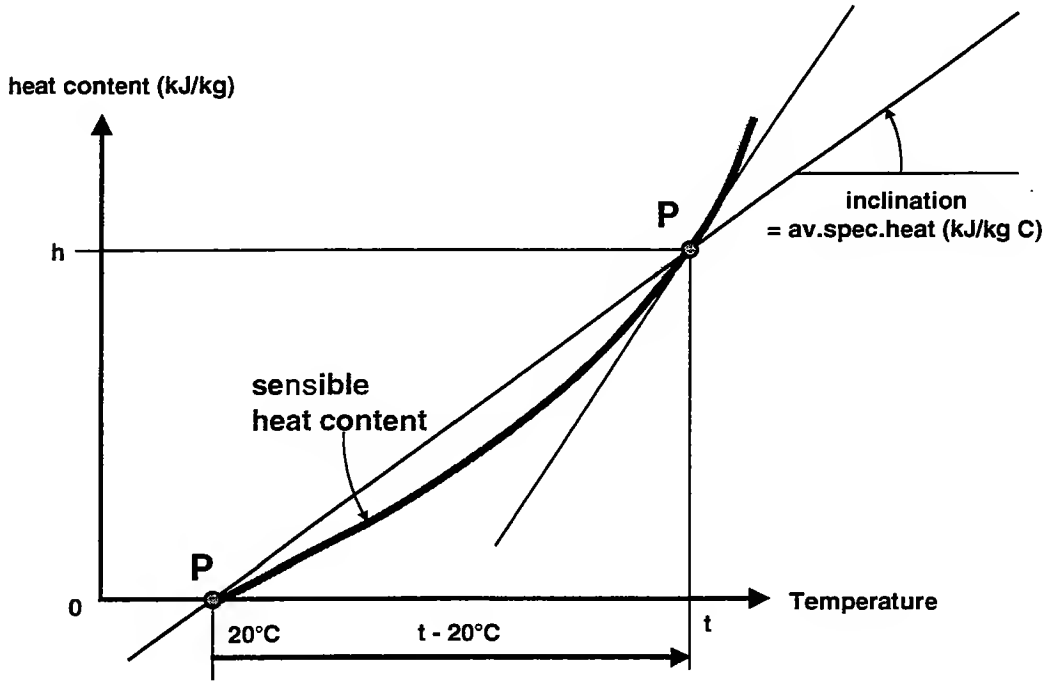
6.1.2 Specific Heats

The specific heats used in this chapter are average values which are consequently based on 20°C reference temperature. We may illustrate this fact by an upheating process starting at 20°C as shown in the following graph:

Fig.13



The average specific heat can be graphically explained as the inclination of the straight line from P_o to P. It is obvious that this inclination depends from the reference point P_o at 20°C.

At the point P the tangent to the heat curve is also shown by a dotted line. The inclination of this line is the actual specific heat at the temperature t. This value is different from the average specific heat. Moreover, it does not depend from any reference point.

As a practical consequence we should never use specific heats from any source in the literature without checking what they really mean. In most cases actual values are given and not averages. Therefore, an integration or averaging would be required before we can use them for calculation of heat contents. Special caution should be made in the high temperature range where actual value and average value may be considerably different! In the 20° to 200°C range the practical differences, however, are rather small.

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6.2 Heat of Formation

6.2.1 General

The heat of formation expresses the theoretical heat required for producing 1 kg of clinker. The following steps are taken into account:

		Approx. Temperature	Heat Effect
Step 1	Expelling of hydrate water (+ transformation of clay minerals)	~ 500°C	negative (heat consumed)
Step 2	Decarbonation of $MgCO_3$ (first) and $CaCO_3$ ($CaCO_3 \rightarrow CaO + CO_2$)	~ 850°C	negative (heat consumed)
Step 3	Formation of clinker minerals (C_2S, C_3S, C_3A, C_4AF)	~ 1400°C	positive (heat produced)

The net heat produced by the overall reaction is negative i.e. heat is consumed (listed as output).

The heat formation is defined at 20°C. In other words it expresses the theoretical amount of heat to transform raw meal at 20°C to 1 kg clinker at 20°C (if no heat losses would occur). It is therefore also considered as the theoretical minimum heat required for burning clinker.

The heat of formation is not at all depending on the way of the actual reaction (i.e. temperature versus time). The only precondition of course is that the reactions (step 1, 2, 3) have really been completed.

Although the basic principle of the heat of formation seems quite simple a few conventions (or definitions) are necessary here:

1) The hydrate water is expelled is considered as vapour (therefore no additional heat of evaporation has to be introduced into the balance!).
2) As a „standard case" the assumption is made that the CaO is present as $CaCO_3$ in the raw meal (in non carbonatic CaO occurs we do not change the heat of formation, but add heat inputs according to paragraph 4.7).
3) Burnable components in the raw mix are considered separately as inputs (see paragraph 4.4.2).

6.2.2 Calculation of Heat of Formation

6.2.2.1 Introduction

It is difficult to determine the heat of formation directly be calorimetric measurements. Therefore calculation methods have been developed in order to take into account the varying influences of raw mix properties.

Although formulas are given the following they should be applied with care. As long as the raw mix does not clearly deviate from the „standard" value of ~ 1750 kJ/kg the calculated figures must not necessarily give more accurate results than the 1750 kJ/kg.

Even extensive formulas for heat of formation contain a certain incertitude due to the partial lack of accurate thermodynamic standard values (for the minerals which are involved).

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6.2.2.2 Calculation Formulas

The following formulas are partly based on the work of H. zur Strassen (Lit. 1) however a few recent adaptations have been made:

We will use the following notation:

CaO, MgO, SiO_2, Fe_2O_3, Al_2O_3 = Clinker analysis, expressed as weight fractions (kg/kg cli)

H_2O (kg/kg cli)= Hydrate water in raw material, referred to kg clinker (not to raw meal)

The following general formula for the heat of formation applies:

$$h = CaO \cdot 3200 + MgO \cdot 2710 - SiO_2 \cdot 2140 - Fe_2O_3 \cdot 250 + h_{Res} \quad \text{(kJ/kg cli)}$$

The last contribution „h_{Res}" depends on the Al_2O_3 and H_2O (hydrate) content. The calculation of „h_{Res}" depends on the information which is available on hydrate water and/or type of clay. Usually, the more information is known the better the result will be.

Case No.	Hydrate Water (H_2O)	Type of Clay	Formula for h_{Res} (kJ/kg cli)
1	not known	not known	$h_{Res} = Al_2O_3 \cdot 1720$
2	known	not known	$h_{Res} = Al_2O_3 \cdot 120 + H_2O \cdot 5520$
3	not known	known	$h_{Res} = (Al_2O_3)_K \cdot 2220 + (Al_2O_3)_M \cdot 1310 + (Al_2O_3)_I \cdot 1640$
4	known	known	$h_{Res} = (Al_2O_3)_K \cdot 1400 + (Al_2O_3)_M \cdot 620 + (Al_2O_3)_I \cdot 760 + H_2O \cdot 2450$

In case 3 and 4 a distinction has to be made from which type of clay the Al_2O_3 originates. The indices $_{K, M, I}$ denote:

K = Kaolinite

M = Montmorillonite

I = Illite

Note on the formulas for h_{Res}:

No. 1:
For general purposes, if no information on hydrate water available.

No. 2:
Takes into account the actual hydrate water content. It is also a very good approach if the hydrate water does not only originate from clays but for example from $Ca(OH)_2$.
Note that the coefficient for H_2O (hydrate) is more than twice the normal heat of evaporation [1)]

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Nos. 3 + 4:
Take into account the type of clay. (Hydrate water must originate from clay only, otherwise take No. 2).

When the CaO is introduced into the formula it is not required to know whether the CaO results from $CaCO_3$ or not. If any non carbonatic CaO is entering into the system we will take into account a balance heat input by definition (see paragraph 4.7) and therefore the calculated heat of formation is not affected.

[1] The older formula from H. zur Strassen did allocate only 2450 kJ to H_2O, but instead 930 kJ to the Al_2O_3, which made it difficult to extrapolate it for other hydrates than from clay. It is certainly more logical to refer the heat to the H_2O, which is actually responsible for the heat consumption.

Examples

1) - Clinker composition:

CaO	=	64.8 %
MgO	=	1.2 %
SiO_2	=	22.6 %
Fe_2O_3	=	3.3 %
Al_2O_3	=	5.2 %

SO_3	=	0.52%	TiO_2	=	0.29%
Na_2O	=	0.08%	Mn_2O_3	=	0.03%
K_2O	=	1.20%	L.o.I	=	0.54%

- No information on hydrate water in raw meal available

h = $0.648 \cdot 3200 + 0.012 \cdot 2710 - 0.226 \cdot 2140 - 0.033 \cdot 250 + 0.052 \cdot 1720$ = 1704 kJ/kg cli

2) -Clinker composition:

CaO	=	65.2 %
MgO	=	1.2 %
SiO_2	=	22.9 %
Fe_2O_3	=	3.0 %
Al_2O_3	=	5.0 %

- Raw material = 1.1% hydrate water
  R = 1.57 kg/kg → 0.017 kg hydrate water/kg clinker

h = $0.652 \cdot 3200 + 0.012 \cdot 2710 - 0.229 \cdot 2140 - 0.03 \cdot 250 + 0.050 \cdot 120 + 0.017 \cdot 5520$ = 1721 kJ/kg cli

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6.2.2.3 Sulfatization Effects

The SO_2 can react with the alkalis K_2O or Na_2O (but also with CaO). For the heat effect the following (simplified) reaction is considered as typical:

$Na_2O + SO_2 + ½ O_2 \rightarrow Na_2SO_4 +$ heat

(molar heat effects of K_2O and Na_2O are of similar size!)

The overall degree of sulfatization reactions is usually not known. For the heat effect the following lower and upper boundaries can be calculated:

lower boundary = $- 8370 \cdot [(SO_3)_{cli} + (SO_3)_D - (SO_3)_R]$ (kJ/kg cli)

upper boundary = $- 10800 \cdot [(Na_2O)_R - (Na_2O)_{cli}]$
$- 7120 \cdot [(K_2O)_R - (K_2O)_{cli}]$
$+ 8370 \cdot [(SO_3)_R - (SO_3)_{cli}]$ (kJ/kg cli)

where SO_3, Na_2O, K_2O are expressed as kg/kg cli (referred to clinker!).

The indices denote: cli = in clinker
D = in dust
R = in raw meal

An average value can be introduced into the heat balance. Preferably this item is just listed under the item heat of formation (under output). Its sign is then negative, i.e. the outputs are reduced.

6.2.2.4 Lime Kilns

The formula for heat of formation of burnt lime is quite similar to that of cement clinker.

The following main reactions are taken into account:

$CaCO_3 \rightarrow CaO + CO_2$
$MgCO_3 \rightarrow MgO + CO_2$
$CaO + 2\ SiO_2 \rightarrow C_2S$ (dicalciumilicate)

Unlike on cement clinker the CaO in the lime product does not exclusively consist of „non carbonatic" CaO, but some residual $CaCO_3$ is also present. Therefore the non carbonatic proportion CaO_{nc} has to be calculated first:

$CaO_{nc} = CaO_{tot} - CO_2 \cdot (56/44)$

The heat of formation becomes:

$h = CaO_{nc} \cdot 3150 + MgO \cdot 2710 - SiO_2 \cdot 2100$ (kJ/kg lime)

CaO_{nc}, MgO, SiO_2 denote the weight fractions in lime product.

If the lime is present in lump form and the SiO_2 does originate e.g. from coal ash it is unlikely that C_2S mineral is formed. In such cases only the SiO_2 which comes from the limestone feed has to be considered in the formula.

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6.3 Radiation Heat Transfer

The basic radiation formula is

$Q_f = C_R \cdot \varepsilon \cdot A \, [(T/100)^4 - (T_0/100)^4]$ [W]

Q_f = heat flow [W]
C_R = radiation constant = 5.67 W/m^2K^4
ε = emissivity (0...1)
A = radiating area (m2)
T = absolute temperature of radiating surface [K] (T = 273+t)
T_o = absolute temperature of ambient [K]

Strictly speaking above formula is only valid for cylinders of infinite length which radiate towards an ambient which is far away from the cylinder. No obstacles may shield off the radiation.

Though its simplifications this formula may be taken as a good approach for most cases.

By dividing Q_f by T - To and A, the α-value (heat transfer coefficient) can be obtained:

$$\alpha = \frac{C_R \cdot \varepsilon}{T - T_0}\left[\left(\frac{T}{100}\right)^4 - \left(\frac{T_0}{100}\right)^4\right] (W/m^2C)$$

Or alternatively the following formula, which is identical to the latter comes out:

$$\alpha = C_R \cdot \varepsilon \cdot \frac{4}{100}\left(\frac{\bar{T}}{100}\right)^3\left[1 + \frac{1}{4}\left(\frac{\Delta T}{\bar{T}}\right)^2\right] (W/m^2C)$$

$\bar{T} = \frac{T + T_0}{2}$ = average temperature (K)

$\Delta T = T - T_0$ = temperature difference (K or C)

From the second formula it becomes evident that (at small temperature differences) α increases with the third (not fourth!) power of the average temperature. In addition, the α-value does converge towards a finite value at zero temperature difference.

For practical evaluation of the α (radiation) Fig. 9 (see paragraph 4.10.2) is available. It is based on a constant ambient temperature of To = 293 K (20°).

If the true ambient temperature is actually not exactly 20°C one commits a slight error. Usually this can be neglected. In some extreme or exceptional cases the above formulas can be used for calculations.

Since the emissivity ε is often not known precisely and does also depend on the temperature it is usually not worthwhile to do small corrections on the α-values obtained from Fig. 9.

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6.4 Convective Heat Transfer

6.4.1 Free Convection

Free convection occurs due to density differences between hot air (at the shell surface) and ambient air. A natural updraft causes a convective air movement.

For vertical plates or horizontal cylinders the following relationship applies:

$Nu = 0.13\ (Pr \cdot Gr)^{1/3}$ (-)

where: $Nu = \alpha D / \lambda =$ Nusselt number (-)

$Pr = c_p \eta / \lambda =$ Prandtl number (-)

$$Gr = \frac{D^3 \cdot g \cdot \rho^2 \cdot \Delta T}{\eta^2 \cdot T_0} =$$ Grashof number (-)

α	(W/m^2K)	heat transfer coefficient
D	(m)	characteristical dimension *
λ	(W/mK)	heat conductivity
c_p	(J/kg K)	specific heat
g	(m/s^2)	gravity constant = 9.81 m/s^2
ρ	(kg/m^3)	gas density
ΔT	(K)	temperature difference (T - To)
η	(kg/ms)	dynamic viscosity
To	(K)	absolute ambient temperature

* for a cylinder with diameter d $\rightarrow$ $D = (\Pi/2) \cdot d$

Application range: $Pr \cdot Gr > 10^9$

As long as there is a minimum temperature difference of a few degrees and the diameter range is over 1 m, above limitation does not affect the calculation.

A mathematical transformation of the basic equation reveals that the free convection heat transfer does not at all depend on the characteristical dimension! It follows:

$$\alpha = 0.13 \left(\frac{c_p \cdot \lambda^2 \cdot g \cdot \delta^2}{\eta} \right)^{\frac{1}{3}} \left(\frac{\Delta T}{T_0} \right)^{\frac{1}{3}} (W / m^2 K)$$

At $\Delta T = 0$ the free convection becomes zero (which is different from the behavior of the radiation heat transfer!).

The numerical values for c_p, λ, ρ, η must be taken from tables for air at the average temperature between surface and ambient (use SI-units only).

Note that the density ρ depends also on the barometric pressure and therefore the result will depend on the altitude above sea level ($\alpha \sim p^{2/3}$). As a general guideline the convective heat transfer drops by about 8% per 1000 m of altitude.

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By using numerical approximations for the properties of air the following relationship has been developed (dimensional equation):

$\alpha \cong 1.4 \cdot (\rho_0 \cdot \rho \cdot \Delta T)^{1/3}$ (W/m^{2}K)

ρ_o (kg/m^3)= density at ambient temperature

ρ (kg/m^3)= density at average temperature

Though its simplicity the latter formula covers the temperature range from 0...500°C with an accuracy of better than 1%! This is more than enough for practical purposes.

At sea level and at 20°C ambient it follows

$$\alpha \cong 1.6 \left(\frac{\Delta T}{1 + \frac{\Delta T}{2 \cdot T_0}} \right)^{\frac{1}{3}} \quad (W/m^2K)$$

Above relationship is also an excellent numerical approximation of the curve for v = O in Fig. 10 (see paragraph 4.10.3), which is actually based on complete computer calculations out of properties for air.

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Table Properties of Air at Pressure = 1 bar

Temp.ϑ °C	ρ kg/m³	c_p kJ/kg K	β 10^3/K	λ 10^3W/mK	η 10^6kg/ms	ν 10^6 m²/s	α 10^6 m²/s	Pr 1
-180	3.8515	1.071	11.701	9.0	6.44	1.67	2.18	0.77
-160	3.1258	1.036	9.320	10.9	7.85	2.51	3.37	0.75
-140	2.6391	1.021	7.758	12.7	9.20	3.48	4.71	0.74
-120	2.2867	1.014	6.659	14.6	10.49	4.587	6.30	0.73
-100	2.0186	1.011	5.846	16.4	11.72	5.806	8.04	0.72
-80	1.8073	1.009	5.219	18.16	12.89	7.132	9.96	0.72
-60	1.6364	1.007	4.719	19.83	14.02	8.567	12.0	0.71
-40	1.4952	1.006	4.304	21.45	15.09	10.09	14.3	0.71
-20	1.3765	1.006	3.962	23.01	16.15	11.73	16.6	0.71
0	1.2754	1.006	3.671	24.54	17.10	13.41	19.1	0.70
20	1.1881	1.007	3.419	26.03	17.98	15.13	21.8	0.70
40	1.1120	1.008	3.200	27.49	18.81	16.92	24.5	0.69
60	1.0452	1.009	3.007	28.94	19.73	18.88	27.4	0.69
80	0.9859	1.010	2.836	30.38	20.73	21.02	30.5	0.69
100	0.9329	1.012	2.684	31.81	21.60	23.15	33.7	0.69
120	0.8854	1.014	2.547	33.23	22.43	25.33	37.0	0.68
140	0.8425	1.017	2.423	34.66	23.19	27.53	40.5	0.68
160	0.8036	1.020	2.311	36.07	24.01	29.88	44.0	0.68
180	0.7681	1.023	2.209	37.49	24.91	32.43	47.7	0.68
200	0.7356	1.026	2.115	38.91	25.70	34.94	51.6	0.68
250	0.6653	1.035	1.912	42.43	27.40	41.18	61.6	0.67
300	0.6072	1.046	1.745	45.91	29.20	48.09	72.3	0.67
350	0.5585	1.057	1.605	49.31	30.90	55.33	83.5	0.66
400	0.5170	1.069	1.485	52.57	32.55	62.95	95.1	0.66
450	0.4813	1.081	1.383	55.64	34.00	70.64	107	0.66
500	0.4502	1.093	1.293	58.48	35.50	78.86	119	0.66
600	0.3986	1.116	1.145	63.5	38.30	96.08	143	0.67
700	0.3577	1.137	1.027	67.8	40.87	114.3	166	0.69
800	0.3243	1.155	0.932	71.3	43.32	133.6	190	0.70
900	0.2967	1.171	0.852	74.3	45.65	153.9	214	0.72
1000	0.2734	1.185	0.786	76.8	47.88	175.1	237	0.7

↓

(actual c_p, not average)

6.4.2 Forced Convection

Forced convection occurs at comparatively high wind velocity and dominates the convective heat transfer, i.e. the free convection is suppressed.

The calculation of forced convection is depending on many factors, such as:

- Wind velocity
- Direction of the wind
- Velocity distribution and flow obstacles
- Uniformity of wind
- Reynolds number (depends on kiln diameter).

There are a few more influence factors than in case of the free convection. Generally speaking the calculation of forced convection heat transfer contains more possible sources of error than free convection.

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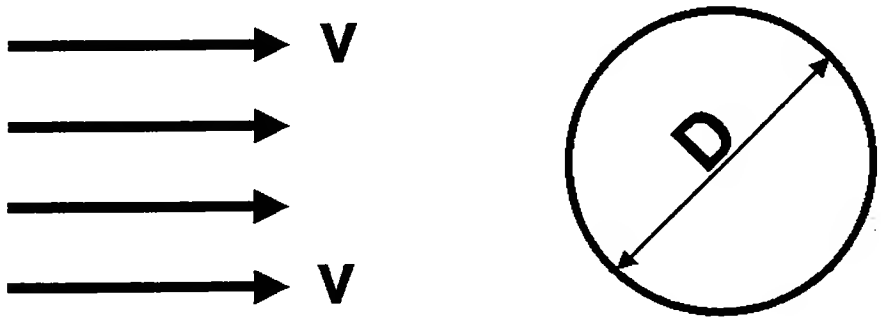
As a simplification, we will assume a cylinder in a non disturbed flow of a constant velocity v (at 90° against kiln axis).

Fig. 14



For air the following formula apply:

Nu	=	$0.0239 \cdot Re^{0.805}$	for Re =	40'000...400'000
Nu	=	$0.00672 \cdot Re^{0.905}$	for Re >	400'000

$$Re = \frac{vD}{\nu} = \frac{v \cdot D \cdot \rho}{\eta} = Reynolds Number$$

$$Nu = \frac{\alpha D}{\lambda} = Nusselt Number$$

The properties η, λ, ϑ have to be taken at average air temperature.

There are other formulas in use which can give different results, the above formulas are preferred due to their simplicity. In any case there is always a incertitude from the mode of calculation itself.

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Two main factors determine the ∝-value:

- Velocity v
- Average temperature (between surface and ambient)

In addition the ∝ does also depend on the diameter D. If the equation for high Reynolds number (Re > 400'000) is solved for ∝, the following relationship is obtained:

$$\propto \quad \sim \quad D^{-0.095}$$

$$\downarrow$$

proportional

This means that the ∝ value does not much depend on D! Therefore it is possible to work with constant values within a certain diameter range.

This actually the base of Fig. 10 (see paragraph 4.10.3) which is calculated for a common range from 3...4...6 m diameter.

6.4.3 Free Convection Plus Forced Convection

If the convection is clearly dominated either by free or forced convection the final determination of the representative ∝ does not give any problem, since the higher value has to be taken.

If the two values are of the same order they must be combined with an appropriate method. It would be certainly wrong to add the two values. A better approach is the square addition:

$$\alpha_{tot} = \sqrt{\alpha^2_{free} + \alpha^2_{forced}}$$

It is also valid if either the free convection or the forced convection dominates.

Fig. 10 is based on the above method.

6.5 Effect of Thermal Improvements

According to the actual condition of an existing kiln system (heat consumption, heat balance, other operating data) we can envisage an optimization campaign. From the thermal point of view we can take certain measures in order to reduce the specific heat consumption. For example:

- Better insulation of rotary kiln or preheater/precalciner in order to reduce radiation losses (except the sintering zone).
- Improvement of the cooler efficiency (optimization of grate cooler operation or installation of highly efficient internal equipment in planetary or rotary cooler).
- Reducing of false air inleaks at kiln seals or at preheater.
- Reducing of internal dust circulations in cooler, kiln or preheater (improves the internal counter-current heat transfer).
- Modification of raw mix in order to decrease the sintering temperature which in turn will also reduce heat losses. A further effect of such a measure can be the reduction of internal dust circulations due to better clinker granulometry (see above).
- Modification of raw mix in order to decrease the heat required for decarbonation, e.g. by making use of non-carbonatic CaO raw material sources. This possibility, however, is very rare and often not feasible. (Note the basic difference to the measures which tend to decrease the sintering temperature or increase the proportion of melting phase!)

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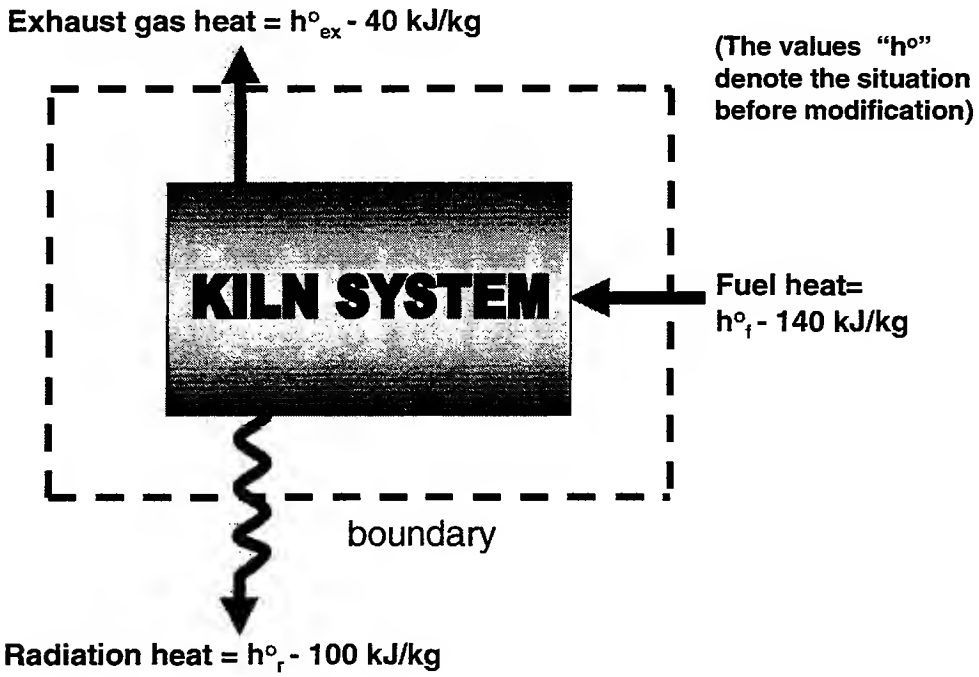
The above measures are just a few typical examples. When one goes into such items, an important phenomenon will appear soon: The so called „loss multiplication" factor for thermal losses (or savings). What does this mean?

If a saving (or loss) in the high temperature zone in a kiln of say 100 kJ occurs the possible gain in fuel heat consumption will not be 100 kJ but rather 130 to 150 kJ. That means that the primary saving (in terms of heat balance item) will be multiplied by a factor of up to 1.5.

At the first glance the above principle seems to be contradicious because it would violate the principle of heat balance or the energy law. However, what really happens is a differential change of more than only one heat balance item. To illustrate this fact we start from an example where we have reduced the shell radiation losses by 100 kJ/kg cli.

The following differential balance situation occurs:

Fig. 15



The corresponding multiplication factor for the above case is:

$$\text{multiplication factor} = \frac{-140\ \text{kJ/kg}}{-100\ \text{kJ/kg}} = 1.40$$

The above fact does result from the thermal behavior of the system and can be verified by simulation models (not by a simple balance only).

A factor in the 1.4-range is quite typical for the situation in the high temperature zone (above 800°C) of a cement kiln. The main effect of a change in this zone will be a corresponding change at the exhaust gas, but also other minor effects will occur (e.g. at cooler losses).

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A "differential balance" of heat can still be applied according to:

Input Change			Output Change		
fuel	=	- 140 kJ/kg	radiation	=	- 100 kJ/kg
			exhaust	=	- 40 kJ/kg
total	=	- 140 kJ/kg	total	=	- 140 kJ/kg

The principle of loss multiplication does not only apply for radiation in the high temperature zone but also for the heat which is recuperated in the clinker cooler.

Regarding the false air inleaks the corresponding deterioration of heat consumption is often under-estimated. If false air inleaks into the high temperature zone it does not only cause a heat loss because this air must be heated up to the exhaust gas temperature of the kiln system! It actually causes much more losses than what would result from such a simplified calculation approach. As a rule of thumb we may consider the following two main effects in order to come to a realistic result:

- Heating up of false air at the temperature of the high temperature zone which can be set approximately to 800°C (end of HT-zone).
- Multiplication of the above heat requirement by a loss multiplication factor.

The above calculation is a rough approach. By more sophisticated simulation models we find e.g. that a false air inleak into the high temperature zone of 0.08 Nm^3/kg cli can cause an additional heat consumption of 100 kJ/kg cli. In contrast, the inleak in the low temperature zone (e.g. air lift on SP-kiln) is much less critical.

6.6 Heat Transfer in Preheaters and Coolers and Improvement Potential

Normally we are considering a cement kiln as an equipment for burning cement clinker. As an essential feature we must be able to generate a high sintering temperature in the order of 1450°C (material temperature).

But a kiln system is of course much more than a generator of sintering temperature. It is also a system of heat exchangers which allows for burning at low heat consumption. Generally speaking we will find two essential heat-exchanging systems on any cement kiln:

a) raw meal preheater

b) clinker cooler

Low heat consumption is only possible if the above two „heat exchangers" perform optimally.

How can we get optimum heat exchange performance? Form the basic theory it is known that even in case of an exchanger which is perfectly insulated against ambient temperature influence, three important conditions are required for optimum heat exchange:

1) Optimum heat transfer rate (here: from gas to solid) → high specific contact or surface area, high ∝ (W/m^2C).
2) The two heat exchanging streams must flow in counter-current manner, or at least in an arrangement which has similar characteristics to a counter-current system (e.g. overall arrangement of a 4-stage cyclone preheater).

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3) If we want to recover heat from a „flow 1" completely into a „flow 2" the „flow 2" must have at least the same heat equivalence as „flow 1":

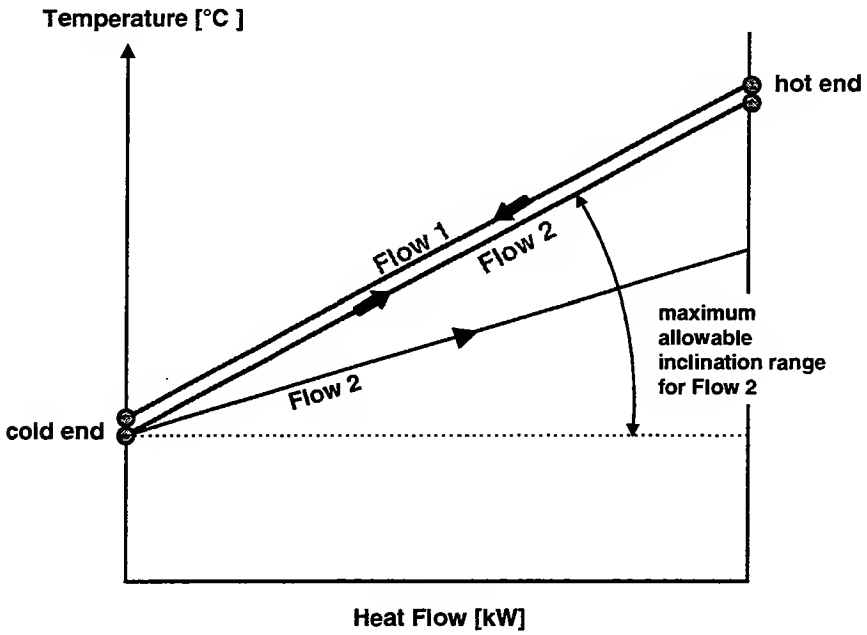
(flow 2) x (c_{p2}) ≥ (flow 1) x (c_{p1}) [kW/C] or [kJ/kg cli C]

flow: [kg/s] or [kg/kg cli]

cp: [kJ/kg C]

In the above equation the c_p values are considered as constant (approximation).

Graphically this means that the heat characteristic curves of the two heat flows must be in a certain relationship as shown here:



What are the practical consequences for a cement kiln? We consider:

A) Raw meal preheater
B) Clinker cooler

A) Raw meal preheater

1) Heat transfer rate:
   * Optimum heat transfer rate and optimum specific surface (small particles) and distribution has been almost achieved in the cyclone suspension preheater. After every cyclone stage material and gas come to almost complete temperature approach and there is usually little to improve on that.
   * Improvement are rather to be done where the heat exchange takes place in the rotary part itself. Especially on wet and long dry kilns the kiln internal fittings are essential for good heat exchange.

2) Counter-current principle:

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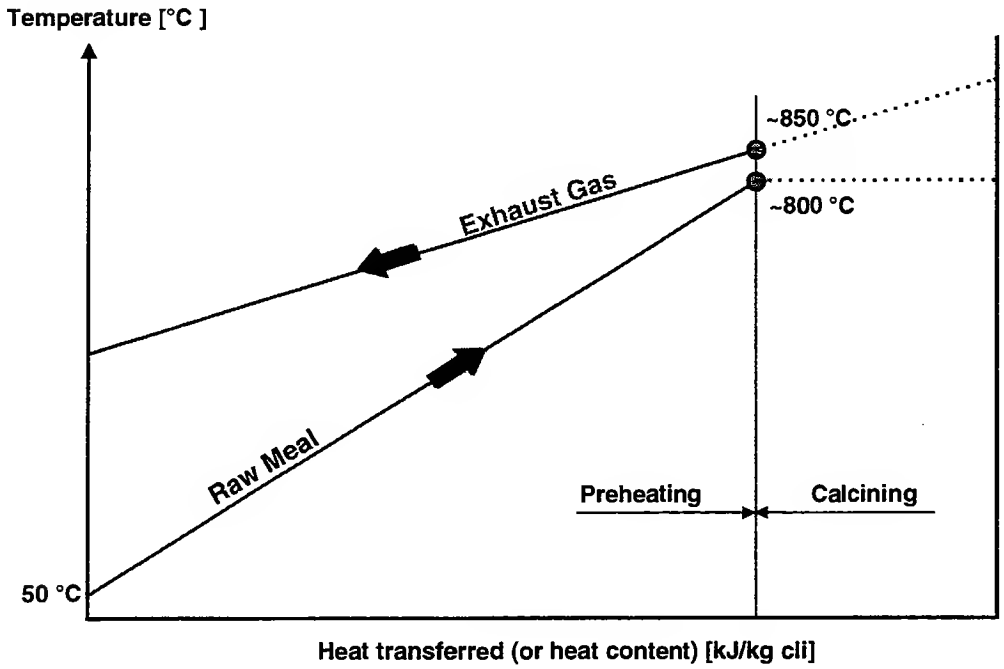
* Counter-current flow in its proper sense does not exist in case of a cyclone suspension preheater. Instead, we have usually four co-current heat exchanging units, but the overall arrangement acts as counter-current system. To reach an ideal state one would have to apply an infinite number of cyclone stages. Practically the common arrangement of 4 or 5 stages can be considered as sufficient.
* True counter-current preheaters are shaft preheaters or preheating in long rotary kilns. Such preheaters would theoretically be ideal. Practically they are less efficient because of distribution problems and backmixing effect (internal material circulations) and comparatively high losses to outside (in case of a rotary kiln).

3) Equivalence of heat flow characteristics:
   * A general feature of any raw meal preheater is the surplus of heat input by the hot gases. After the calcining step the hot gas has a temperature of approx. 850°C and the specific quantity is always above ~ 1.3 Nm^3/kg cli even in case of an optimum kiln system (4-stage SP kiln). This amount of heat is too much, it cannot be used completely for preheating of raw meal (~ 1.56 kg RM/kg cli) up to calcining temperature (~ 800°C). Therefore a certain amount of waste heat will occur even in an ideal case. Theoretically we should not exceed ~ 1 Nm^3/kg cli for ideal recuperation. Practically this cannot be realized, not even on modern kiln systems which produce comparatively little exhaust gas.
   * Graphically we have the following heat situation in a raw meal preheater:

Fig. 16



Above diagram is simplified but typical for any preheater.

Because of the „heat surplus" of the exhaust gas it is not possible to achieve an ideal recuperation even at perfect counter-current heat transfer (e.g. infinite number of cyclone stages). The exhaust gas will always give a certain residual heat content.

Practically this means that all kind of improvements on the preheater have a limited potential. In the example of a 4-stage preheater kiln we can add a fifth stage which causes a reduction of about 100 kJ/kg cli in fuel heat consumption. More than 5 stages will bring only marginal economical point of view. *) Even at 5 stages we may check if the necessary investment and the (possible) increase of pressure drop can be justified by the local cost structure.

*) Instead of constructing more than 5 conventional cyclone stages one would prefer here „non-conventional" preheaters, such as the cross-suspension-preheater (two strings with cross flow of raw meals).

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B) Clinker Coolers

1) Heat transfer rate:
   * On a planetary or a rotary cooler we have the possibility to increase the heat transfer rate by installation of efficient internal equipment (tumblers, lifters) which increase the active heat transfer area by better moving and scattering of the clinker. If a cooler has worn out internal equipment or equipment of inadequate design we may realize a considerable potential for improvements. Improvements may also result from a more uniform clinker granulometry (less internal dust circulations).
   * On a grate cooler we find quite a different situation. The real problem is not the heat transfer rate between a piece of clinker and the cooling air but rather the uniform air distribution through the clinker bed. Also here we may realize a considerable improvement (thick bed operation, mechanical modifications at inlet zone etc.).

2) Counter-current principle:
   * There is an obvious difference between planetary/rotary coolers and the grate cooler:
     - planetary / rotary → counter-current flow
     - grate cooler → almost cross-current flow
   * The grate cooler has a nearly cross-current performance and has therefore, from its principle, a limited heat recovery potential. Improvements are possible when air recirculation (of hot waste air) to the first grate section is applied in order to increase the heat content of the secondary air. Another quite different measure is the increasing of the bed thickness in order to come more towards a counter-current-like exchange (similar to a shaft cooler). The disadvantage is an increase of the cooling air pressure.

3) Equivalence of heat characteristic
   * On a clinker cooler we would theoretically require approx. 0.77 Nm^3/kg cli of cooling air in order to transfer the clinker heat completely to the secondary air (under perfect counter-current heat exchange).
   * The practical figures are usually above 0.8 Nm^3/kg cli. Compared to a raw meal preheater we have an inverse situation: The heat from the hot clinker could theoretically be completely recovered (under ideal conditions)! Of course, we know that the common, practical figures are often below 70%. The latter fact illustrates that from the basic principle there is still a considerable heat potential which is not used for reducing the kiln heat consumption.
   * In this context it is also important to note that the practical efficiency of any type of clinker cooler increases with higher cooling air quantity. As a consequence we should draw as much cooling air as possible through the cooler and therefore avoid or reduce excessive primary air quantities or false air inleaks at the kiln hood or kiln seals. The improvement does not come from the heat transfer proper but rather from the improved „heat characteristic" (air/clinker ratio).

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7. TEST QUESTIONS

1) Give an example where it can be worthwhile to execute a complete heat balance on a kiln.
2) Summarize all important measurement points which are needed for doing a complete heat balance on a suspension preheater kiln.
3) What is the usual value (or range) for the specific heat (kJ/Nm^3 C) of exit gas of an SP-kiln at 350°C?
4) What is the sensible heat content of 1 kg clinker at 1450°C, expressed as kJ/kg cli?
5) What is the standard value (or range) for heat of formation (kJ/kg cli) for clinker burning? Which heat effects are included in above value?
6) Determine the heat transfer coefficient (W/m^{2}C) for the total heat transfer by radiation plus free convection. The temperature of the kiln shell section is 200°C (ambient = 20°C, $\varepsilon = 0.9$).
7) When has the forced convection heat transfer (instead of free convection) to be considered?
   How is the above value affected in case of smaller dimensions (say 1 m instead of 5 m diameter)?
8) If the shell losses in the calcining zone can be reduced by 50 kJ/kg cli, what will be the approximate saving of fuel heat (kJ/kg cli)?
9) What is the approximate fuel heat which can be saved through the installation of a fifth cyclone stage on a suspension preheater kiln?
   What would be the approximate amount of false air reduction (Nm^3/kg cli into the high temperature zone) in order to achieve a similar fuel heat saving?

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9. SYMBOLS AND UNITS

A		m^2	area
C_R		W/m^2K^4	radiation constant
c_P		kJ/kg C	specific heat (at const. pressure),
	or	kJ/Nm^3 C	specific heat capacity
CV		kJ/kg	calorific value
D		m	diameter
g		m/s^2	gravity constant
h		kJ/kg	heat content (specific)
	or	kJ/Nm^3	
	or	kJ/kg cli	
L		m	length
m		kg	mass
	or	kg/kg	specific mass
m_f		kg/h	mass flow
Q_f		kW	heat flow (1 kW = 1 kJ/s)
t		C	temperature (Centigrade)
T		K	temperature (Kelvin)
v		m/s	velocity
w		kg/kg	water content

Greek Letters

α	W/m^2K	heat transfer coefficient
ε	-	emissivity (for radiation)
λ	W/m C	heat conductivity
ρ	kg/m^3	density

Dimensionless Numbers

Nu	Nusselt number (for heat transfer)
Pr	Prandtl number
R	kiln feed (raw meal) / clinker-ratio
Re	Reynolds number

Indices

conv	convection
rad	radiation
tot	total
o	ambient condition or zero condition

Conversion Factors

Length	1 inch	0.0254 m
	1 ft	0.3048 m
Area	1 sq. ft	0.092903 m^2
Volume, Volume Flow	1 cu.ft	0.028316 m^3
	1 cu.ft/min	1.699 m^3/h (actual m^3)
Mass	1 lb.	0.45359 kg
	1 short ton (USA)	907.185 kg

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Pressure	1 bar	10^5 N/m^2
	1 atm.	1.013 bar
Energy	1 kJ	1000 J
	1 MJ	1000 kJ
	1 kWh	3600 kJ
	1 kcal	4.187 kJ
	1 BTU	1.055 kJ
Temperature Conversion	C =	5/9(F - 32)
	K =	273.15 + C
Heat Flow	1 kW	1000 W = 1 kJ/s
	1 kcal/h	1.163 W
	1 BTU/h	0.29307 W
Specific Heat	1 kcal/kg C	4.187 kJ/kg C = 4187 J/kg C
	1 BTU/lb F	1 kcal/kg C = 4.187 kJ/kg C
Heat Transfer Coeffic.	1 kcal/m^{2}h C	1.163 W/m^2 C
	1 BTU/ft^{2}h F	5.678 W/m^{2}C
Standard Conditions for Gases	Standard Conditions	0°C and 1 atm. (1.013 bar)

$$Nm^3 = act.m^3 \times \frac{2.73.15}{273.16 + t(c)} \times \frac{p(bar)}{1.013bar}$$

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Energy Balance of a Kiln

Calculation example

1. General kiln data

Supplier	Polysius
Type of process	4-stage suspension preheater
Start up	1975
Nominal Capacity	2'000 t/d
Kiln dimension	ϕ 4.6 x 70 m
Cooler type	planetary
Cooler tubes	9 x ϕ 2.0 x 19.8 m
Firing system	coal, indirect
Coal dosing	belt scale
Burner type	KHD/Pyrojet
Kiln feed system	air lift
Simplified flow sheet:	see attached sheet

2. Results of measurements

2.1 Mass and gas flows

Clinker (from weighing)	91.7	t/h (2'200 t/d)
Coal	10.5	t/h
Kiln feed	151	t/h
Dust in exhaust gas*	10	t/h
Air lift air	9'000	Nm^3/h
Total air into burner pipe	6'500	Nm^3/h
Exhaust gas after preheater	n.d. (to be determined by calculation!)	
Water feed into planetary cooler (internal)	3	m^3/h

* collected in EP and weighed during direct operation

2.2 Results from laboratory

- Calorific value of coal (net, as fed to burner)	28'600	kJ/kg
- Moisture of kiln feed	0.6	%
- LOI of kiln feed	35.8	%

2.3 Gas analysis

At kiln inlet:	O_2	= 1.5 %
At preheater exit:	O_2	= 4.5 %
	CO	= n.d. (< 0.1 %)

2.4 Temperature measurements

- Gas temperature after preheater	370	°C
- Kiln feed (= air lift)	50	°C
- Clinker outlet (planetary cooler)	160	°C
- Air to burner (weighted average)	80	°C
- Ambient air	20	°C

2.5 Measurements of radiation and convection losses (by radiation pyrometer)

- Kiln shell losses, calculated section by section from shell temperature profile — 4.8 MW

- Planetary cooler losses, calculated section by section from temperature profile of 9 tubes and by adequate assumption for reference are (outer envelope) — 5.0 MW

- Preheater losses
  Because of comparatively uniform shell temperatures the following simplified approach can be applied:

Total of all surfaces (cyclones, riser pipes, ducts)	≈ 2'000	m^2
Average surface temperature	≈ 100	°C
Emissivity	≈ 0.9	
Wind velocity	≈ 1	m/s

3. Energy balance

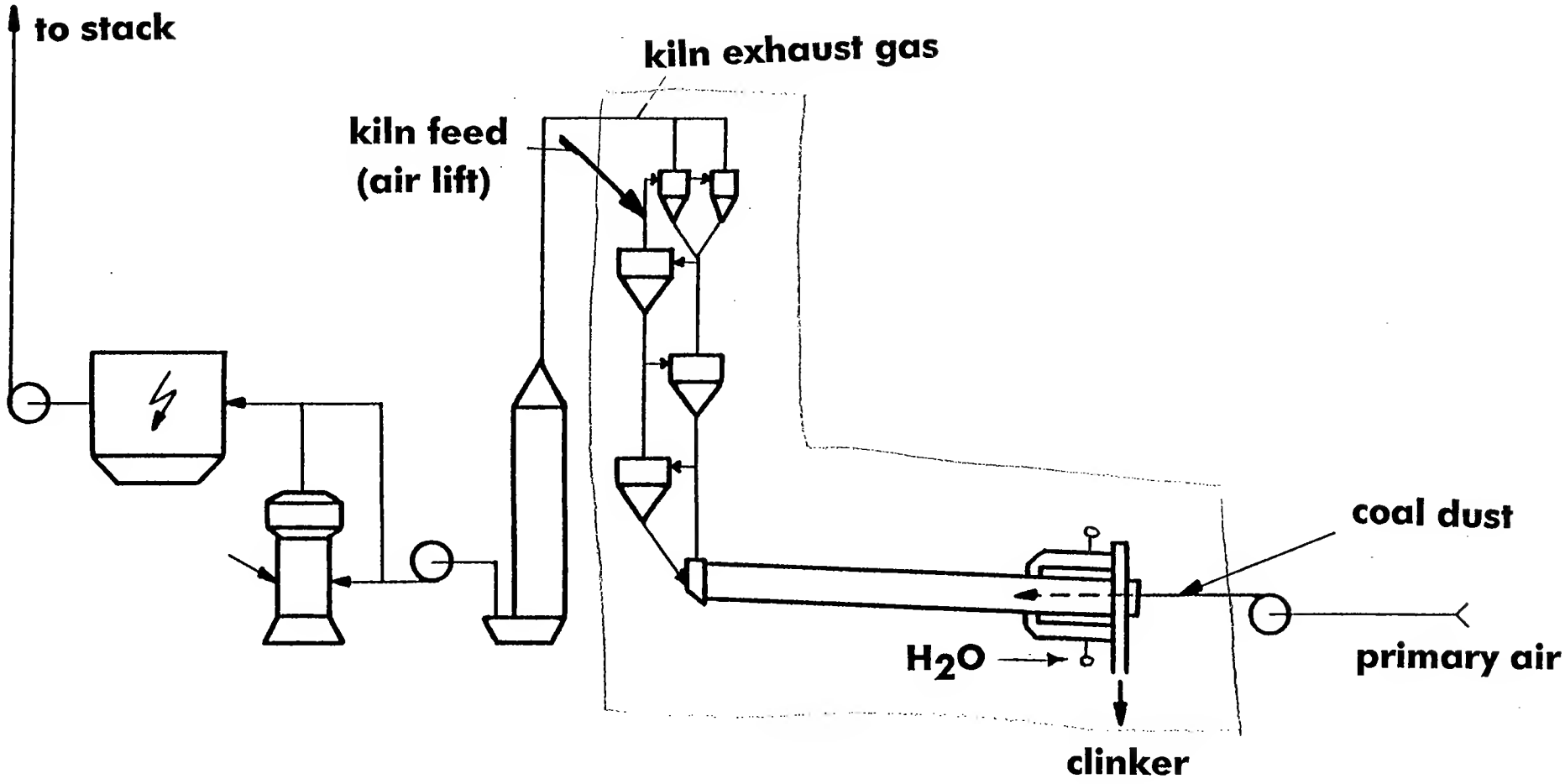
Calculate the heat items according to course documentation and fill in the attached sheet "Energy balance of kiln".

ENERGY BALANCE OF KILN

	Specifications	Temp. [°C]	Energy [kJ/kg cli]
INPUT Fuel combustion Primary air: sensible heat			
Kiln feed: sensible heat Air lift air: sensible heat			
Total			
OUTPUT Heat of formation			
Water evaporation: - kiln feed - cooler			
Exhaust gas: - sensible heat - dust sensible heat - CO-loss			
Clinker ex cooler: - sensible heat			
Radiation and convection: - kiln - cooler - preheater			
Rest			
Total			

Reference: 20 °C

Energy Balance
Simplified Flow sheet



ENERGY BALANCE OF KILN (Solution)

	Specifications / Remarks	Temp. [°C]	Energy [kJ/kg cli]
INPUT			
Fuel combustion	0.1145 kg x 28'600 kJ	-	3'275
Primary air: sensible heat	0.071 Nm^3, c_p = 1.30 kJ/Nm^3C	80	6
Kiln feed: sensible heat	1.647 kg, c_p = 0.86 kJ/kg C	50	42
Air lift air: sensible heat	0.098 Nm^3	50	4
Total			**3'327**
OUTPUT			
Heat of formation	"standard case"	-	1'750
Water evaporation:			
- kiln feed	0.010 kg}	-	24
- cooler	0.033 kg} x 2'450 kJ/kg	-	80
Exhaust gas:			
- sensible heat	1.55 Nm^3 1), c_p = 1.51 kJ/Nm^3C	370	819
- dust sensible heat	0.109 kg, c_p = 1.03 kJ/kg C	370	39
- CO-loss	<0.0013 Nm^3(dry)x 12600kJ/Nm^3	-	(< 16)
Clinker ex cooler:			
- sensible heat	1 kg, c_p = 0.82 kJ/kg C	160	115
Radiation and convection:			
- kiln	4.8 MW	-	188
- cooler	5.0 MW	-	196
- preheater	2.4 MW 2)	(100)	94
Rest	-	-	22
Total			**3'327**

Reference: 20 °C

1) **Calculation of exhaust gas at 4.5 % O_2**

See paper combustion, gas flow and gas composition (Chapter 4.3.2)

. From combustion	= $3.275 \cdot 0.28$	=	0.917 Nm^3/kg
. From raw meal (CO_2 + hydrate water)		=	0.28 Nm^3/kg
. Excess + false air : $(3.275 \cdot 0.25 + 0.28) \frac{4.5}{21 - 4.5}$		=	0.300 Nm^3/kg
. H_2O cooler and kiln feed = $(0.033 + 0.010)/0.8$		=	0.053 Nm^3/kg
Total exhaust gas		=	1.550 Nm^3/kg cli

2) **Calculation of radiation and convection losses of preheater**

From Fig. 11

$t - t_o = 80°C$, $\epsilon = 0.9$ $v = 1m/s \rightarrow \alpha$ = 15 W/m^2C

$Q = 2'000 \cdot (100 - 20) \cdot 15$ = 2'400 kW

$h = \frac{2'400 \cdot 3.6}{91.7\ t/h}$ = 94 kJ/kg cli

Heat losses of kiln shell (radiation and convection) "Holderbank"
********** HMC VA/Ot
Plant : Typical calculation example 11.03.91
Date : 11/3/91 11:12

Kiln shell diameter	[m]	4.6	Amb.temp.	[C]	20
Kiln length	[m]	70	Abs.press	[bar]	0.98
Tot.shell area	[m2]	1011.59	Air dens.	[kg/m3]	1.17
Number of sections	[-]	30			
			Emissivity	[-]	0.9
Clinker production	[t/h]	91.7	Cr	[W/m2K4]	5.67

Pos. from outlet From [m]	To [m]	Section length [m]	Shell section [m2]	Temp. [C]	Radiation [W/m2C]	Conv. [W/m2C]	Q tot [KW]
0.0	2.4	2.40	34.68	211	12.69	8.13	137.9
2.4	4.8	2.40	34.68	279	16.84	8.75	229.9
4.8	7.2	2.40	34.68	283	17.11	8.78	236.2
7.2	9.6	2.40	34.68	282	17.04	8.77	234.6
9.6	12.0	2.40	34.68	247	14.78	8.48	183.1
12.0	14.4	2.40	34.68	208	12.53	8.10	134.5
14.4	16.8	2.40	34.68	166	10.41	7.58	91.1
16.8	19.2	2.40	34.68	161	10.17	7.51	86.5
19.2	21.6	2.40	34.68	191	11.63	7.90	115.9
21.6	24.0	2.40	34.68	173	10.74	7.68	97.7
24.0	26.4	2.40	34.68	303	18.52	8.93	269.4
26.4	28.8	2.40	34.68	300	18.30	8.91	264.2
28.8	31.2	2.40	34.68	261	15.66	8.60	202.8
31.2	33.6	2.40	34.68	288	17.46	8.82	244.2
33.6	36.0	2.40	34.68	248	14.84	8.49	184.5
36.0	38.4	2.40	34.68	208	12.53	8.10	134.5
38.4	40.8	2.40	34.68	180	11.08	7.77	104.6
40.8	43.2	2.40	34.68	128	8.74	6.98	58.9
43.2	45.6	2.40	34.68	233	13.94	8.35	164.7
45.6	48.0	2.40	34.68	250	14.96	8.51	187.2
48.0	50.4	2.40	34.68	239	14.30	8.41	172.4
50.4	52.8	2.40	34.68	185	11.33	7.83	109.6
52.8	55.2	2.40	34.68	246	14.72	8.47	181.8
55.2	57.6	2.40	34.68	232	13.88	8.34	163.4
57.6	60.0	2.40	34.68	230	13.76	8.32	160.9
60.0	62.4	2.40	34.68	221	13.25	8.23	149.7
62.4	64.8	2.40	34.68	224	13.42	8.26	153.4
64.8	67.2	2.40	34.68	221	13.25	8.23	149.7
67.2	69.6	2.40	34.68	241	14.42	8.43	175.1
69.6	70.0	0.40	5.78	199	12.05	8.00	20.7
Total		70.00	1011.59				4799.0
		**********	**********				*********
Minimum		0.40	5.78	128	8.74	6.98	
Maximum		2.40	34.68	303	18.52	8.93	

HEAT BALANCE OF SUSPENSION PREHEATER KILN WITH PLANETARY COOLER

PLANT Cement Course
KILN N° Example
DATE

Reference temperature	20 °C
Ambient temperature	20 °C
Ambient pressure	972 mbar
Production rate	91.7 t/h
Specific heat consumption	3275 kJ/kg cli

		SPECIFICATIONS			ENERGY		*)
		[kg/kg cli], [Nm3/kgcli] [ppm], etc.	LHV cp [kJ/kg°C]	T [°C]	[kJ/kg cli]	[kcal/kg cli]	
INPUT							
Fuel	conventional fuel				3274.8	782.2	21
	alternative fuel						
	sensible heat						
Kiln feed	sensible heat	1.647	0.85	50	43.0	10.3	22
	combustible matter						
Primary air	sensible heat	0.071	1.299	80	5.5	1.3	23
Cooler	sensible heat of air	0.781			0.0	0.0	24
	sensible heat H2O		4.194	20	0.0	0.0	
Other input	air lift air	0.098	1.3	50	3.8	0.9	25
TOTAL OF INPUTS					3327.1	794.7	
OUTPUT							
Heat of clinker formation					1750.0	418.0	31
H2O evaporation	kiln feed	0.010	2450		24.2	5.8	32
	planetary cooler	0.033	2450		80.2	19.1	
Exhaust gas	sensible heat	1.549	1.513	370	820.2	195.9	33
	unburnt gases (CO)						
	dust	0.109	1.035	370	39.5	9.4	
Cooler clinker exit	sensible heat	1	0.809	160	113.3	27.1	34
Bypass losses	sensible heat						35
	unburnt gases (CO)						
dust	sensible heat						
	CaO (non carbonatic)						
Radiation & convection	preheater				94.0	22.5	36
	rotary kiln				188.0	44.9	
	planetary cooler				196.0	46.8	
Other output							37
Rest 0.7 %					21.8	5.8	38
TOTAL OF OUTPUTS					3327.1	794.7	

*) Remarks

Chapter 10

Main Fans

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Main Fans

Authors: W. Zeller, Th. Richner, D. Brassel
PT 99/14500/E

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1. DESIGN AND EFFICIENCY OF FAN IMPELLERS

Fans are essential components of the cement manufacturing process and merit therefore particular considerations with respect to

- design and efficiency of the impeller
- fan size and operating point
- flow control
- reliability

Main fans in a Cement Plant are found as

- Kiln ID Fan
- Raw Mill Fan
- Cooler Exhaust Gas Fan
- Kiln Dedusting Fan

Altogether these fans consume between 30 and 50% of the plants total electrical energy.

1.1 Fan impeller types

There are four basic blade forms used in industrial induced draft service:

- backward airfoil blades
- backward curved blades
- backward inclined blades
- radial straight blades

Fig. 1.1 shows design and efficiency of these impellers.

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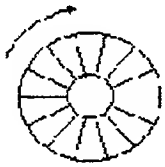
Fig. 1.1: Design and efficiency of impellers

TYPE		EFFICIENCY η	APPLICATION
backward airfoil blades		**> 84 %**	**for clean gas applications (dust < 50 g/m^3)**
backward curved blades		**up to 82 %**	**for gas with a dust concentration < 100 g/m^3**
Backward inclined blades		**up to 80 %**	**for gas with a dustload up to 100 g/m^3**
radial straight blades		**60 - 75 %**	**for gas with a high dustload (dust > 100 g/m^3)**

1.2 Selection criteria

It is of great importance that critical process equipment, such as fans, is selected on the basis of proven ability in order to provide maximum reliability rather than on an efficiency rating.

In situations where more than one blade form will meet a performance requirement, it then becomes necessary to select the one form that will be most overall cost-effective.

For the selection process the supplier should provide the operating and service manual for the equipment type being considered.

The operating and service report should include all occurrences that require fan maintenance.

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To assist in fan type selection, there are at least four important points to be considered.

- **Fan efficiency:**
  Because many of the higher efficiency fans can only achieve their stated efficiency within a narrow operating range, a true energy evaluation must consider the actual operating point and alternate operating points on a time basis.
  Many systems include a built-in safety factor, which results in reduced efficiency when operated at constant speed with damper regulation.
- **Continuous operation:**
  Any type of equipment will require maintenance. In blade form selection, blade build-up (cp. section 4.2) and erosion (cp. section 4.3) have the greatest effect on fan operation. Build-up on the wheel results in reduced performance. There is an increased tendency for material to build up on blades as the blade angle is tilted back from radial. This build-up can accumulate to the degree that it restricts and alters air passages, reducing both efficiency and performance.
- **Mechanical design:**
  All fan rotors are subject to centrifugal force. Depending on blade form and angle, different types of stress occur in a blade. The radial Blade is in tension, while bending and tensile stresses act on the backward inclined/Airfoil design. Bending stresses are more subject to fatigue stresses.
- **Equipment costs**

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1.3 Fan applications in the cement industry

The following table shows an overview of fan applications in the cement industry.

Location	Dust load [g/Nm3]	Blades mainly used	Max. Temp [°C]	max. speed [rpm]	Flow regulation	Rotor protection	Stator protection	Efficiency [%]
Coal Mill								
Filter exhaust	< 0.15	F/C/A	150	1800	VC/ILD	(WP)	-	70 - 85
Separator / cyclone exhaust	< 100	R/F/C	150	1200	ILD	HSWP	WP(HSWP)	55 - 75
Raw Mill								
Filter exhaust	< 0.15	F/C/A*	300	1800	VC/ILD/VS	-	-	70 - 85
Separator / cyclone exhaust	< 100	F/C	300	1200	ILD/VS	(HW/HWSP)	WP	65 - 75
Cement Mill								
Filter exhaust	< 0.15	F/C/A	200	1800	VC/ILD	-	-	75 - 85
Separator / cyclone exhaust	< 100	F/C	200	1200	ILD	(WP/HSWP)	WP	65 - 75
Dry Process								
Preheater exhaust	< 100	R/F/C	450	1200	ILD/VS	(WP)	-	55 - 75
Kiln line filter exhaust	< 0.15	F/C/A*	350	1200	VC/ILD/VS	-	-	70 - 85
Semi-dry process								
Recirculation fan	< 20	R/F	450	750	ILD/VS	HSWP	WP	60 - 70
Kiln line filter exhaust	<0.15	F/C/A*	450	1800	VC/ILD/VS	-	-	70 - 85
Cooler								
Filter exhaust	<0.15	F/C	450	1200	VC/ILD/VS	(WP)	-	70 - 88
Filter exhaust	0.5	F/C	450	1200	ILD/VS	WP	(WP)	70 - 80
Cyclone exhaust	< 5	R/F	450	1200	ILD/VS	HSWP	WP	65 - 75
Recirculating	< 15	R	450	750	ILD	HSWP	WP(HSWP)	60 - 70
Cooler F.D. fans	Traces	F/C/A	50	2200	VC/VS	-	-	75 - 85

* in case of bag filter

Abbreviations:

a. Blade Type
- R: radial blade
- F: backward inclined flat plate
- C: backward curved plate
- A: backward airfoil blade

b. Flow regulation (cp. section 3)
- VC: Vane control set at the fan inlet
- ILD: Inlet louver damper set on the inlet box
- VS: Variable speed

c. Rotor and stator protection against abrasion (cp. section 0)
- WP: bolted or welded wear plates
  - mild steel
  - quenched steel
  - wear-resistant steel
- HSWP: bolted mild steel wear plates with hard surfacing
  - chromium carbide
  - tungsten carbide
- (WP/HSWP): optional

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2. FAN PERFORMANCE CURVES

2.1 System Resistance Curve

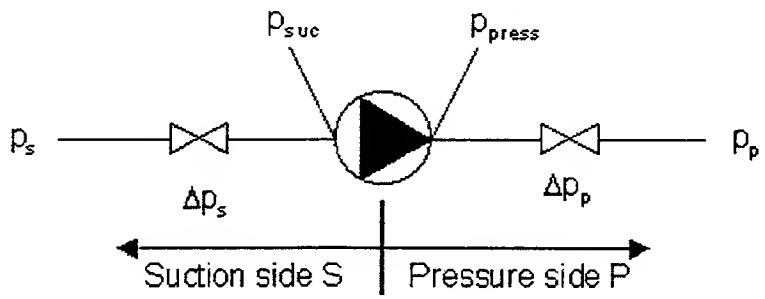
The System Resistance Curve is the relation between pressure and volume flow for the given system. For a system in which only air is moved and the geometry of the ductwork is constant, the pressure drop is proportional to the square of the flow rate.

In most of the applications around the cement plant, this curve is constantly changing because of the changing system conditions. For instance, the cooler under-grate fan system resistance curve depends not only on the geometry of the ductwork and cooler, but also on size distribution and thickness of the clinker bed on the grate above the compartment.

The system resistance curve of a roller mill system is dependent on the geometry of the ducts, the amount of material being transported by the gas, the composition of the gases and the speed of the classifier. Because of these variables, it is essential to understand that a system resistance curve can vary significantly from the slope of the curve illustrated in e.g. Fig. 2.3.

Principally the curve is derived as described next. Fig 2.1 schematically shows a system.

Fig. 2.1: System with fan



Total pressure (dynamic and static part) at S:

$p_{suc} = p_s - \Delta p_s$ with Δp_s = pressure loss due to friction of the suction side $= f(\dot{V}^2)$

Total pressure (dynamic and static part) at P:

$p_{press} = p_p - \Delta p_p$ with Δp_p = pressure loss due to friction of the pressure side $= f(\dot{V}^2)$

Pressure difference Δp_{fan}, which has to be produced by the fan, consists of a static and dynamic part and is a function of the gas velocity or gas flow.

$$\Delta p = p_{press} - p_{suc} = = \Delta p_{stat} + \Delta p_{dyn} = f(w^2) = f(\dot{V}^2)$$

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2.2 Fan curves

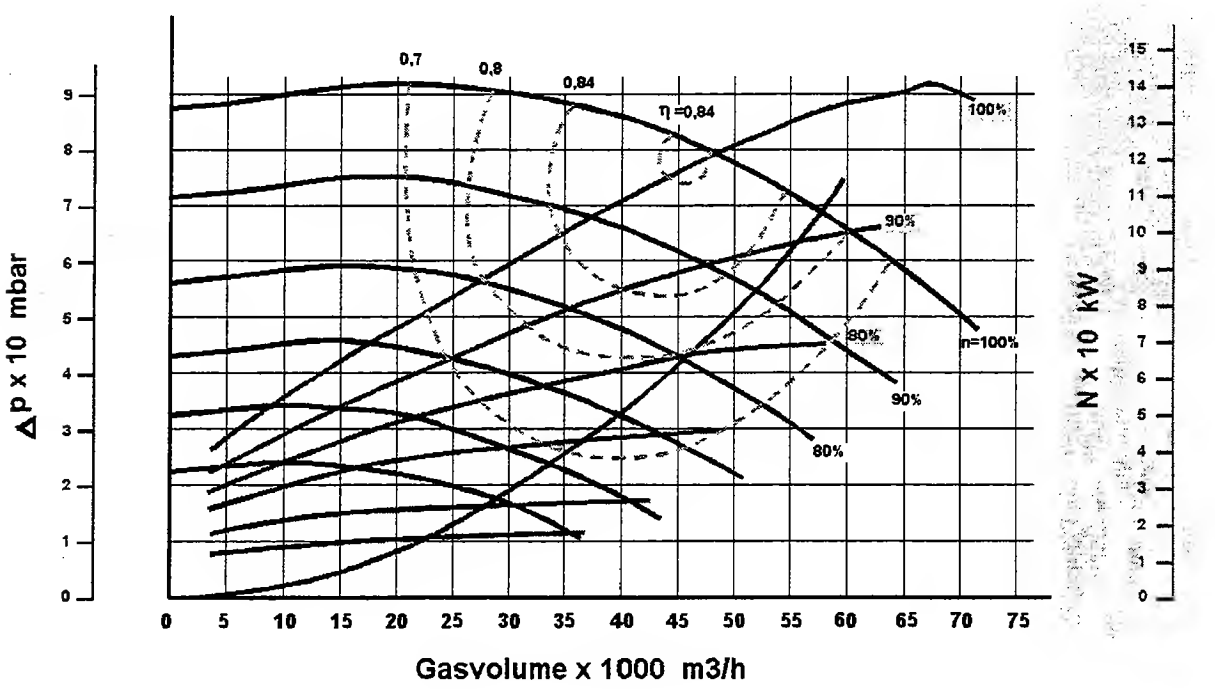
The fan performance curve is derived from laboratory data when the flow conditions to and from the fan are ideal. Since these conditions seldom exist in cement plant fan locations. The fan curve data cannot be considered reliable when applied to field conditions. Because of this uncertainty of the prevailing conditions, we find that engineers specify oversized fans during the plant design stage.
For example, typical factors that can effect the performance of an under-grate cooler fan include a dirty inlet screen; a structural column too close to the inlet; a silencer at fan inlet or the influence of an adjacent fan. Likewise, similar factors influencing an induced-draft fan include asymmetrical inlet duct which effects streamline distribution to the fan or heavier than normal blade wear pads that restrict blade passage geometry.

A typical fan curve (see Fig. 2.2) shows the quantity of air on the horizontal axis and the fan static pressure and fan power plotted on the vertical axis. The conditions of density and flow are at the inlet of the fan.

The actual **operating point** of the fan will be the intersection of the fan curve with the system resistance curve (cp. section 2.1).

The efficiency of a fan is best close to its design point of operation. The farther off these "ideal" conditions, the lower the efficiency. The operating points of equal fan efficiency are located on oval shaped curves around the maximum point.

Fig. 2.2: Fan characteristics and efficiencies by different fan speed



The fan curve depends on the impeller speed and the physical properties of the gas (cp. section 2.2.2)

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The pressure which is produced by the fan equals the pressure difference between the total pressure at the fan outlet and the fan inlet:

$$\Delta p = p_{press} - p_{suc} = = \Delta p_{stat} + \Delta p_{dyn}$$

For most fans in cement plants, it is satisfactory to assume that the fan's pressure is the difference between the static pressure at the inlet and outlet of the fans.

The fan power is the power at the fan shaft. For most plant applications where the fan has constant speed and is direct coupled, this can be assumed to be 96 % of the power consumed by the drive motor when the motor is fully loaded. Since power factor and motor efficiency vary with motor load, it is difficult to measure fan shaft power input without a kilowatt meter. However, near full motor load conditions, the amp reading is a good indicator of shaft power. For fans with variable speed drives, especially hydraulic or magnetic clutch drives, the efficiency of the drive can be very poor so the power at the fan shaft cannot be assumed to be a linear function of the power consumed by the motor.

2.2.1 Fan equations

Power requirement:

$$N = \frac{\dot{V}\,\Delta p\,10^{-3}}{\eta} \quad [kW]$$

N	: Required power	[kW]
$\dot{V}$	: Gas flow	[m^3/s] (**not** [Nm^3/s])
Δp	: Total increase of pressure in fan	[Pa]
η	: Fan efficiency	[-]

Depending on the blade shape of the impeller, the power requirement to draw the gas through a given system (and therefore for determined pressure losses) can vary in a relatively wide range.

As the required fan power is given by the equation above, it is obvious that considerable savings can be achieved with the most efficient impeller.

Total pressure increase:

Formula for a rough calculation of the total pressure $\Delta p = p_{press} - p_{suc} = \Delta p_{stat} + \Delta p_{dyn}$:

$$\Delta p \approx \frac{0.6\,\rho\,n^2\,d^2\,\pi^2}{3600} \quad [Pa]$$

ρ	: Density	[kg/m^3]
n	: Fan speed	[rpm]
d	: Impeller diameter	[m]

Remark: Δp is just a rough guideline and depends very much on the blades shape and the rotor efficiency.

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Example: Effect of fan efficiency

Plant: 4-stage SP kiln, 3000 t/d clinker

Requirement for kiln ID-fan: $\dot{V}$ = 115 m³/s at 350°C
Δp = 6000 Pa
→ two efficiencies available: η_1 = 0.75
η_2 = 0.85

$$N_1 = \frac{\dot{V}\,\Delta p\,10^{-3}}{\eta_1} = \frac{115 \cdot 6000 \cdot 10^{-3}}{0.75} = 920\ kW$$

$$N_2 = \frac{\dot{V}\,\Delta p\,10^{-3}}{\eta_2} = \frac{115 \cdot 6000 \cdot 10^{-3}}{0.85} = 812\ kW$$

Power saving = **108 kW** = **0.86 kWh/t cli**

Assuming an operation time of 7500 h/year and an energy price of 0.05 US$/kWh the yearly saving will amount to **40'500 US$!**

2.2.2 Adjusting fan performance curves

It was mentioned above that the fan performance curve is given at specific conditions of density and fan speed. Most often, it is necessary to correct the fan curve for density and speed other than the predicted conditions. Fig. 2.3 shows the influence of density changes on the fan performance.

Indices: 1: reference conditions (or original fan curve characteristic)
2: actual conditions

Gas density correction:

A correction of the density is necessary if:

- temperature changes
- the chemical composition of gas changes
- the altitude changes (height above sea level).

$$\frac{p_{stat1}}{p_{stat2}} = \frac{\rho_1}{\rho_2} = \frac{T_2}{T_1}$$

Volume flow and efficiency of the fan are unaffected by gas density changes.

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Remark: ① Calculation of actual density

$$\rho = \frac{M}{22.4} \cdot \frac{p}{1013} \cdot \frac{273}{T + 273}$$

ρ	density	[kg/m^3]
M	molecular weight of gas	[kg/kmol]
p	actual pressure	[mbar]
T	actual temperature	[°C]

② Ambient pressure, depending on the altitudes can be calculated by:

$$p = 1013 \cdot \exp[-0.001255 \cdot h]$$

h	altitude above sea level	[m]

Gas	Density [kg/Nm^3]	Molecular weight [kg/kmol]
O_2	1.429	32
CO_2	1.964	44
N_2	1.250	28
Air	1.292	29
H_2O	0.804	18

Fan speed correction:

Volume flow: $$\dot{V}_2 = \dot{V}_1 \cdot \left(\frac{n_2}{n_1}\right)$$

Static pressure: $$p_{stat2} = p_{stat1} \cdot \left(\frac{n_2}{n_1}\right)^2$$

Power requirement: $$N_2 = N_1 \cdot \left(\frac{n_2}{n_1}\right)^3$$

(for $\eta_1 = \eta_2$)

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Fan wheel dimensions correction:

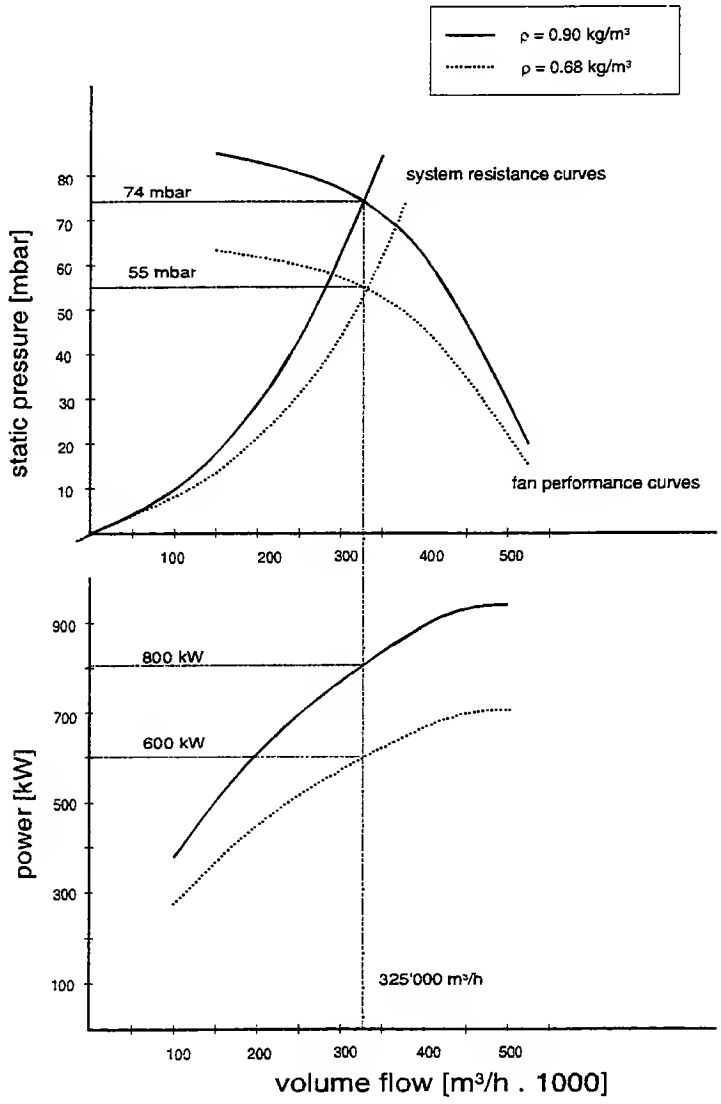
Volume flow:
$$\dot{V}_2 = \dot{V}_1 \cdot \left(\frac{d_2}{d_1}\right)^3 \cdot \left(\frac{b_2}{b_1}\right)$$

Static pressure:
$$p_{stat2} = p_{stat1} \cdot \left(\frac{d_2}{d_1}\right)^2$$

Power requirement:
$$N_2 = N_1 \cdot \left(\frac{d_2}{d_1}\right)^5 \cdot \left(\frac{b_2}{b_1}\right)$$

whereas d impeller diameter [m]
b impeller width [m]

Fig. 2.3: Influence of density changes on system resistance and fan performance curves



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3. FLOW CONTROL

Whether a fan ever will operate at a high efficiency is to a large extent already decided when the fan size is selected.

Since the efficiency of the fan has a maximum at a specific operating point the fan ought to be sized so that it can operate at such conditions most of the time. However, since the fan performance needs to be flexible to meet variable requirements, the fan size will, to some extent at least, be designed for the maximum requirement. Reduced requirements have to be met by fan control dampers or variable speed control.

Besides the efficiency of the fan itself, the type of flow control has the strongest influence on the overall power consumption.

Damper control generally results in higher power consumption and so more if the damper is installed at the fan outlet. Therefore the damper should always be installed at fan inlet. Inlet vane control may be satisfactory from 100 % down to about 70 % of maximum flow, but the power demand becomes high when the flow is reduced further. Speed control is virtually ideal.

Fig 3.1 shows different arrangements for inlet and outlet dampers.

Fig. 3.1: Design possibilities for fan dampers

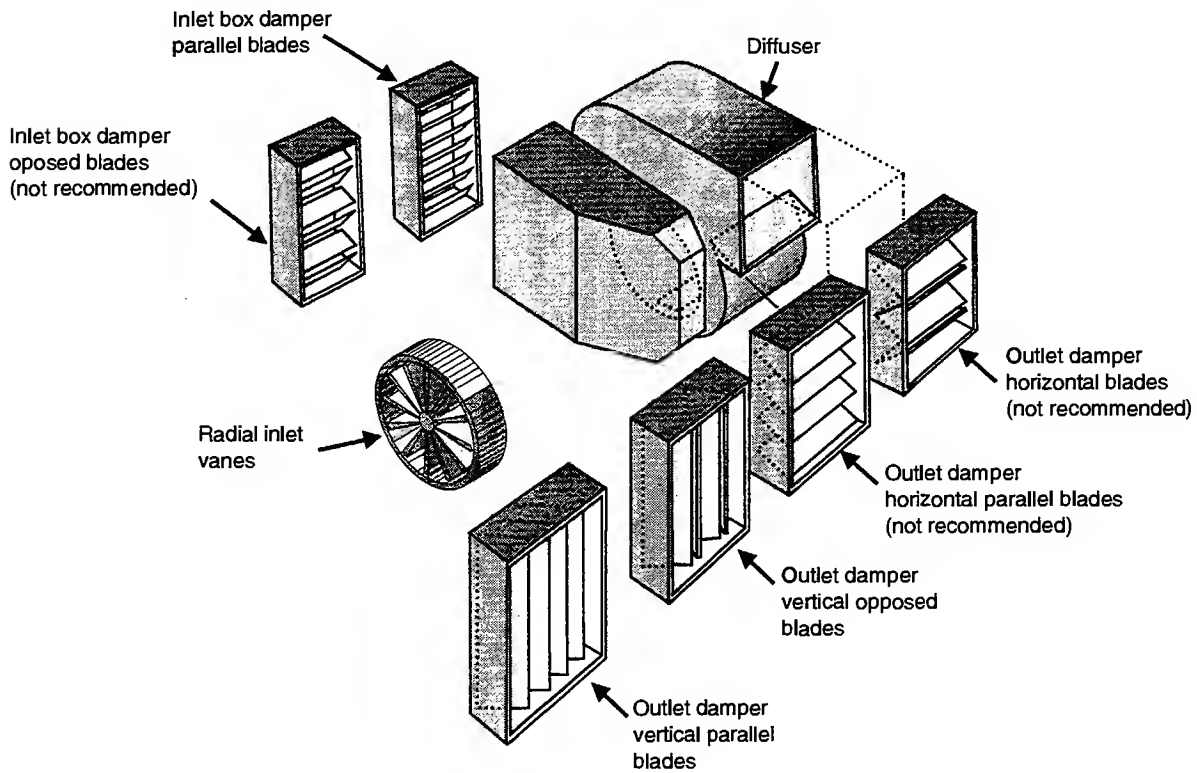
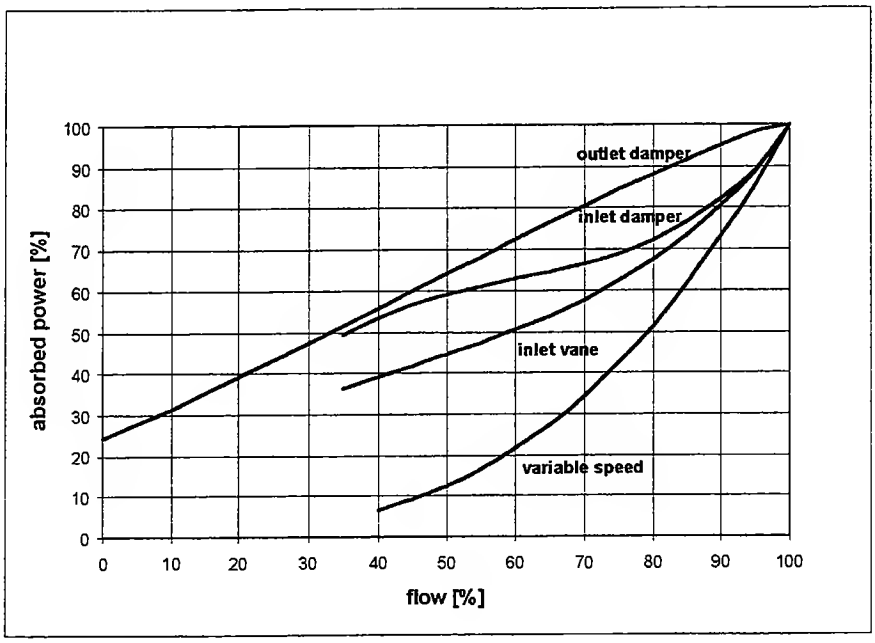


Figure 3.2 shows how fan power consumption is affected by the different flow regulation methods.

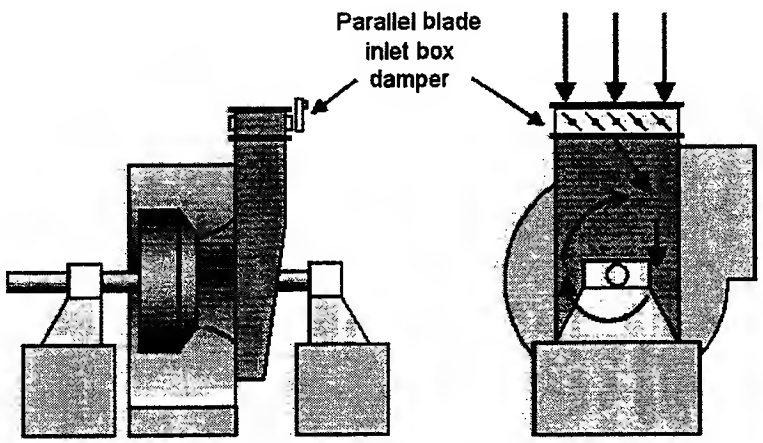
Fig. 3.2: Comparison: Types of Flow Regulation



3.1 Damper control

A parallel blade inlet damper is preferred over either outlet damper or an opposed-blade inlet damper. The parallel-blade inlet damper pre-spins the incoming air in the direction of wheel rotation, resulting in lower energy consumption in the regulation range of 100 - 80% of maximum flow. Fig. 3.3 shows the parallel inlet damper arrangement.

Fig. 3.3: Parallel inlet box damper



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The inlet box damper influences the fan curve (see Fig. 3.4).

Fig. 3.4: Flow regulation by inlet box damper

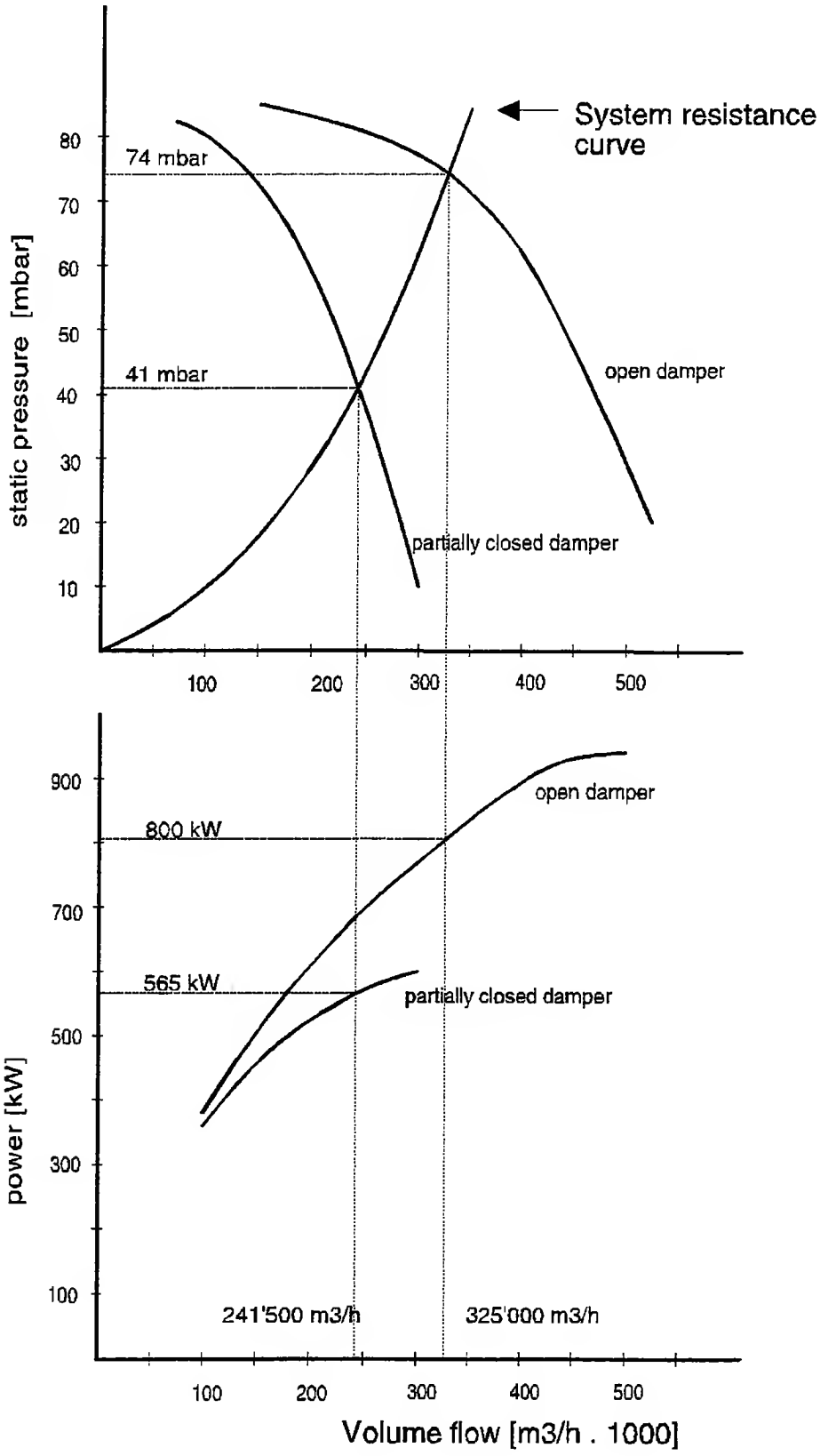
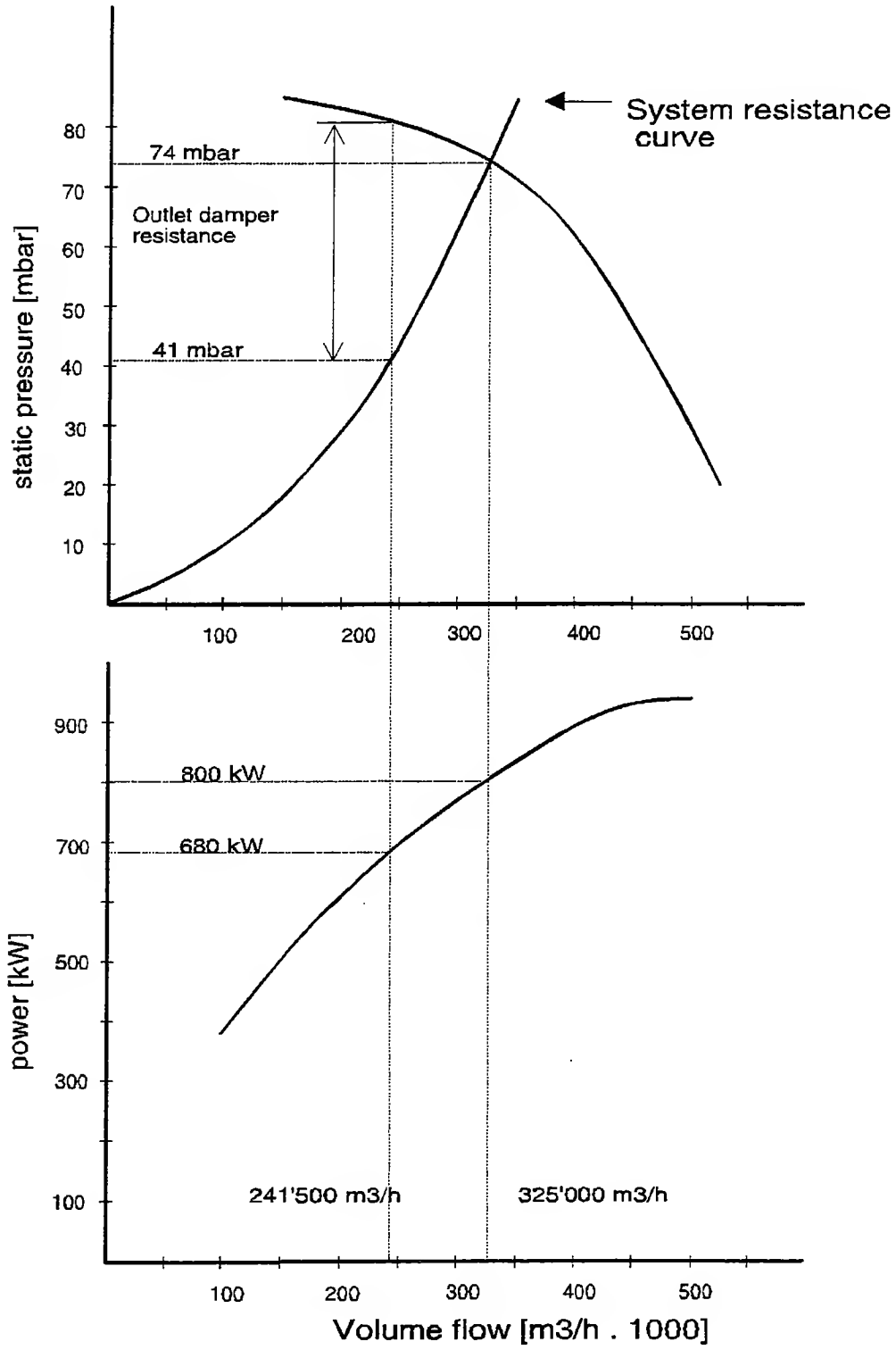


Fig. 3.5 shows the operating point at the intersection of the system resistance curve and the fan curve. By reducing the airflow, the fan theoretically produces a pressure above 80 mbar. Since the system behaves like the system resistance curve, the damper induces a pressure loss of about 40 mbar.

Fig. 3.5: Flow regulation by outlet damper



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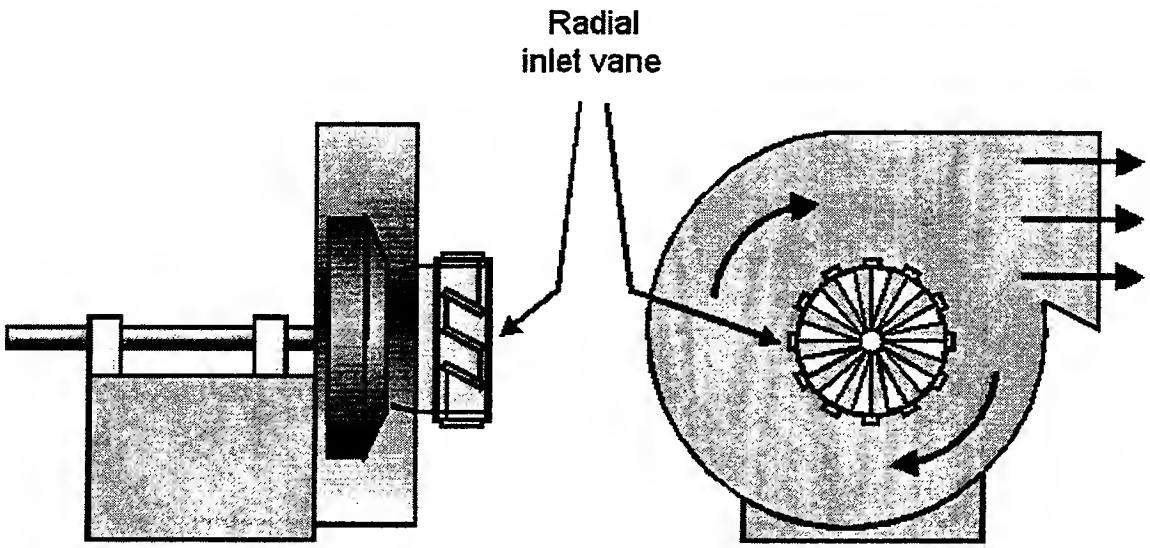
3.2 Radial inlet vane

The radial inlet vane mounted direct at the fan inlet pre-spins the incoming air still better in the direction of wheel rotation, resulting in a wider range of stable regulation (100 - 70%) and less energy consumption. Fig. 3.6 shows the parallel inlet damper arrangement.

Fig 3.6: Radial inlet vane



Their use is mainly recommended in connection with over hung arrangement fans with low rates of dust, thus limited to applications after filters or in clean air, e.g. for cooler under-grate fans.

This arrangement is normally more costly.

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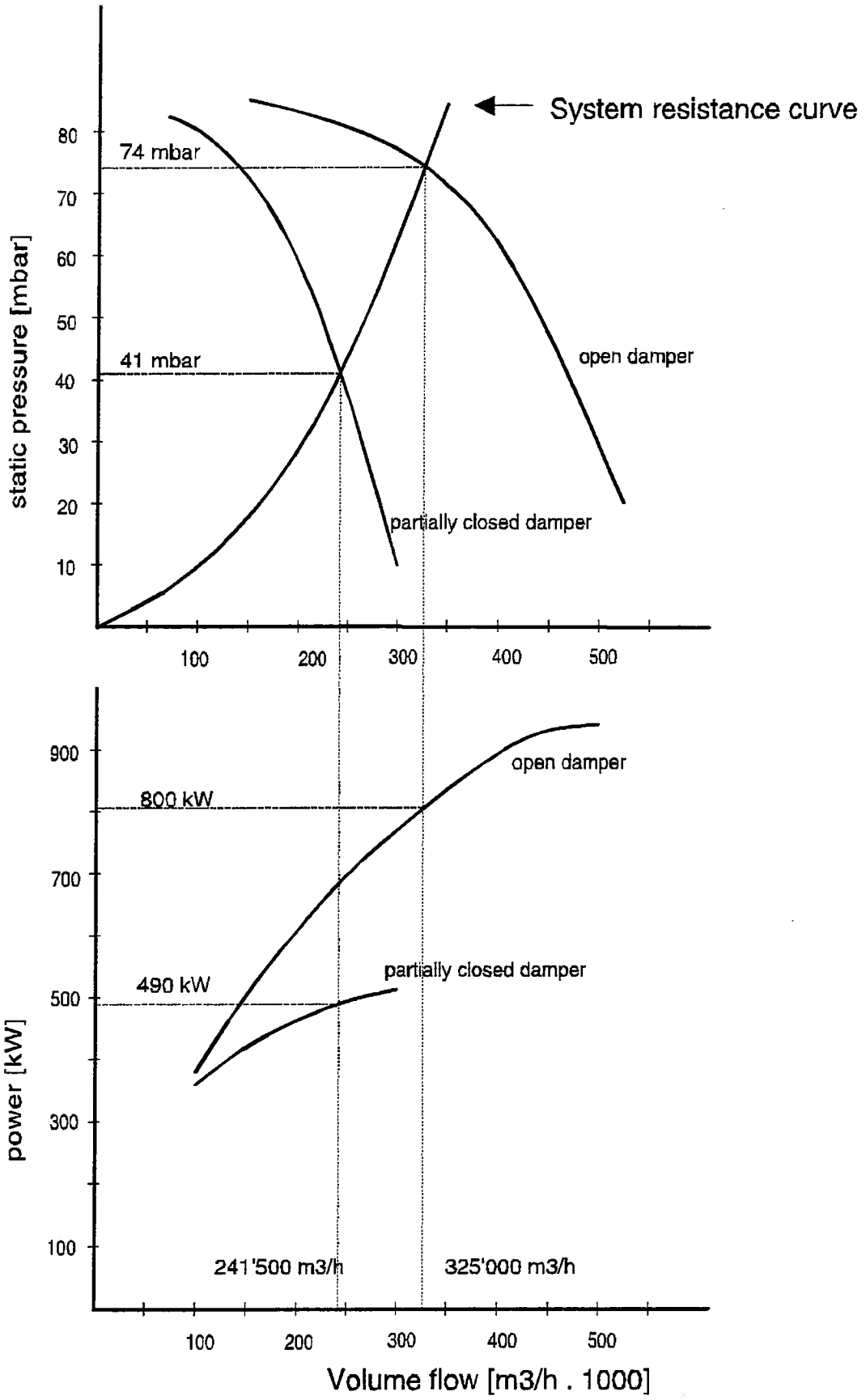
Fig. 3.7: Flow regulation by inlet vane damper



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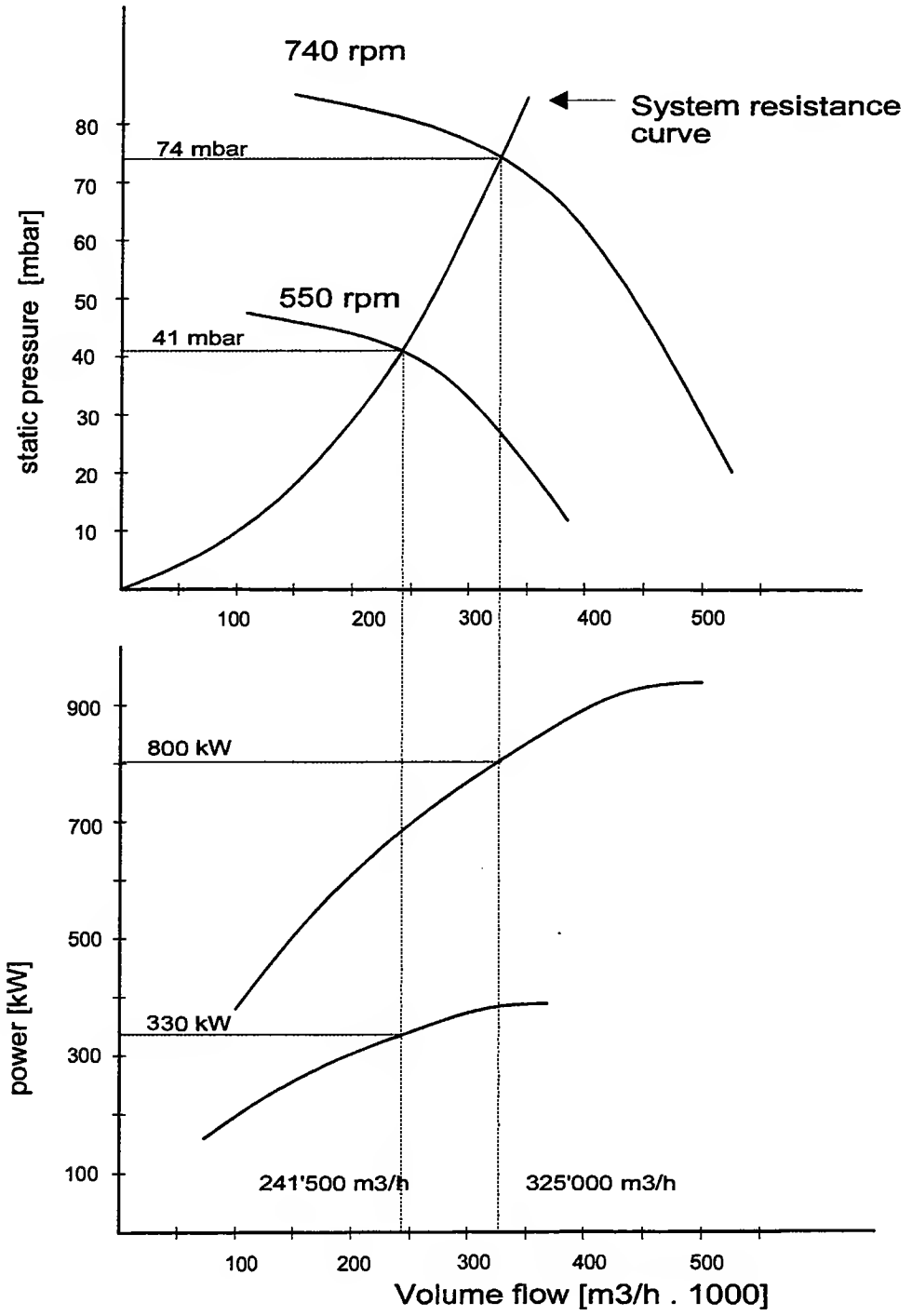
3.3 Speed control

Variable speed controlled fans have more fan characteristic curves but only one characteristic curve of the duct system (see Fig. 3.8).

It is obvious that the variable speed drive is the most efficient type of fan control since no additional resistance for e.g. damper is built into the system. On the other hand a variable speed control is more capital intensive than a damper control.

Fig. 3.8: Flow regulation by speed control



3.3.1 Hydraulic transmission with fixed speed motor

Hydraulic transmission in connection with a fixed speed motor can be an option for speed ranges from 100% down to 85% of maximum speed, but the energy efficiency becomes low when the flow is further reduced.

3.3.2 Speed-controlled electric motors

Flow control by variation of the fan speed is most efficient with regard to energy savings and permits also the reduction of wear on the fan wheel.

- DC motors have limitations in high speed and power (roughly 1000 rpm for 1500 kW, 600 to 800 rpm for 2000 kW motor) and require a lot of maintenance work (motor ventilation circuits, carbon brushes etc.), so their use is not recommended any longer.
- AC motors with slip recovery. These motors have a limited speed range down from 100 - 30%. Its cost increases with the width of the range.
- Synchronous motors with AC variable frequency control

Both types of AC motors are well suited for high power (500 to 5000 kW).

Fan equations at variable speed:

Volume flow: $$\dot{V}_2 = \dot{V}_1 \cdot \left(\frac{n_2}{n_1}\right)$$

Static pressure: $$p_{stat2} = p_{stat1} \cdot \left(\frac{n_2}{n_1}\right)^2$$

Power requirement: $$N_2 = N_1 \cdot \left(\frac{n_2}{n_1}\right)^3$$

(for $\eta_1 = \eta_2$)

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4. POSSIBLE PROBLEMS WITH FANS

4.1 Vibrations

Main reasons for fan vibrations are build-ups of process material (refer to section 4.2) and wear/erosion on the fan impeller (refer to section 4.3/0). A moderate level of vibration can be tolerated from a mechanical design point of view, although it certainly reduces the bearing lifetime to a certain extent. Therefore it should be tolerated only to avoid additional main equipment downtime.

If vibrations are excessive, fan impeller balancing, cleaning or replacement is required. For balancing special vibration detectors are used on site or the wheel has to be shop balanced.

General Machinery vibration severity as per ANSI S2.41 (Fig. 4.1 a) for use as a guide in judging vibration as a warning of impending trouble)

1. Rigid support
   The fundamental natural frequency of the machine/support system is **higher** than the operating speed

excellent	0.	to	2.54	mm/s vibration velocity (Peak)
good	2.55	to	6.35	mm/s
alarm	6.36	to	12.7	mm/s
shutdown	>		12.7	mm/s

2. Flexible support
   The fundamental natural frequency of the machine/support system is **lower** than the operating speed

excellent	0.	to	3.81	mm/s vibration velocity (Peak)
good	3.81	to	10.16	mm/s
alarm	10.17	to	19.1	mm/s
shutdown	>		19.1	mm/s

Vibration severity criteria (10 Hz to 1 kHz) per ISO 2372 (Fig. 4.1 b)

1. Large machines with rigid foundations whose natural frequency exceeds machine speed

good	0.0	to	2.54	mm/s vibration velocity (Peak)
allowable	2.55	to	6.35	mm/s
just tolerable	6.36	to	15.84	mm/s
not permissible		>	15.84	mm/s

2. Large machines operating at speeds above foundation natural frequency

good	0	to	4	mm/s vibration velocity (Peak)
allowable	4	to	10	mm/s
just tolerable	10	to	25.4	mm/s
not permissible		>	25.4	mm/s

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If the tolerable vibration levels are exceeded, the fan must be shut down, which usually results in costly production losses. To extend the periods between shutdowns due to vibration, the use of automatic balancing devices, mounted on the fan shaft may be considered. Depending on their size, they are capable of automatic compensation of a certain rotor unbalance. For manufacturers of such devices, refer to the information source at the end of this report.

Fig. 4.1 a: Vibration severity chart (ANSI S2.41)

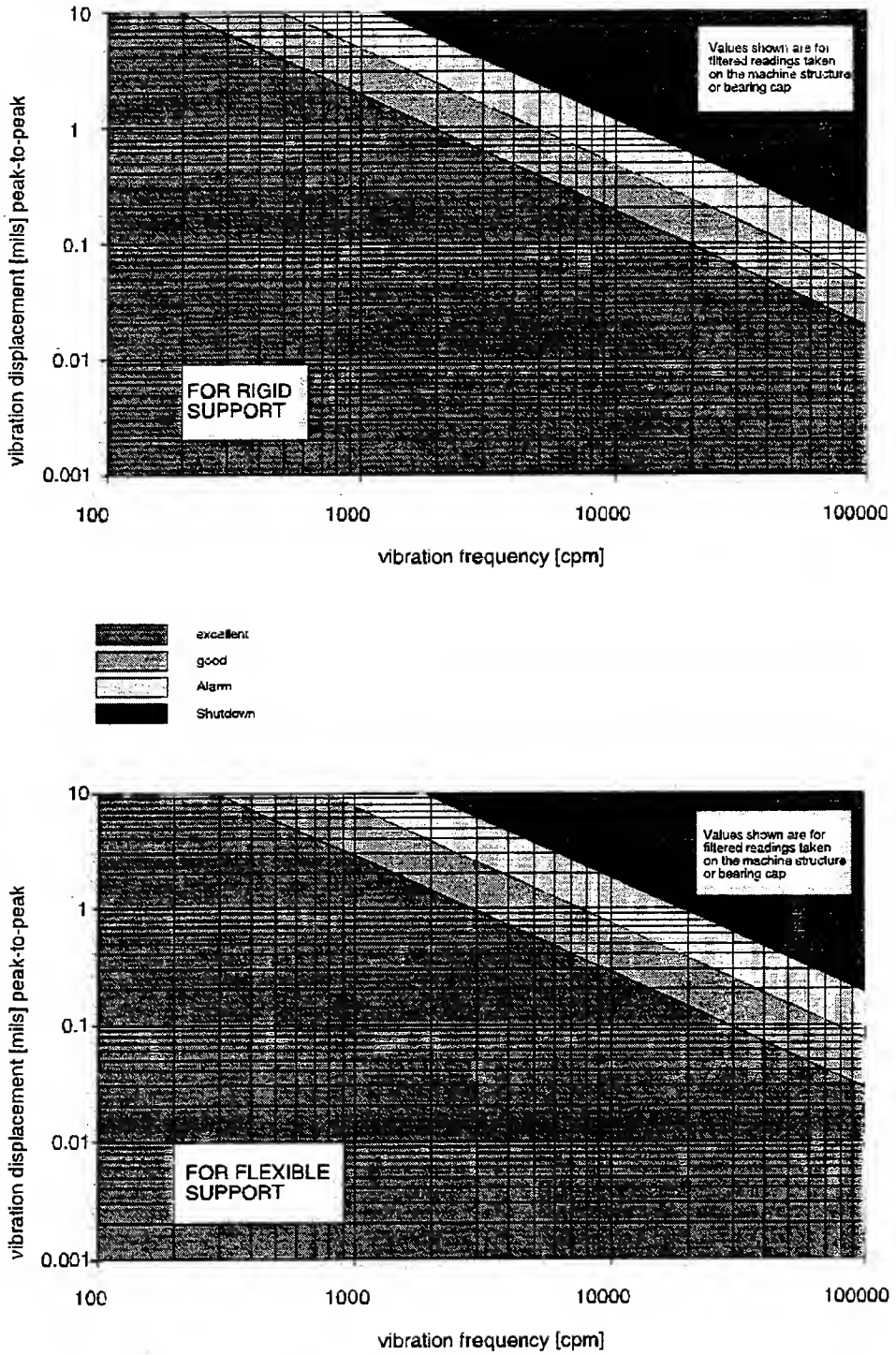

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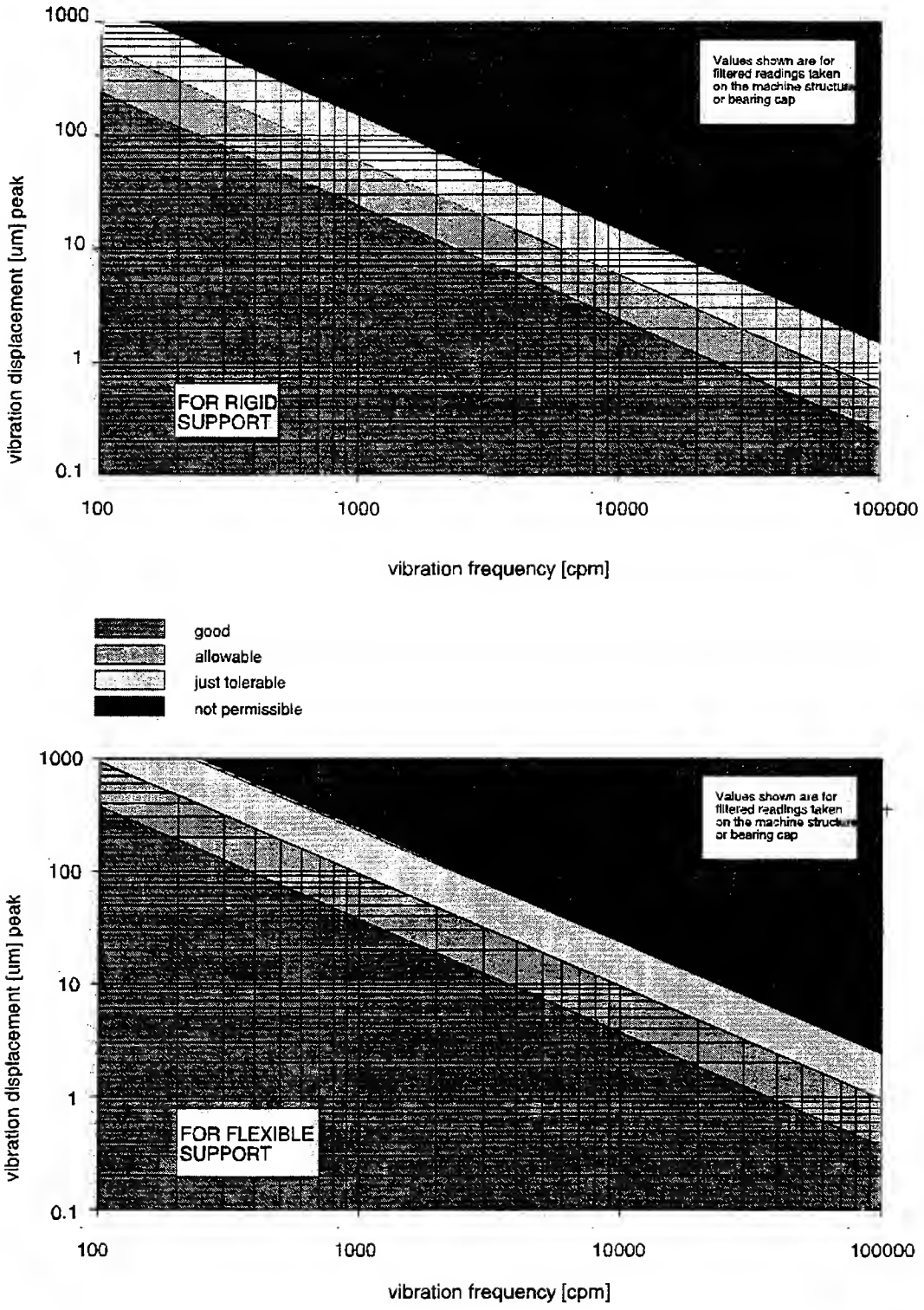
Fig. 4.1 b: Vibration severity chart (ISO 2372)



4.1.1 Variable speed operation

All electrical variable speed drive systems can generate harmful harmonics that result in torque pulsation. Such harmonics can be predicted and filtered, but often at high cost. For fans with variable speed control it must be verified by the motor supplier that the pulsating torque do not cause excessive vibrations.

4.1.2 Thermal effects

Some typical problem areas are:

- Due to expansion joint problems, forces due to thermal expansion of ducts are transmitted to the fan housing, resulting in damage of the housing or interference between wheel and housing. It can also cause excessive force on foundation bolts that sometimes can result in foundation cracks.
- Rapid temperature changes in a system require proper design of the wheel-to-shaft fit. This will assure that looseness and resulting vibration sensitivity will be avoided.

4.1.3 Hot shutdowns

Thermal shaft set is a usual concern on centre-hung fans above 120°C when shut down in the hot condition. It is generally agreed that this thermal bowing of the shaft occurs due to uneven thermal gradients across the wheel and shaft assembly at hot shutdown. This results in small asymmetrical distortions, often sufficient to cause excessive unbalance forces during start-up.

In some cases the resulting vibrations are within acceptable limits. If so, it is usually found that the unbalance forces disappear after 12 to 36 hours of operation.

The preferred, but expensive solution is to have an auxiliary drive to slowly rotate the wheel and shaft assembly, which should be engaged immediately after hot shut-down to avoid the undesirable thermal distortion. Auxiliary drives are typically designed to maintain a minimum speed (40 to 60 rpm) as the fan slows down. They are not intended for use in starting the fan rotor from a dead stop.

Most hot gas fans work well without auxiliary drive, since hot shutdowns are infrequent and of short duration and the natural draft of the chimney keeps the rotor in slow motion for some time.

The tendency is to install hot gas fans without auxiliary drives. Often it is also the case that variable speed drives have a turndown ratio of 10:1, which corresponds normally to less than 70 rpm.

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4.2 Material build-up

A well maintained fan is generally a reliable piece of equipment. However, in case of material build-ups within the fan, serious operational limitations could occur, such as vibrations.

Possible reasons for material build-ups are:

- Mineralogical composition of the material
- Burning conditions
- Duct/fan arrangement
- High temperatures
- Inappropriate blade shape and angles
- High fan speed
- Dew point problems

After balancing, the rotor of a fan will still have a certain residual imbalance, the value of which depends on the balance quality grade. The unbalance force can be calculated with the formula:

$$F = m \cdot \omega^2 \cdot e_{per} \qquad [N]$$

m	rotor mass	[kg]
ω	angular velocity $2\pi n/60$	[s^{-1}]
n	rotor speed	[rpm]
e_{per}	permissible residual specific unbalance	[m]

Industrial fans often handle dust-laden gases and thus dust deposition on the impeller may occur, increasing the initial unbalance. Asymmetrical wear on the impeller has the same effect. These additional unbalances can be considerable compared to the permissible unbalance according to the balance quality grade. If the resulting vibration exceeds the tolerable limits, the fan has to be shut down to clean the wheel. See Fig. 4.1 c for permissible unbalance.

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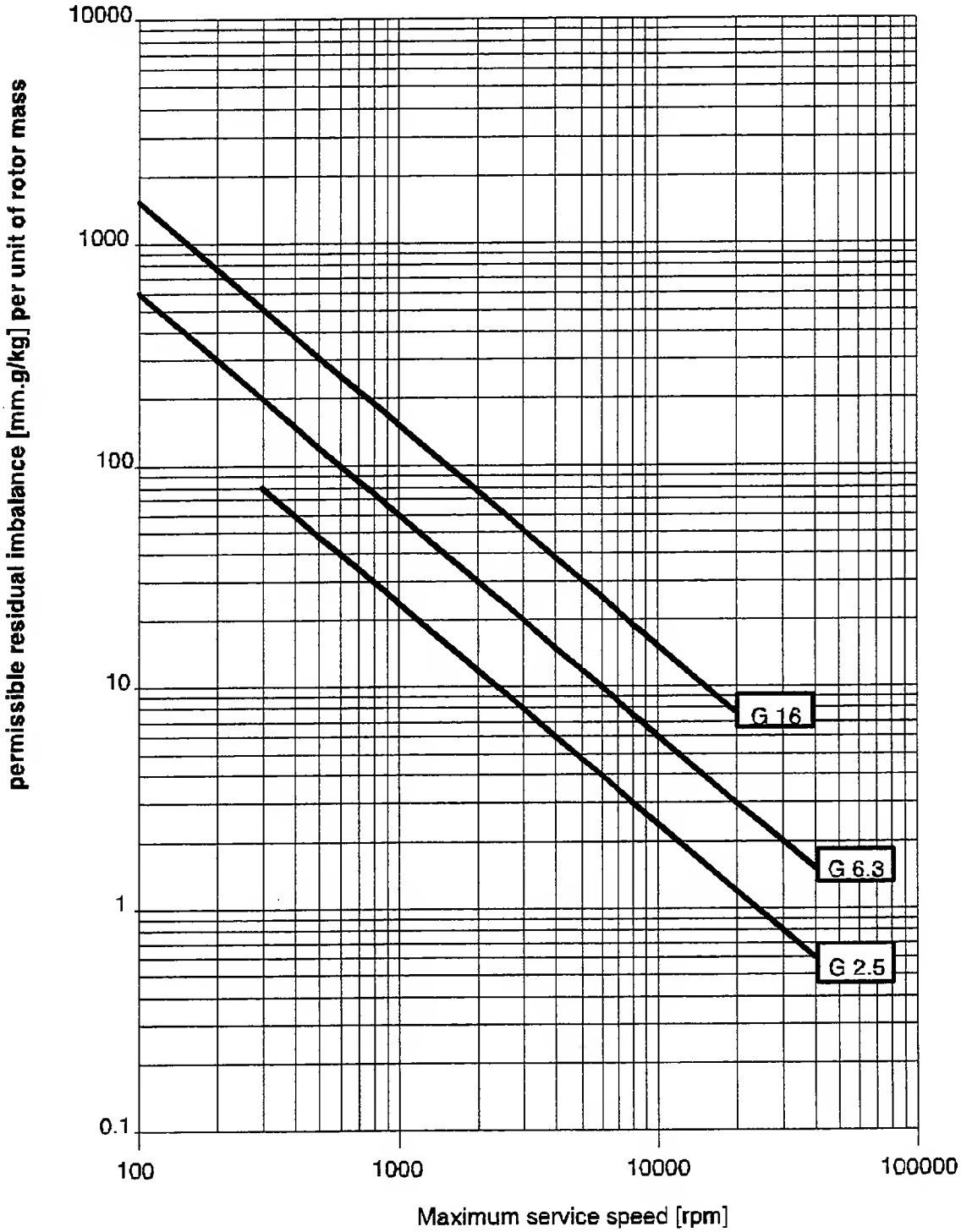
Fig. 4.1 c: Balance quality grades



G16 Drive shafts with special requirements
Parts of crushing machines
G6.3 Fans
G2.5 Fans with special requirements
rigid turbo-generator rotors

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4.2.1 Kiln exhaust fan build-up

One idea about the causes of build-up is that some particles are "sticky" at temperatures above 300°C and begin to build up on the rotor surfaces. The impact energy of the particles striking the rotor surface (especially at an angle of 90°) is also converted to heat and results in additional softening. Other particles with a higher softening point are caught in the sticky material and increase the coating layer thickness. The originally soft build-up gets harder under the influence of heat and pressure (from centrifugal force and the impact of other particles).

4.2.2 Recommendations against build-up

- The fan rotor should be designed for the smoothest possible flow lines to reduce the impact energy of dust particles. Backward curved and airfoil are the best blade forms. Airfoil blades must be designed carefully to prevent material from getting inside and regular inspection of the fan blades is mandatory.
  Backward curved blades must be inclined enough to prevent the "hard" build-up on the front surface and radial enough to prevent the "soft" build-up on the back-surface.
- Fans should be designed for low gas and particle velocity at the fan inlet. This reduces the impact energy of particles against the rotor and can be achieved by
  1) Double inlet instead of single inlet fans
  2) Larger diameter / lower speed fans
  3) If possible, the peripheral speed at the rotor inlet opening should be limited to about 76 m/s, and wheel inlet velocity should not exceed 38 m/s
  4) The fan shaft should be oversized to reduce unbalance effects.

The design critical speed (considering a bearing oil film thickness and a build-up thickness of 25 mm on all leading surfaces of the rotor blades) should be at least 1.25 times the operating speed of the fan.

Suppliers generally do not recommend to spray water directly onto a hot fan wheel, but in practice it has been done over years with success. The rapid cooling effect removes build-up effectively, however, the strength properties of the wheel material may be affected adversely if the injection rate is too high.

Several different techniques are used to clean the fan, none with guarantied success however:

* "Sand blasting" by addition of sand to the gas flow for short periods (Fig. 4.2)
* Acoustic resonance (Fig. 4.3)
* Compressed air injection (Fig. 4.4)
* Steam injection (Fig. 4.4)

The following measures are performed below operational fan speed or even require a fan stop.

* "Knocking" with pneumatic hammer
* High pressure water spray (up to 300 bar)
* H_2O injection (Fig. 4.4) (rather delicate; it has to be carefully dealt with)

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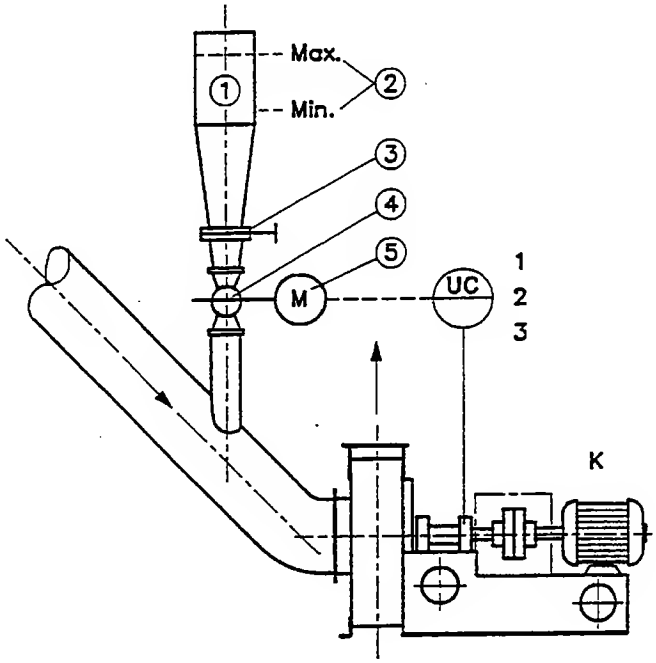
Fig. 4.2: Sand cleaning device

UC = Vibration control unit
(Schenck Vibrocontrol 1'000 C)
1 = Normal range
2 = Cleaning range
3 = Hihi alarm



① = Sand bin ~ m^3 (grain size 100–200 µm)
② = Level indicators
③ = Shut–off gate
④ = Rotary valve (event. double) ø 350 mm
⑤ = Variable speed motor ~ 1,5 kW

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Fig. 4.3: Acoustic cleaning device

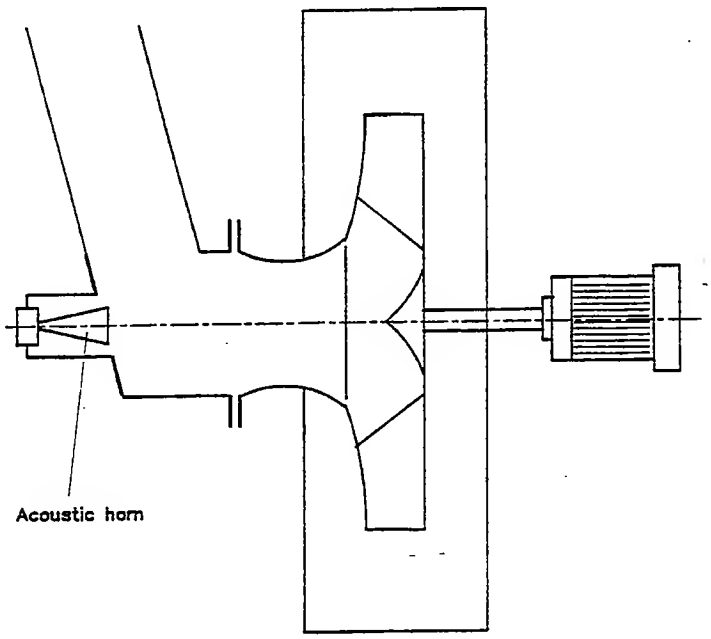
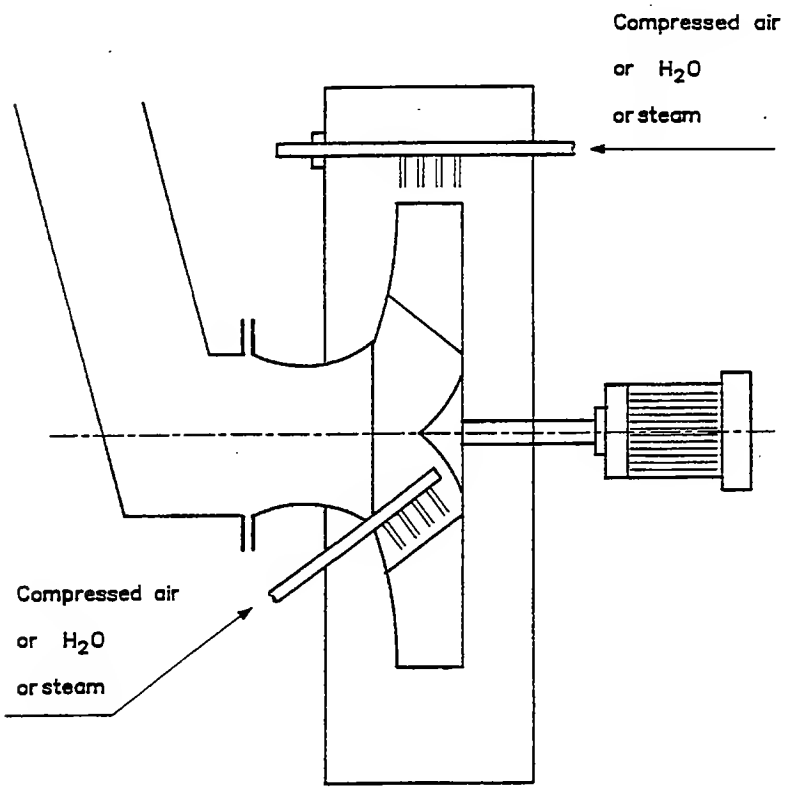


Fig. 4.4: Compressed air, H_2O or steam Cleaning Devices



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4.3 Erosion

4.3.1 Erosion types

It has been established that maximum erosion occurs when the angle of incidence between particle and surface is between 20 and 40 degrees. This erosion, referred to as "ductile", is thought to be the consequence of microscopic melting, which occurs when sharp-edged, hard particles scratch the surface.

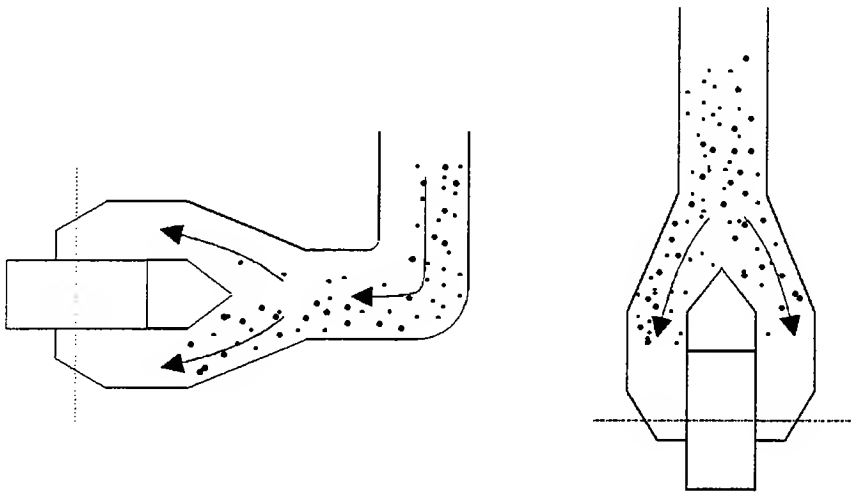
Particles striking at an angle of about 90 degrees to the surface erode according to another mechanism. The kinetic energy of the particle creates a stress at the contact surface that can exceed the elastic limit of the material, thus forming a surface crack. This type of erosion is called "fragile".

4.3.2 Improper duct connection

Figure 4.5a shows a duct arrangement encountered on a raw mill system. Due to the uneven material distribution one side of the double-inlet impeller wore out much faster.

Figure 4.5b shows the recommended modification.

Fig. 4.5a, Fig. 4.5b



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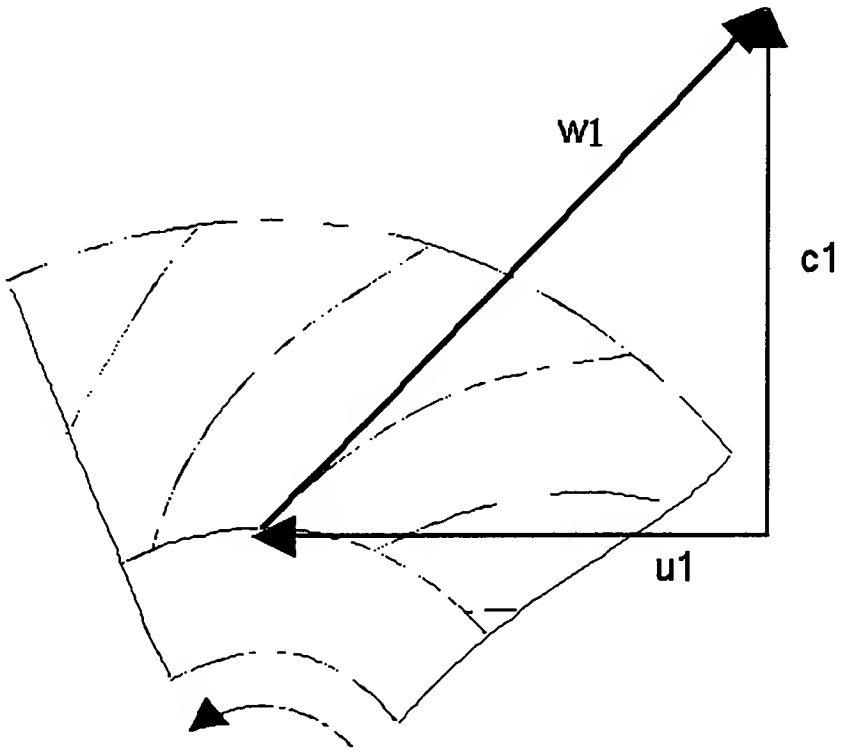
4.3.3 Effects of impeller speed and wheel inlet velocity

As mentioned earlier, erosion is proportional to the square of wheel inlet velocity and to the second to third power of the relative gas velocity w1 at the rotor inlet. With a given rotor size this velocity is proportional to rotor speed [rpm].

As a rough guideline we can expect that the erosion will vary with the speed to the power of 3. A large fan will therefore resist erosion better then a smaller one, operating in the same conditions, due to lower fan speed, larger surfaces.



4.4 Wear protection

4.4.1 Protection of parts subjected to abrasion

4.4.1.1 Direct protection

- Increased thickness of wear parts

  This should only be done when abrasion is very light (ID behind electrostatic precipitators)

- Direct hard surfacing on impeller by deposits by electrode or fusion projection

  This process is very efficient if the deposits are well chosen and properly applied. However, the thickness of the deposit is limited. The base structure of the impeller can be affected (dilution of hard surfacing and significant addition of energy) and the mechanical characteristics of the steel can deteriorate, especially after several maintenance operations.

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- Glued-on ceramics

  Their hardness is very high and they are very resistant to ductile abrasion

  Temperature must be limited and the ceramics tend to come off at the blade end (high centrifugal forces). Application on static parts is easier and more reliable.

4.4.1.2 Protection by wear plates

- Corners or mild steel plates (or steel of the same grade as the base structure) added on to the blade and on the center plate. This protection is applicable only where abrasion is low. If the protection area is not wide enough, the base plate can also be attacked.
- Wear-plates in abrasion resistant steel. This process is applied particularly on flat-bladed impellers. The wear-plates are fitted on with countersunk screws. Replacement is easy, but the solution can be insufficient if abrasion is very high.
- Mild steel wear plates with hard surfacing by electrode or hardsurface spray coating. This solution has the double advantage of easy replacement and high resistance to abrasion when the type of deposit is well chosen. Moreover, hard surfacing on site is easily carried out with no risk for the base structure.

4.4.2 Deflection of abrasive particles

4.4.2.1 Deflector plates on impeller (Fig. 4.6)

Fig. 4.6a shows the fan inlet and the rotating impeller. A, B and C are flow lines for the gas and lines 1, 2 and 3 represent the trajectories for particles of varying size. Line 1 refers to a very small particle that closely follows the gas flow line. With increasing particle size the trajectories 2 and 3 deviate from the gas flow lines. The particles hit the back-plate of the impeller and erode it.

Fig. 4.6b depicts the passage of two different particle sizes between two blades. Line 1 is the trajectory of a small particle, line 2 of a coarser one. Most particles will hit the blade during their passage through the impeller.

With suitably located deflector plates (Fig. 4.6c), the particle trajectories can be altered so that no particles will hit the blades. The location and direction of the deflectors depends on the particle size distribution, density of gas and particle, and fan speed and size.

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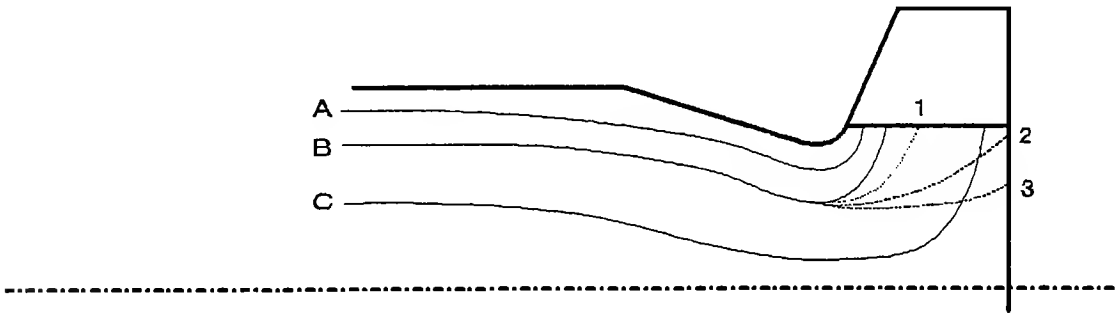
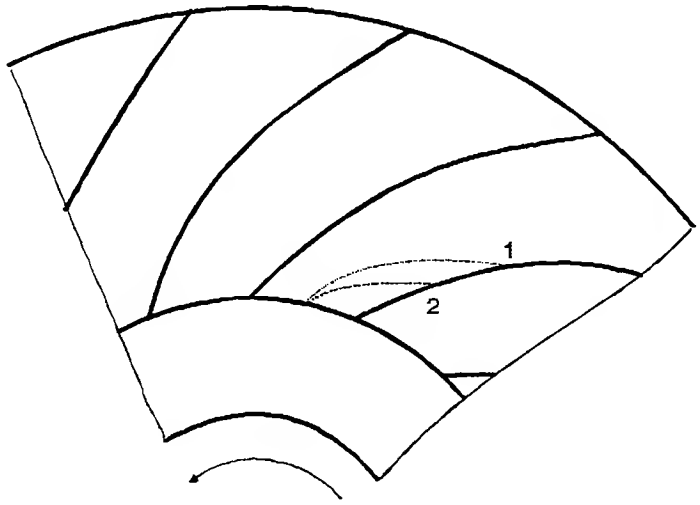
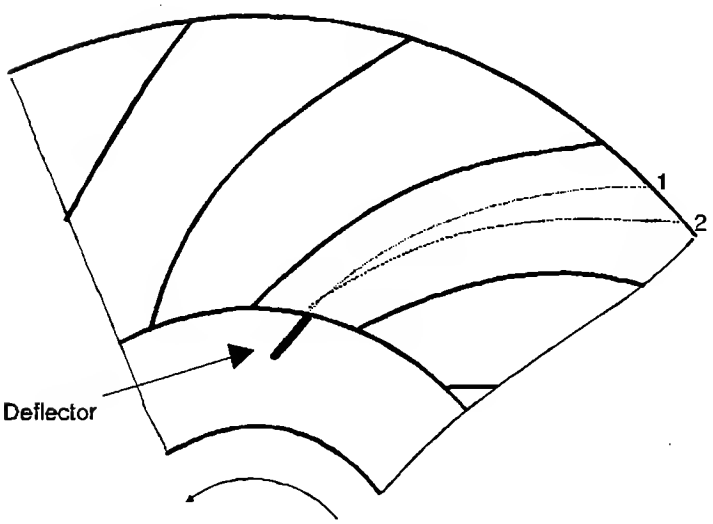
Fig. 4.6: Wear protection by particle deflection







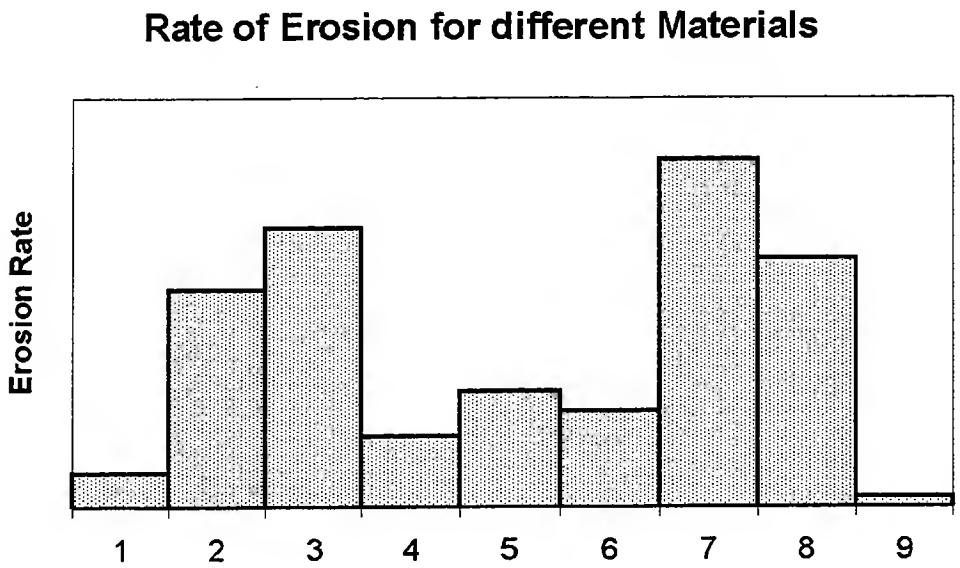
4.4.3 Liner materials

Due to the need for ductility in the structural members of the wheel, the wheel itself is usually not capable of high resistance to erosion. It is therefore necessary to install liners with a higher hardness classification to provide sufficient protection against wear.

Material hardness is an indication of its resistance to erosion. A very good liner material is chromium carbide with an average hardness of 600 Brinell.

Figure 4.7 shows erosion test results of nine different materials.

Fig. 4.7: Erosion Test on Fan (Solyvent-Ventec)



1) Alloy of tungsten and nickel-chromium. Flame spray coating.
2) Alloy of nickel, chromium and cobalt. Flame spray coating followed by fusion.
3) Alloy of tungsten, cobalt carbides and nickel-chromium. Flame spray coating followed by fusion.
4) Special chromium cast iron. Special electric arc welding.
5) Chromium cast iron with chromium carbides. Electric arc welding.
6) Chromium cast iron. Semi-automatic electric arc welding.
7) Ceramic powder containing basically aluminium oxide. Flame spray coating.
8) Same as 7, but of different hardness.
9) Ceramic tiles containing basically aluminium oxide. Glued to the blades.

Carbon steel sheets with a protective layer of chromium carbide are normally available in standard sizes and various thicknesses. Cutting discs or cutting plasma can cut them to the necessary form and size.

The liner fixation can be done by bolting or by welding the carbon steel base plate to the existing structure of the wheel or casing.

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4.5 Bearings

Bearing problems are usually caused by operating conditions and not by deficiencies in the bearings. By following regular operating and maintenance procedures, many bearing problems will be avoided.
Most problems are due to hot bearings. For hints see section 5.

5. FAN CAPACITY ADJUSTMENT

Fan capacity adjustments in the field are practically limited to the speed adjustment and this within a limited range: ~ 0 - 15 % for flow increase and ~ 0 - 30 % for flow reduction.

Other capacity adjustments require mainly dimensional modifications of the fan wheel, which have to be carried out in the workshop. Those are more costly and critical to execute.

5.1 Fan capacity too low

Important variables that are hampered by insufficient gas flow are e.g.

- kiln production
- separator efficiency
- mill performance (throughput, drying capacity, mill venting)

Possible remedies:

- Design changes in the system (reduce false air, reduce pressure drop) the fan consumes no additional energy!
- reduce/rectify false air inleaks
- reduce unnecessary pressure drops caused by e.g.
  - not fully open dampers
  - improper design of inlet/outlet connections
  - no turning vanes in bends (refer to section 8.3)
  - dust deposits in ducts and bends
  - too narrow ducts (appropriate air speed must however be maintained to avoid settling of dust)
- Changes in fan design, speed
  - Speed increase
    - volume flow is directly proportional to fan speed
    - fan absorbed power rises with the third power of fan speed
    - fan works with lower than design efficiency
    - normally only possible with V-belt drives
    - sound emission rises with fifth power of speed
    - higher wear, if fan handles dust-laden gas
    - increased sensitivity to rotor imbalance caused by dust deposits on blades
    - speed increase is limited by mechanical strength of rotor
    - speed increase is limited by fan critical and resonant speed

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- Fan wheel diameter increase
  - practically limited to wheels where the blades do not extend to the full diameter of the rotor sideplates (plates can then be welded-in to increase the effective fan wheel diameter)
  - volume flow rises with the third power of wheel diameter
  - fan absorbed power rises with the fifth power of wheel diameter
- Replacement of rotor with inherent low efficiency by one designed for high efficiency
  - e.g. replacement of radial-blade wheel by backward-inclined or backward-curved blade wheel.
  - limitations with regard to diameter, width (and speed, if wheel is directly coupled to motor via flexible coupling) of the new wheel exist if fan housing and motor are to be kept.

◆ New fan
Can be designed for optimum performance at the new operating point

5.2 Fan capacity too high

May be due to e.g. oversized fan, decline in production, process changes

Possible remedies:

◆ Flow reduction by damper
Widely used solution, but inefficient and expensive (fan energy consumption)

◆ Flow reduction by inlet vane damper
More energy-efficient than damper, but can be recommended only for flow regulation purposes, not for permanent use at lower capacity (expensive, may be difficult to fit into existing equipment)

◆ Changes in fan design, speed
- Lower speed
  - volume flow decreases proportional to fan speed
  - absorbed power decreases with third power of fan speed
  - lower sound emission
  - lower wear, if handling dust-laden gas
  - lower sensitivity to rotor imbalance caused by e.g. dust deposits on blades
- Rotor width reduction
  - applicable if fan must deliver the same pressure as before, but at lower volume flow
- Rotor diameter / width reduction
  - volume flow decreases with the third power of rotor diameter
  - volume flow decreases proportional to rotor width
  - fan delivery pressure decreases with the square of rotor diameter
  - absorbed power decreases with the fifth power of rotor diameter
  - absorbed power decreases proportional to width
  - lower sound emission

  To maintain a good efficiency, it is not sufficient to decrease the diameter only, but also to adapt the rotor width.
  For a permanent capacity reduction this is a good solution

If the old transmission is kept, speed reduction is normally reversible, i.e. fan capacity can be increased again if necessary, whereas a reduction in rotor diameter / width are not.

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6. TROUBLESHOOTING

Problem	Check for
Noise	1. Squealing V-belts, due to misalignment or improper tensioning 2. Defective bearings, or bearing seal rubbing 3. Misalignment of bearing seal 4. Misaligned housing-shaft seal 5. Foreign matter in fan housing 6. Rubbing of shaft seal, wheel to inlet piece, or wheel to housing 7. Heat flinger is contacting guard 8. Coupling failure 9. Untreated expansion joints
Poor performance	1. Incorrect fan rotation 2. Wheel is off-center, poor inlet piece fit-up allows recirculation of air 3. Fan speed too low/high 4. Poor duct design, installation of elbow or turning vanes could remedy problem 5. Inlet damper installed backwards (counter-rotation) 6. System resistance is excessive compared to design requirements (partially closed damper may be the cause) 7. Density may be different from design density
High bearing temperature	1. Defective bearings caused by electrical arc due to improper grounding of nearby welding 2. Over-lubrication 3. Improper lubrication or contaminated lubricant 4. Lack of lubrication, cooling fluid, or circulation 5. High ambient temperatures or direct exposure to sunlight 6. Misalignment 7. Excessive thrust loading 8. High vibration 9. Inadvertently exchanged bearing caps (mismatched) 10. Bearing race turning inside housing 11 Moisture in bearing 12. V-belts too tight 13. Improper location ; not enough room for free axial movement of floating bearing in its housing at elevated temperatures) 14. Heat flinger missing
Excessive starting time	1. Motor improperly sized for fan wheel WR^2 2. Inlet dampers not closed during start-up 3. Properly selected time-delay starter/fusing required (many industrial fans take up to 20 - 25 seconds to reach operating speed) 4. Temperature at inlet is excessively low (high density) 5. Low voltage at motor terminals 6. Inadequate system resistance

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Vibrations	1.	Loose bolts in bearings and pedestals, or improper mounting
	2.	Defective bearings
	3.	Improper alignment of bearings or couplings
	4.	Out-of-balance fan wheel
	5.	Loose set-screws holding wheel to shaft
	6.	Weld cracking
	7.	Improper fan wheel clearance to inlet piece(s)
	8.	Material build-up and/or wear on wheel
	9.	Ensure expansion joints in ductwork are not fully compressed
	10.	Misalignment or loose V-belts
	11.	Improper wheel rotation
	12.	Operation near system critical speed
	13.	Shaft bent or distorted during high-temperature shutdown
	14.	Defective motor
	15.	Resonant frequencies of structural steel mounting
	16.	Beat frequency with other fans on common base
	17.	Loose hub-to-shaft fit
Duct pulsation	1.	Control volume with a radial inlet damper
	2.	Install speed variation
	3.	Change to a special "surgeless" blower design
High motor temperature	1.	Improper ventilation of cooling air to motor (may be blocked by dirt)
	2.	Input power problems (especially low voltage)
	3.	High amperage
	4.	High ambient temperature

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Component	Problem	Probable cause, remedies
Bearings	Noise	1. Imperfection in bearing elements 2. Improper clearance 3. Internal wear of bearing parts
	Freezing water jacket	1. When stopping water flow in freezing weather, blow out lower portion of bearing housing water cavity
Fan Wheel	Erosion	1. Reduce dust loading 2. Reduce rpm 3. Redesign inlet ductwork 4. Damper setting 5. Damper design 6. Better liner material 7. Alternate blade design
	Buildup	1. Reduce dust loading 2. May be affected by system temperature change 3. Blade form 4. Alternate wheel material, apply "slippery" material
	Vibration	1. Rectify build-up 2. Rectify erosion 3. Tighten foundation bolts 4. Correct misalignments 5. Improve supporting structure 6. Check effects of ductwork thermal expansion 7. Shaft bow due to "thermal set" 8. Special considerations, refer to factory
Shaft	Cracks at section change	- Get new shaft that is machined to eliminate stress raisers
	Natural frequency too close to running value	- Redesign shaft
	Out of round at bearing	- Replace shaft
	Bowing and torsion problems	- Refer to factory
	Shaft dropped or damaged during transit or installation	- Get new shaft
Hubs	Loose fit on shaft	- Tighten interference fit
	Insufficient stiffness	- Redesign
	Cracks in casting	- Change to weldment
	Erosion	- Build up or replace

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7. START-UP OF FANS

Before starting the fan for the first time, complete the following list:

1) Uncouple motor from fan and check motor (fan) for proper rotation.
2) Shut off power by disconnecting motor main breaker.
3) Check and tighten hold-down bolts.
4) Check and tighten rotor set-screws.
5) Check couplings and bearing for proper alignment.
6) Move rotor to see if it is rotating freely and maintains proper inlet piece/rotor clearance.
7) Check that fan wheel is balanced.
8) Check fan and ducts for any foreign material or dirt build-up.
9) Check that physical position of damper corresponds to indication at actuator and control panel.
10) Secure all access doors.
11) Check lubrication of bearings, couplings, drive unit etc.
12) Couple the motor again to fan and secure and check safety guards for clearance.
13) Close dampers for adequate system resistance to prevent drive unit from overloading.
14) Supply water to water-cooled bearings and start lubrication pump.
15) Make sure that all persons are away from fan and out of any other equipment of the system to which the fan is connected.
16) Connect electric motor by closing main circuit breaker of the motor. Start equipment according to recommendations of drive unit and starting equipment supplier.
17) Allow fan to reach full speed, then shut down. Make immediate corrections if any vibrations or unusual sounds have been detected.
18) During a run-in period make observations of bearings at least once an hour. Higher bearing temperatures may occur if bearings are over-lubricated.
19) Refer to trouble-shooting guide for any unusual occurrences encountered during the run-in period. Only after any vibrations, misalignments, etc. have been corrected, may the fan be restarted.

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8. FAN IMPELLER ARRANGEMENT AND CONNECTIONS

8.1 Assembly

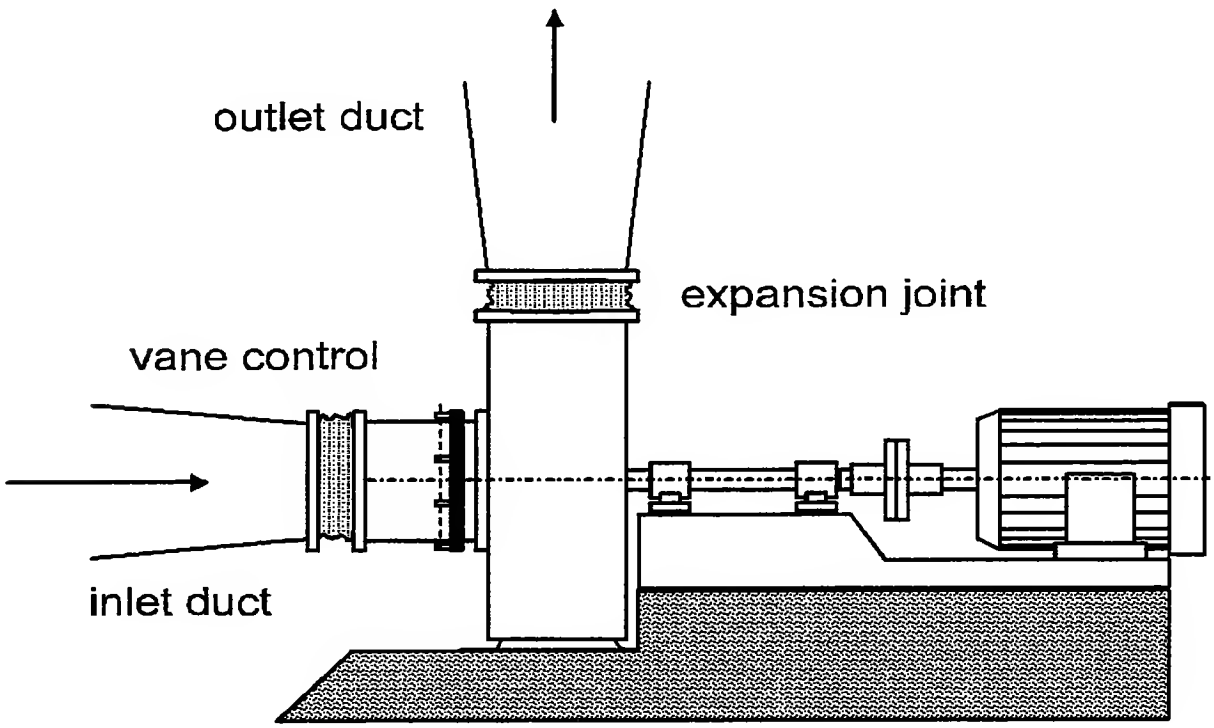
8.1.1 Overhung assembly (Fig. 8.1)

This is possible if the fan operates in an environment with little risk of clogging and associated imbalance and the impeller diameter is not too large (less than approx. 2.7 m).

Advantages:

- lower purchase and installation costs than centre hung assembly
- connection of upstream duct directly to fan inlet is possible (no inlet box)

Fig. 8.1: Fan arrangement - overhung arrangement



8.1.2 Center hung assembly (Fig. 8.2)

This assembly is more robust and absorbs the dynamic effects of rotor imbalance better than the above solution.

Its use is recommended if

- the dust load is high with a risk of clogging or wear of the impeller, causing rotor imbalance
- when the size and weight of the impeller makes overhung assembly delicate

Fig. 8.2: Fan arrangement - arrangement between bearings

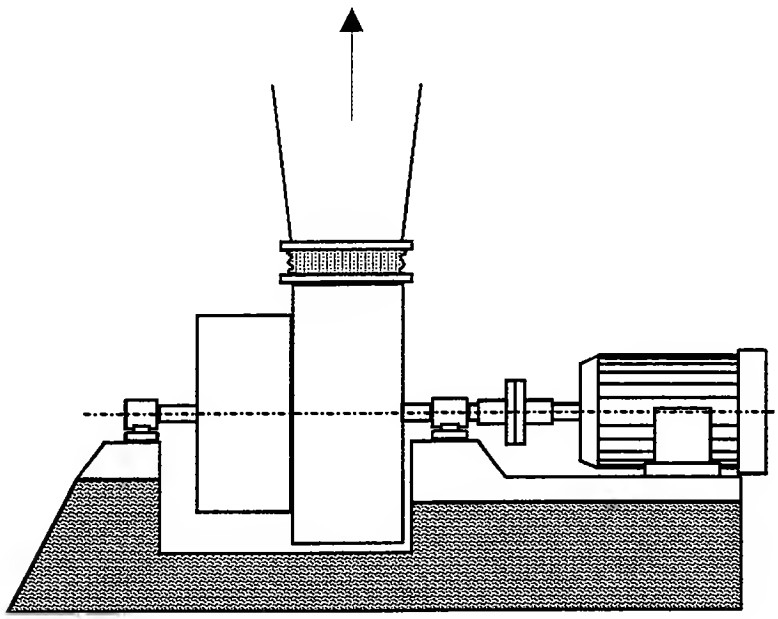
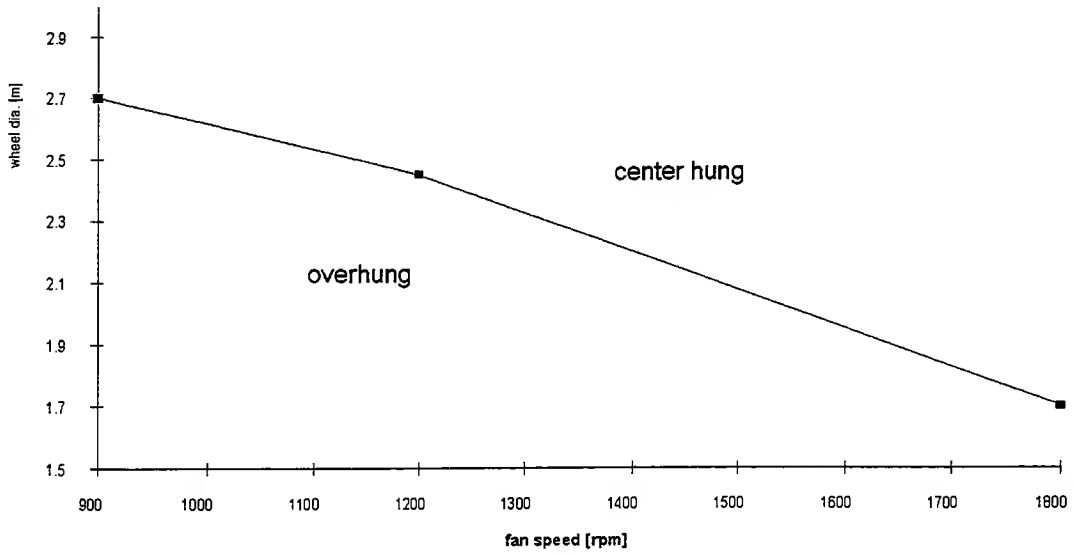


Figure 8.3 indicates the application range for the two arrangements.

Fig. 8.3



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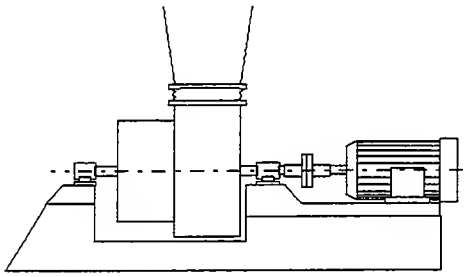
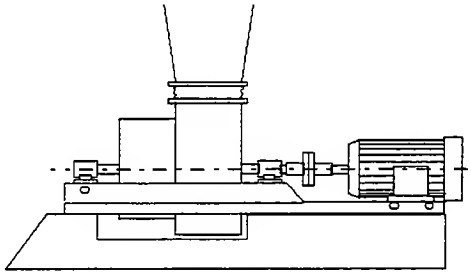
8.2 Foundations

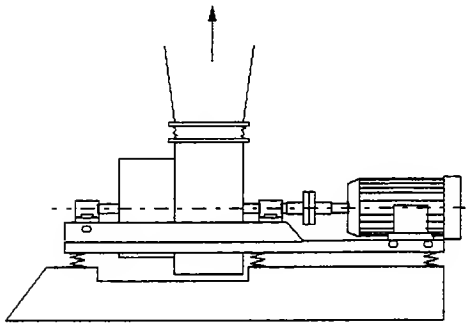
Fig. 8.4



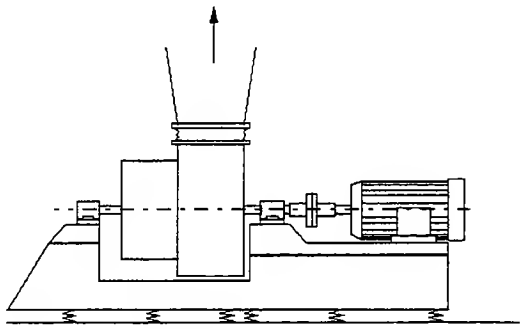
1. separate motor baseframe



2. common baseframe



3. common baseframe on anti-vibration mounts



4. concrete slab

1) Directly onto concrete with separate motor-bearing base-plate (this is the most economical solution and most widely used).
2) Onto concrete with common base plate with centerline axis support maintaining the rotor-bearing-motor alignment with the stator (more expensive solution, but easy to install onto the foundations).
   Used for hot gas fans where thermal expansion must be taken into account.
3) Common base-plate with centerline axis support resting on anti-vibration mounts (this chassis must be perfectly rigid and is thus very expensive).
   Used only in very special cases (fans mounted on steel structures).
4) With spring-supported concrete block resting on anti-vibration mounts and supporting the fan (a spring supported block is less expensive than a common base-plate with centerline support and allows for remarkable vibration absorption. Its weight, between 10 - 60 tons must be included in design calculations).
   Used only in very special cases.

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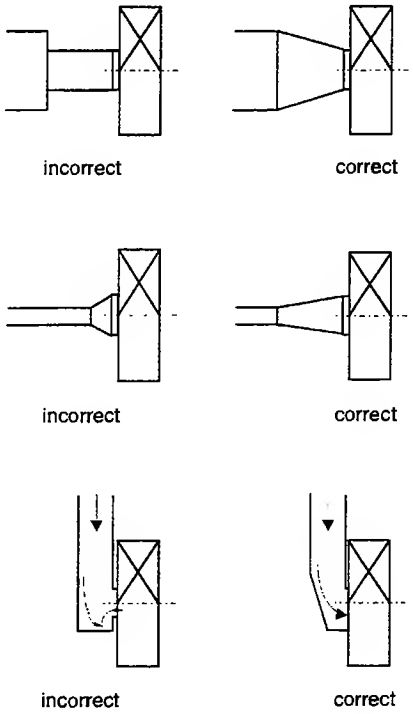
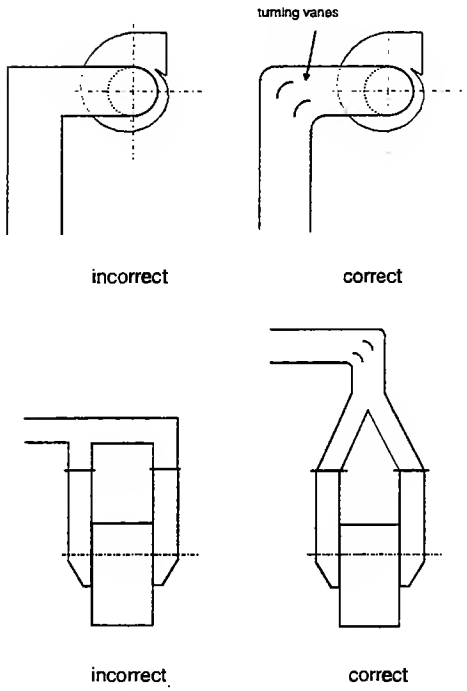
8.3 Connections

8.3.1 Inlet connections

Inlet connections



Inlet connections



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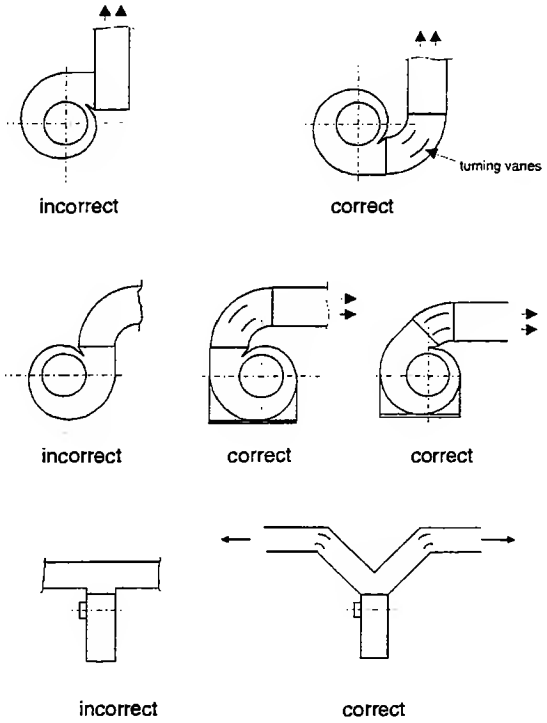
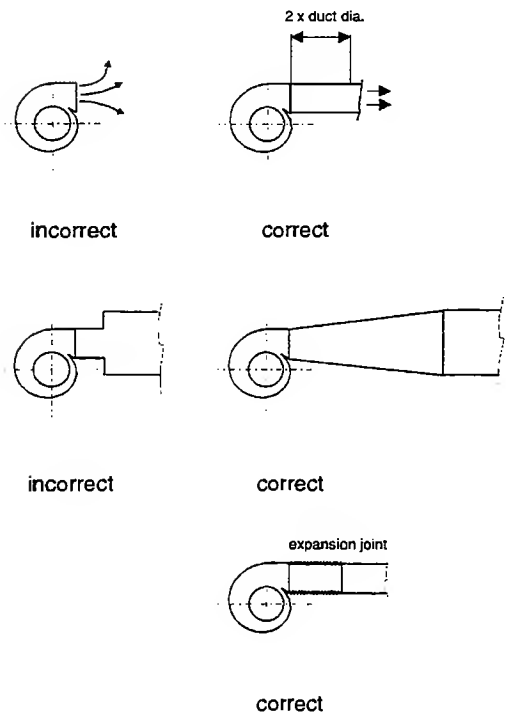
8.3.2 Outlet connections

Outlet connections



Outlet connections



9. INFORMATION SOURCES

- Robinson Industries, Inc., Zelienople, PA, USA
- TLT-Babcock, Inc., Akron, Ohio, USA
- Venti Oelde, Oelde, Germany
- Solyvent-Ventec, Chalon-Sur-Saône, Cedex, France
- Balance Dynamics Corporation, Ann Arbor, Michigan, USA, Fax # 313 994 3690

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Binder 6

Process Technology III

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Chapter 1

Emission Control

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Sources and Reduction of NOx-Emissions

P. Kutschera
PT 96/14160/E

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Summary:

Cement kiln NOx emissions are between 300 and 2500 mg NO_2 / Nm^3.

The degree of NOx emission is mainly determined by

- Flame temperature
- Oxygen content
- Residence time of exhaust gas in the kiln
- Fuel-N
- Primary measures against NOx
- Secondary measures against NOx

The NOx control technologies available for cement kilns include:

- Combustion Operational Modifications (COM)
- Low NOx Burners (LNB)
- Staged Air Combustion (SAC)
- Selective Non-Catalytic Reduction (SNCR)

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1. INTRODUCTION

NOx is produced to different degrees in all stationary and mobile combustion sources. Because of the high flame temperatures the NO generation in cement kilns is relatively high.

2. CHEMICAL AND PHYSICAL PROPERTIES AND ENVIRONMENTAL ASPECTS OF SOME NITROGEN COMPOUNDS AND OZONE

2.1 Nitrogen (N_2)

2.1.1 Physical Properties [1]

At atmospheric pressure and room temperature, nitrogen is a colorless, odorless, non-combustible gas. Nitrogen condenses to a colorless liquid at -195.80°C and 101.3 kPa and forms a white solid at -209.86°C.

M_r	28.0134
Triple point	
T	63.15 K
p	12.463 kPa
heat of fusion	25.8 kJ/kg
Boiling point (101.3 kPa)	77.35 K
heat of vaporization	199 kJ/kg
Critical point	
T_{crit}	126.2 K
p_{crit}	3.39908 Mpa
Q_{crit}	314.03 g/L
Properties at 0°C, 101.3 kPa:	
Relative density (air = 1)	0.967
Specific heat capacity	1.039 $Jg^{-1} K^{-1}$
Dynamic viscosity	15.9 x 10^{-6} Pa s
Thermal conductivity	23.86 $mWm^{-1} K^{-1}$

2.1.2 Chemical Properties [1]

Nitrogen has an extremely high heat of dissociation:

$N_2 \leftrightarrow 2\ N$ $\quad\quad$ $\Delta H_0 = 943.8$ kJ/mol

No marked dissociation takes place even at 3000°C and standard pressure. The strength of the N º N bond is responsible for the inertness of N_2.

Important reactions of nitrogen with non-metals are those with hydrogen, yielding ammonia, and with oxygen. The latter, an endothermic reaction, gives nitrogen monoxide:

$N_2 + O_2 \leftrightarrow 2\ NO$ $\quad\quad$ $\Delta H_0 = 180$ kJ/mol

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2.2 Nitrogen Oxides (NO, NO_2 N_2O)

2.2.1 Physical Properties [1]

Compounds of oxygen with nitrogen are considered as a class and called nitrogen oxides (often denoted as NOx). The known oxides and their equilibrium reactions are as follows:

			+I	
			N_2O	Dinitrogen monoxide
	+II		+II	
Nitrogen monoxide	NO	↔	N_2O_2	Dinitrogen dioxide
			+III	
			N_2O_3	Dinitrogen trioxide
	+IV		+IV	
Nitrogen dioxide	NO_2	↔	N_2O_4	Dinitrogen tetroxide
			+V	
			N_2O_5	Dinitrogen pentoxide
	+VI		+VI	
Nitrogen trioxide	NO_3	↔	N_2O_6	Dinitrogen hexoxide

Table 1: Physical properties of nitrogen oxides

Compound	N_2O	NO	NO_2 / N_2O_4	N_2O_3	N_2O_5
Oxidation state	1.000	2.000	+4 / +4	3.000	5.000
T_{cr}, °C	36.410	-93.000	157.850		
p_{cr}, MPa	7.245	6.485	10.132		
Q_{cr}, kg/m^3	452.000	520.000	550.000		
mp, °C	-90.860	-163.650	-11.200	-100.700	32.4*
bp, °C	-88.480	-151.770	21.150	-40 to +3	
Specific heat c_p, kJ kg^{-1} K^{-1}	0.879	0.996	1.326	0.862	0.778
Standard enthalpy of formation $\Delta H°_F$, kJ/kg	1864.190	3007.684	721.199	1101.435	104.589
Heat of vaporization at *bp*, kJ/kg	376.070	459.031	414.257	517.416	
Density, kg/m3 Gas (0°C, 101.3 kPa) Liquid (20°C, 101.3 kPa)	 1.9775 793	 1.3402	 3.4 (20°C) 1446.8	 1.447 (2°C)	 2.05 (solid)
Dynamic viscosity, mPa -s Gas (25°C, 101.3 kPa)	 14.874	 19.184	 12.838		
Thermal conductivity, W m^{-1} K^{-1} Gas (25°C, 101.3 kPa) Liquid (20°C, 101.3 kPa)	 0.01718	 0.02573	 0.1124 0.1336		

* Sublimation point

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N_2O

Under normal conditions (i.e. room temperature and atmospheric pressure), dinitrogen monoxide, also called nitrous oxide, N_2O, M_r 44.01, is a colorless gas with a weak, pleasant odor and a sweetish taste. If inhaled, it can bring about a spasmodic inclination to laugh and a condition resembling drunkenness hence, its historic name, laughing gas.

NO

Nitrogen monoxide, also called nitric oxide, NO, M_r 30.01, is a colorless, toxic, nonflammable gas at room temperature. As soon as it comes in contact with atmospheric oxygen, it is oxidized to nitrogen dioxide, a brown vapor.

NO_2

Nitrogen dioxide, NO_2, M_r 46.01, is a brownish red, toxic gas with a pungent odor; for physical properties, see Table 1.

2.2.2 Toxicology [1]

NO

Pure nitrogen monoxide does not have any irritating effects. It reacts, however, with hemoglobin to form methemoglobin, resulting in cyanosis and possible death. The TLV-TWA value is 25 ppm (31 mg/m^3).

NO_2

Nitrogen dioxide is an irritant gas. Its MAK value is 5 ppm (9 mg/m³). TLV-TWA 3 ppm (5.6 mg/m³), TLV-STEL 5 ppm (9.4 mg/m³). Inhalation of nitrogen dioxide causes pulmonary edema which may result in death (lethal dose 200 ppm). The substance is only slightly water-soluble but highly lipid-soluble. It therefore penetrates the alveoli where it damages the capillary walls resulting in exudative inflammation. The respiratory tract is obstructed due to formation of foam.

Concentrations exceeding 60 - 150 ppm produce coughing and a burning sensation in the chest. Pulmonary edema becomes apparent after 2 - 24 h. The patient suffers respiratory distress and insomnia. Chronic exposure to low doses results in coughing, headache, loss of appetite and gastrointestinal disorders. Patients should be kept under clinical observation. Inhalation of ammonia from ammonium hydrogen carbonate is recommended.

N_2O

Dinitrogen monoxide (laughing gas) does not irritate the mucous membranes. It has a powerful analgesic action but is only weakly narcotic. The gas displaces nitrogen from air-filled body cavities (middle ear, sinuses, intestines, brain ventricles) resulting in an increase in pressure. After chronic exposure, polyneuropathy and myelopathy have been observed. TLV-TWA value is 50 ppm (90 mg/m³).

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2.2.3 Environmental Aspects [2]

Conditions which produce nitrogen oxides do not occur solely in stationary combustion sources. The exhaust gases from **internal-combustion engines**, particularly those of the gasoline-burning spark-ignition variety, contain traces of nitrogen oxides, and because of the sheer number of them they are estimated to contribute about 50% of the total anthropogenic NOx burden.

Vehicle exhaust also contains the other ingredients needed to produce the effect known as photo-chemical smog; this has long plagued large conurbations in badly ventilated situations such as Los Angeles, Tokyo and Mexico City, and it has more recently become a feature of many other urban centers. The nitrogen oxides, activated by solar ultraviolet, react in the urban atmosphere with the unburned hydrocarbons from vehicle exhausts to produce a noxious cocktail of corrosive, oxidizing and irritating chemicals such as organic peroxides.

They also interact with atmospheric oxygen to produce ozone. It is ironic that, at a time when one form of atmospheric pollution is destroying the ozone in the stratosphere, which provides vital protection against excessive solar radiation at the surface, another is creating it near ground level, where it is an economic and health hazard. Vehicle exhaust is, of course, discharged virtually at ground level, and its effects are "local" rather than global: the smog pall from Los Angeles sometimes extends 100 miles or more inland.

But nitrogen oxides are now known to be a key component of a much more widespread problem which is even more serious in terms of its international implications than is photochemical smog. It is now plain that a sizable proportion of the blame for the very topical environmental concern of acid rain which was formerly perceived as an effect only of sulfur oxides, can in fact be attached to nitrogen oxides. Acid rain which ought really to be termed "acid precipitation" since the most spectacular manifestations tend to occur in snow rather than rain is no observer of national boundaries. Pollutants released in one country may come to earth in precipitation not just in neighboring countries but even in other continents.

As the extent of the problem and its causes have become better understood and documented, it has become the subject of international dispute and negotiation. Ecological damage to the forests and lakes of Eastern Canada has been ascribed to (amongst other things) acidity originating from industrial sources in the US Mid-West. Similar damage which has ravaged the forests and lakes of Scandinavia has been blamed on pollution exported from the nearer EEC countries, especially the United Kingdom and Germany, and former communist central European countries such as Poland, Czechoslovakia and Romania. And since the communist regimes in these countries collapsed the appalling extent of the devastation of their own forests has become only too apparent.

By what mechanism acid rain causes forest damage is not known for certain. Two theories are currently given greatest credence. One proposes that the acidity leaches cationic nutrients such as calcium, magnesium and potassium away from the root zone and down into the subsoil, causing starvation. The other postulates that the acidity solubilizes normally immobile aluminum in the soil; this interferes with the normal uptake of other cationic nutrients and, if it reaches a threshold concentration (which varies according to the species of tree), it is directly toxic. This could also explain the dire effect on the aquatic life of lakes and rivers in the affected regions.

Other pollutants reckoned to damage forests include heavy metals which are given off in coal smoke as well as in emissions from metal smelters and ozone which - as already mentioned - is generated in interactions between waste nitrogen oxides and atmospheric oxygen.

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Studies made in the United States for the National Acid Precipitation Assessment Program (NAPAP) - a statutory body set up under the acid Precipitation Act of 1980 - have shown that the acidity is worse in high-level rain clouds than in lower-level clouds, suggesting that the pollutants causing it are carried up into the higher levels rather than diffusing up from ground level. So, while it is necessary to tackle exhaust emissions from vehicles to alleviate the smog problem and any strategy for curbing **the contribution of nitrogen oxides to acid rain must center on the control of emissions from the large stationary sources.**

2.3 Ammonia

Ammonia, NH_3, occurs in nature almost exclusively in the form of ammonium salts. Natural formation of ammonia is primarily by decomposition of nitrogen-containing organic materials or through volcanic activity. Ammonia and its oxidation products which combine to form ammonium nitrate and nitrite, are produced from nitrogen and water vapor through electrical discharges in the atmosphere.

These ammonium salts, as well as those arising from industrial and automotive exhausts, supply significant quantities of the nitrogen needed by growing plants when eventually deposited on the earth's surface. Ammonia and its salts are also byproducts of commercial processing (gasification, cooking) of fuels with vegetable origins such as coal, lignite and peat.

2.3.1 Physical Properties

M_r	17.0312
Liquid density (at -33.43°C, 101.3 kPa)	0.682 g/cm³
Gas density (at -33.43°C, 101.3 kPa)	0.888 g/L
Melting point (triple point)	-77.71°C
Vapor pressure (triple point)	6.077 kPa
Boiling point (at 101.3 kPa)	-33.43°C
Heat of vaporization (at 101.3 kPa)	1370 kJ/kg
Standard enthalpy of formation (gas at 25°C)	-45.72 kJ/mol
Net heating value, LHV	18.577 kJ/g
Gross heating value, HHV	22.543 kJ/g
Ignition temperature acc. to DIN 51 794	651°C
Explosive limits	
NH_3 - O_2 mixture (at 20°C, 101.3 kPa)	15 - 79 vol % NH_3
NH_3 - air mixture	
(at 0°C, 101.3 kPa)	16 - 27 vol% NH_3
(at 100°C, 101.3 kPa)	15.5 - 28 vol% NH_3

2.3.2 Chemical Properties

Gaseous ammonia reacts very violently to explosively with nitrogen oxides to form nitrogen, water, ammonium nitrate or nitrite. The reaction with N_2O does require ignition.

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2.3.3 Toxicology

Ammonia is a strong local irritant. On mucous membranes alkaline ammonium hydroxide forms which dissolves cellular proteins and causes severe necrosis (corrosive effect).

Ammonia or ammonium hydroxide can penetrate the cornea rapidly, leading to keratitis, damage of the iris, cataract and glaucom.

Oral ingestion of aqueous ammonia can corrode the mucous membranes of the oral cavity, pharynx and esophagus and cause the shock syndrome, toxic hepatitis and nephritis.

Ammonia is absorbed rapidly by the wet membranes of body surfaces as ammonium hydroxide, converted to urea and excreted by the kidneys.

Human Exposure: Concentrations of 50 ppm are perceived easily; 50 - 72 ppm does not disturb respiration significantly. Levels of 100 ppm irritate nose and throat and cause a burning sensation in the eyes and tachypnoe. In addition to the symptoms described above, 200 ppm induce headache and nausea; 250 - 500 ppm, tachypnoe and tachycardia; 700 ppm, immediate onset of burning sensations in the eyes; 1000 ppm causes immediate coughing.

The nitrogen metabolism is not significantly changed after exposure to 500 ppm of ammonia.

The TLV value has been set at 25 ppm with a short-term limit exposure value at 35 ppm; the MAK is established at 50 ppm.

2.4 Ozone

Ozone is thermodynamically unstable and spontaneously reverts back into diatomic oxygen. This process is promoted by the presence of transition metals or their oxides.

An irritating pale blue gas, ozone is explosive and toxic, even at very low concentrations. At -111.9°C it condenses to form a dark violet liquid which freezes at -192.7°C. In the Earth's stratosphere, it occurs naturally (5-10 ppm), protecting the planet and its inhabitants by absorbing ultraviolet radiation of wavelength 290-320 nm.

2.4.1 Physical Properties

M_r	48.0
bp (101 kPa)	-111.9°C
mp	-192.7°C
Critical temperature	-12.1°C
Critical pressure	5.53 MPa
Critical density	437 kg/m^3
Critical volume	1.471 x 10-4 m^3/mol
Heat capacity, gas	
0°C	794 Jkg^{-1} K^{-1}
25°C	818 Jkg^{-1} K^{-1}
Heat of vaporization	15.2 kJ/mol
Heat of formation	144.8 kJ/mol

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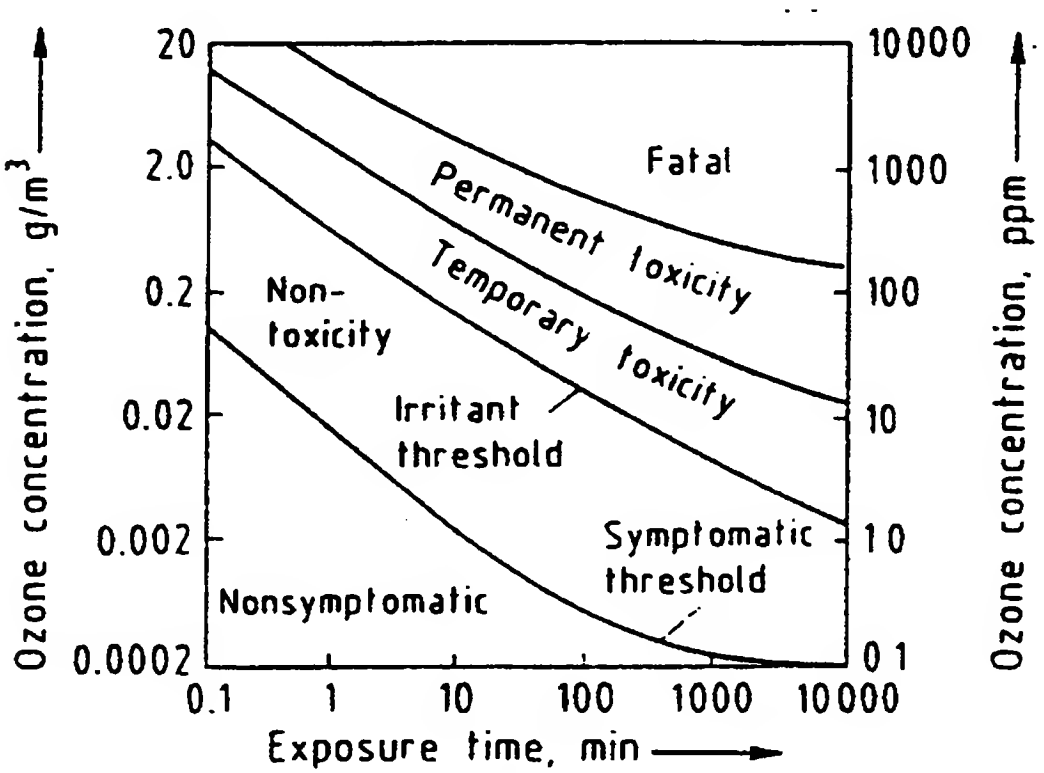
2.4.2 Toxicity

Ozone, being an extremely powerful oxidizing agent, readily oxidizes a variety of functional groups in biochemicals. Studies indicate that free radical formation, lipid peroxidation, carbonyl and aldehyde formation, and oxidation of SH groups, are some of the major sites of attack. Ozone readily reacts with olefinic compounds, particularly polyunsaturated lipids, forming unstable ozonides. Their decomposition results in the formation of toxic free radicals which can in turn amplify the primary cytotoxic or tissue damage. While ozone is considered to be a toxic gas, there are factors which mitigate the immediate danger to individuals working with it. Toxicity is dependent on concentration and length of exposure. OSHA has set an 8-h TWA-PEL of 0.2 mg/m³ (0.1 ppm) for ozone.

Fig. 1 illustrates the relationship between various exposure levels and exposure time for humans. The odor threshold concentration for ozone is approx. 0.02-0.04 mg/m³ (0.01 - 0.02 ppm).

Figure 1: Human toxicity limits for ozone exposure



2.4.3 Formation of Trophospheric Ozone

$$CO + OH \rightarrow H + CO_2$$
$$H + O_2\ H\ M \rightarrow HO_2 + M$$
$$HO_2 + NO \rightarrow OH + NO_2$$
$$NO_2 + \text{Light } (\lambda < 420\ nm) \rightarrow NO + O$$
$$O + O_2 + M \rightarrow O_3 + M$$

Net: $CO + 2\ O_2 + \text{Light } (\lambda < 420\ nm) \rightarrow CO_2 + O_3$

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3. PRESENT SITUATION

3.1 Present State of Cement Kiln Emission

There is no normal or average NO_x emission from cement kilns. Many factors like kiln system and fuel characteristics are influencing the NO_x emission.

3.2 Present Legal Situation [16]

All combustion processes primarily produce nitrogen monoxide NO with a much smaller proportion of nitrogen dioxide NO2 (of the order of 5%). In the free atmosphere, i.e. at relatively low temperature, however, nitrogen monoxide is oxidized further to form nitrogen dioxide NO2. Owing to this oxidation, no difference is made between the two gases when emissions are concerned and they are referred to generally under the formula NO_x (= NO + NO2 expressed as NO2), or nitrogen oxides. However, decisive for the NO_x emission is the formation of nitrogen monoxide (NO) in the kiln system.

Emission Limits

$Nm^3_{dry} = m^3$ at 273 K, 101300 Pa and 0% water

Europe	200 - 1800 mg / Nm^3
Germany	
new plants / modification	500 mg / Nm^3_{dry}
existing plants	800 mg / Nm^3_{dry}
100% waste burning (17. BImSchV)	200mg / Nm^3_{dry}
USA	720 - 1100 mg/Nm^3

US plants burning hazardous waste are regulated under BIF (Burners and Industrial Furnaces). Other plants do have a state permit defining certain parameters like NO_x, SO_2, CO and THC emission. The limits for these emissions are called emission standards. This standards are individually defined for each plant and usually represent the operating situation under certain conditions. Therefore, the US standards are different from the emission limits in Europe where emission limits are valid for a whole state or country.

For comparison reason all emission limits/standards are indicated in mg/Nm^3. The emission standards in the USA are usually not using mg/Nm^3 but ppm, lb/$t_{dry\ feed}$, gr/dscf, lb/1000lb_{gas}, lb/hr, etc. To convert them into mg/Nm^3 certain assumptions were necessary.

All the above explained emission limits do include definitions how and when the compliance tests have to be carried out. It is, e.g. a very important difference whether the emission has to be measured continuously or not.

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4. NITROGEN INPUT INTO THE KILN SYSTEM

Nitrogen is introduced into the kiln system as molecular nitrogen (N_2) in the combustion air (primary, secondary, tertiary) and as nitrogen compounds in the fuel. The quantity of the relevant N_2 introduced into the kiln system is defined by the stoichiometric air required for the combustion plus excess air.

The concentration of N_2 in the air is always 78%.

The quantity of fuel-N is defined by the fuel input and the nitrogen content in the fuel:

Heavy oil:	700 - 1000 ppm by mass of N
Coal:	1000 - 4000 ppm by mass of N

5. BEHAVIOR OF NITROGEN IN THE PROCESS

5.1 NO Formation

NO formation only occurs at elevated temperatures (> 800°C). It is always connected with the combustion process.

5.1.1 Nitrogen Monoxide Formation Reaction Mechanism

The formation of nitrogen monoxide is not a simple process that can be described by a few equations. The complexity of the reactions involved has hitherto prevented the formulation of a conclusive theory regarding the formation of nitrogen monoxide NO. However, it appears to consist essentially of two phenomena, the products of which are referred to as "thermal NO" and "fuel NO".

"Thermal NO"

The "thermal NO" is produced by the oxidation of molecular nitrogen in the combustion air according to the formula:

$$N_2 + O_2 \rightarrow \quad 2\,NO \quad (0)$$

"Fuel NO"

The "fuel NO" is produced by the oxidation of organically bonded nitrogen in the fuel.

Formation of "Thermal NO"

In the zone of combustion products that is after the flame front (oxidation zone), "thermal NO" is produced if the combustion gases remain for a sufficiently long time at temperatures above about 1600°C. This is determined by what is known as the Zeldovich mechanism according to the reactions:

$$N_2 + O^\bullet \rightarrow NO + N^\bullet\ (K1)\ ;\ N_2 + O^\bullet \leftarrow NO + N^\bullet\ (K2) \qquad (1, 2)$$

$$N^\bullet + O_2 \rightarrow NO + N^\bullet\ (K3)\ ;\ N^\bullet + O_2 \leftarrow NO + O^\bullet\ (K4) \qquad (1, 2)$$

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Here the speed is governed by that of the fastest reaction, i.e. the reaction between atomic oxygen (radical) and the nitrogen molecule. The rate of formation is therefore proportional to the concentration of atomic oxygen and molecular nitrogen. Assuming that combustion takes place in the presence of an air surplus, the following equation is obtained:

$$\frac{d[NO]}{dt} \sim 2 \bullet K_1 \bullet [N_2] \bullet [O^\bullet] \qquad (4)$$

The pronounced dependence of K_1 on temperature can be seen in Fig. 3. Thus, at high temperatures the equilibrium of the reaction is on the side of NO formation.

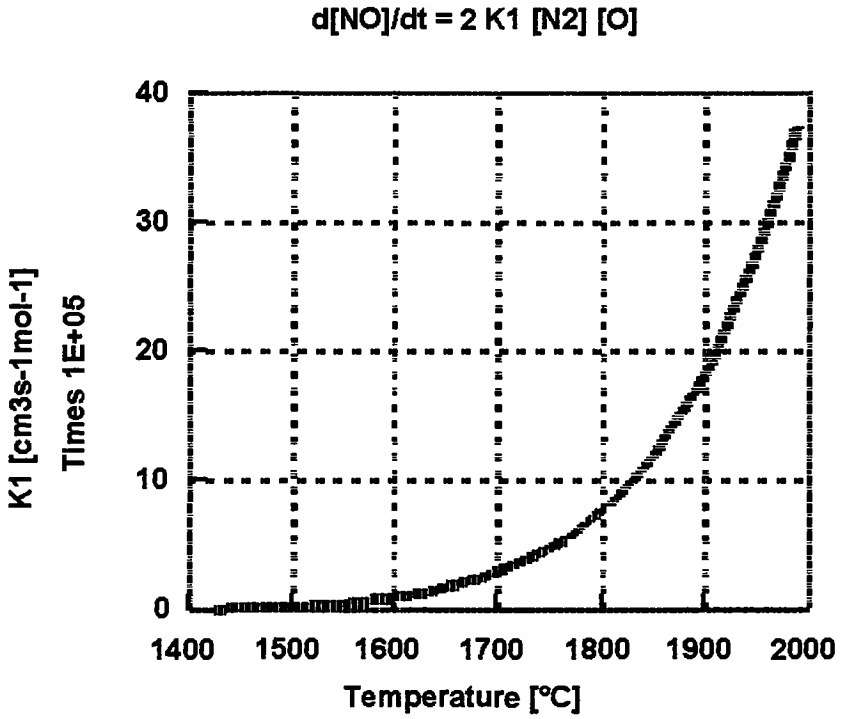
In the flames of rotary cement kilns the atomic oxygen primarily comes from the thermal dissociation of O_2:

$$O_2 \leftrightarrow 2\ O^\bullet \qquad (5)$$

Therefore, for the formation of NO the following equation is obtained:

$$\frac{d[NO]}{dt} = K \bullet [N_2] \bullet [O^\bullet]^{1/2} \qquad (6)$$

Figure 3: Dependence of K_1 on temperature



The amount of nitrogen monoxide actually produced during technical combustion processes is a long way below equilibrium concentration (cf Fig. 4) owing to the marked dependence on temperature of the NO reaction and the relatively short time that it remains at this temperature. In air 50% of the equilibrium value is attained at temperatures around 2000°C but only after about 2.5 seconds. The dwell time of the gas in the flames of rotary cement kilns are obviously a whole order of magnitude below that. Thus, if the gases remain for a long time at high temperatures, the formation of NO is favored.

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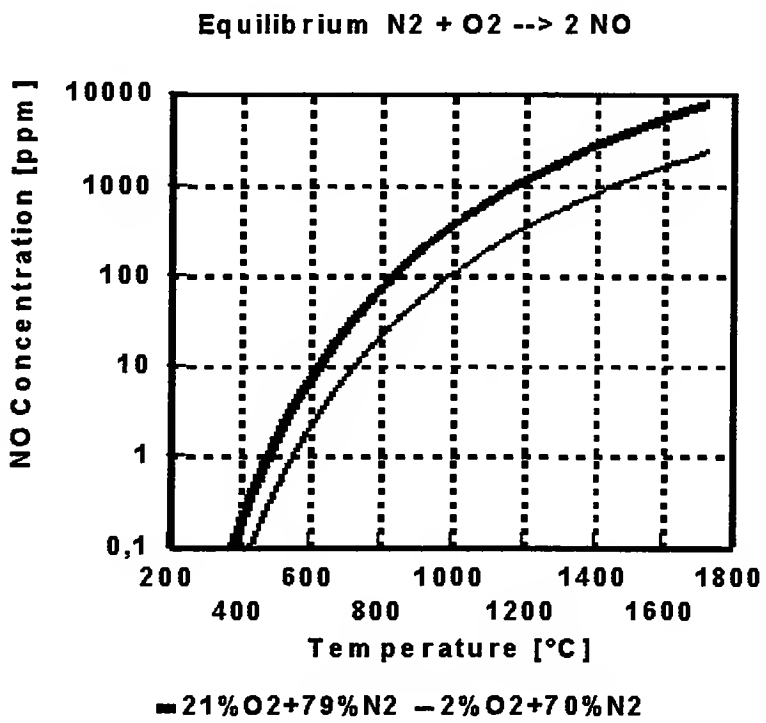
Figure 4: Equilibrium $N_2 + O_2 \rightarrow 2\ NO$



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Hence, the formation of "thermal NO" is governed by the following parameters:

- Temperature
- Gas composition
- Dwell time at high temperature

Formation of "Fuel NO"

Fossil fuels contain varying proportions of nitrogen compounds:

Heavy oil:	700 - 1'000 ppm by volume of N
Coal:	1'000 - 4'000 ppm by volume of N

Already during the pyrolysis of these fuels, i.e. at relatively low temperatures, the nitrogen compounds break off as secondary compounds, such as amines and cyanides, which are then oxidized by OH radicals or O_2 to form nitrogen monoxide, or which react with nitrogen compounds (mainly NO) to form molecular nitrogen. The formation of NO from fuel nitrogen occurs almost regardless of how it is bonded, but is largely influenced by the amount of nitrogen contained in the fuel. It can be assumed that all fuel-bonded nitrogen compounds finally adopt a composition "I", where "I" may be either NH_2 or atomic N. These are then converted into either NO or N_2:

$$I + OX \rightarrow ... \rightarrow NO + ... \quad (7)$$

$$I + NO \rightarrow ... \rightarrow N_2 + ... \quad (8)$$

Hence, the formation of "fuel NO" is governed by the following parameters:

- The formation of NO depends on the quantity but not of the nature of nitrogen compounds contained by the fuel
- With increasing air surplus the proportion of nitrogen compounds in the fuel converted into NO increases
- Under sub-stoichiometric conditions it is possible for NO to be converted into N_2. This takes place by combination of the NO with atomic nitrogen present in large proportions in the flame according to the formula:

$$NO + N^{\bullet} \rightarrow N_2 + O^{\bullet} \quad (9)$$

- The conversion of nitrogen compounds in the fuel to NO depends to a large extent on the design of the burner of the precalciner. Here such operations as
  - heating and distribution of drops or particles in the combustion air
  - driving off the volatile part
  - heterogeneous combustion
  - etc.

play a very important role

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5.1.2 NO-Decomposition Mechanism in the Combustion Process

Homogeneous Reaction

NO reacts with hydrocarbon radicals according to equation 10. The product HCN reacts later like fuel-N and can form NO as well as N_2.

$$CH_x \bullet + NO \xrightarrow{k_{10}} HCN + OH_{(x-1)} \qquad (10)$$

For a significant NO-decomposition according to (10) high hydrocarbon concentration and low air factor (~0.6 - 0.9) are required.

Also reactions between NO and HCN as well as ammonia from the fuel do have an influence on NO-decomposition.

Heterogeneous Reaction

NO can also be reduced on catalytic active surfaces of solids outside of the flame. As catalyst can act coal and ash particles and metal oxides. Prerequisite for this reaction is the presence of CO or H_2 in the exhaust gas. In case of coal particles also a gas-solid reaction between NO and the coal can decompose NO.

The gas-gas reaction proceeds according to equations 11 and 12.

$$NO + CO \xrightarrow{k_{11}} 0.5\, N_2 + CO2_2 \qquad (11)$$

$$NO + H_2 \xrightarrow{k_{12}} 0.5\, N_2 + H_2O \qquad (12)$$

5.2 Formation of Nitrogen Monoxide in a Cement Kiln

If the formation of nitrogen monoxide (NO) in a kiln has to be considered, it has to be distinguished between the formation of NO in the main firing system of the rotating part and its formation in a secondary firing system, if present.

NO Formation in the Main Firing System (rotary part)

In the firing system of a rotary kiln turbulent diffusion flames are used in which the greater part of the combustion air is only fed in after the fuel has been ignited (secondary air, proportion about 90%). As a result a severely sub-stoichiometric pyrolysis zone and flame front are produced. The nitrogen compounds in the fuel are therefore converted into NO to only a small extent. This means that only few "fuel NO" is produced, the nitrogen in the fuel is converted into N_2. In the zone of the combustion products (flue gas) temperatures up to 1800°C are attained and there "thermal NO" is produced. Thus, in the flames of the main firing systems it is mainly "thermal NO" that is produced.

NO Formation in Secondary Firing Systems

In a secondary firing system the fuel fed in burns at much lower temperatures (800 to about 1200°C) than in the main firing system. This means that the NO produced in the secondary firing system cannot have formed thermally, i.e. from N_2 and O_2 of the combustion air. In other words, it is primarily "fuel NO" that is produced in a secondary firing system.

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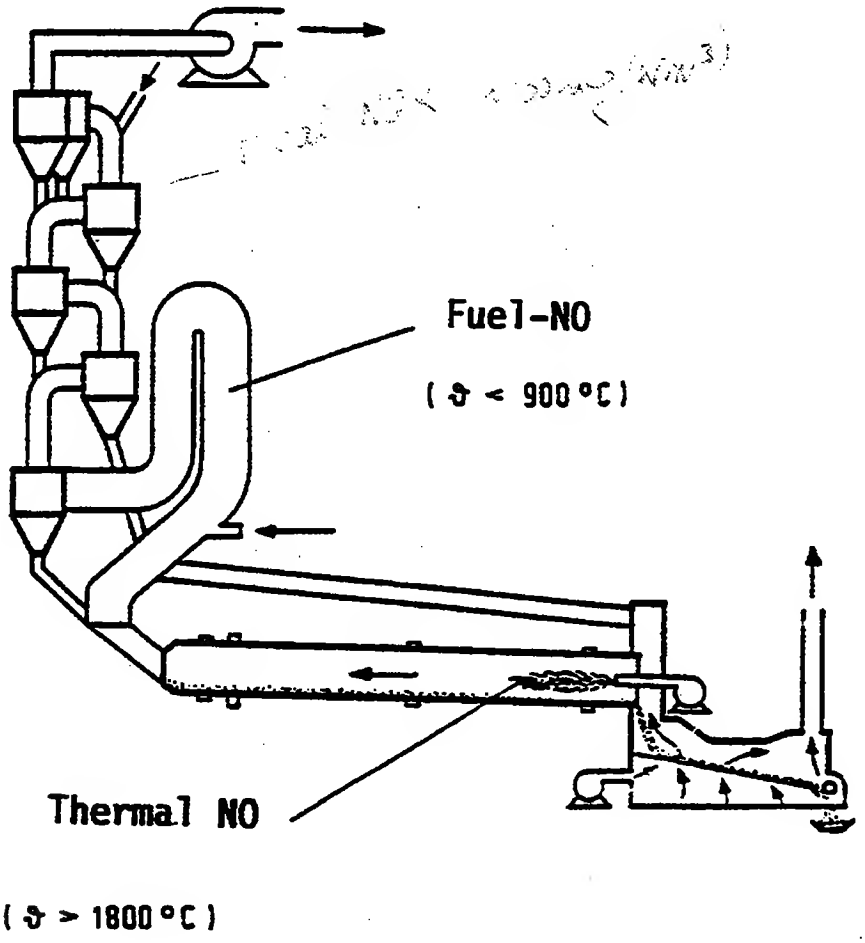
Figure 5: Regions where NO formation occurs in a rotary kiln plant with cyclone preheater, calciner and tertiary air duct



Kilns without secondary firing

All kilns without secondary firing, i.e. wet process kilns, long dry process kilns and preheater kilns without riser duct firing, have one thing in common, viz. NO emission is determined exclusively by the conditions in the kiln burning zone. After leaving the burning zone, the kiln gases drop rapidly in temperature to a level at which the NO in the gas decomposes very slowly.

Wet kilns are characterized by a specific heat consumption which is 1.5 to 2 times the specific heat consumption of a preheater kiln. This means a high specific amount of exhaust gas. On the other hand, the high specific combustion air consumption will also mean a somewhat lower secondary air temperature. This combined with the long material retention time in the burning zone should reduce the NOx concentration in the exhaust gas from the burning zone.

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SP kiln systems with riser duct firing

In many SP kiln systems 10 - 20% of the fuel is fired into the riser duct. Measurements at several riser duct fired kiln systems indicate that firing coarse fuel (e.g. old tires) into the kiln riser duct will reduce NOx emission from the kiln system.

This is probably explained by the fact that a large part of the fuel directly falls down into the kiln charge, creating a reducing atmosphere in the bottom part of the kiln back-end in which NOx from the burning zone is reduced.

Conversely, when firing finely ground fuel into the kiln riser duct, the specific NOx content in the exhaust gas will often increase on passing through the riser duct.

As the NOx emission from the kiln may also increase slightly due to an increased excess air rate the total NOx emission from the kiln system will often increase when starting up riser duct firing with finely ground fuel.

Precalcining kiln systems

In precalcining kiln systems with tertiary air duct, firing into the rotary kiln typically accounts for only 40 - 50% of the total heat consumption and the specific amount of combustion gases from the kiln burning zone is reduced proportionally. On the other hand, the NOx concentration in the kiln gas may be considerably higher than in preheater kilns.

This is probably due to the shorter material and longer gas retention times in the precalciner kiln burning zone combined with a very high secondary air temperature.

When examining the contribution from the calciner firing to the emission of NOx, we must distinguish between two basically different types of precalcining kiln systems, viz. the In-Line (ILC) type in which the kiln gas passes the firing region of the precalciner and the Separate Line (SLC) type in which the kiln exhaust gas bypasses the firing region of the precalciner.

ILC systems

In these systems the fuel combustion in the calciner takes place in a mixture of kiln exhaust gas and hot air from the cooler (tertiary air). Some of the nitrogen in the fuel reacts with NO from the kiln exhaust gas while another part reacts with oxygen (from the tertiary air) to form NO.

The result may be a net production as well as a net reduction of NO in the calciner. However, in most cases the calciner contributes a little to the NO emission.

SLC systems

In these systems the combustion in the calciner takes place in pure hot air. In the case of oil firing, NO production in the calciner is negligible, but when applying fuels containing nitrogen up to 50% of the nitrogen compounds in the fuel may be converted into NO. The specific NO production in the SLC calciner may be as high as 4 lb NO2/st (1400 mg NO_2/Nm3). This was measured in a calciner fired with pet coke which has a high content of nitrogen and a low content of volatiles.

The NO in the calciner exhaust gas is added to the NO in the gas from the rotary kiln which leaves this type of kiln system without being reduced. When fired with solid fuels, SLC systems must therefore be expected to generate somewhat higher NOx emissions than ILC systems.

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5.3 Main Influencing Variables for NO Formation

Table 2: NOx forming mechanisms in the kiln system

NO Formation	Main influencing variables	
thermal NO	♦ temperature	• flame shape • temperature peaks in flame • secondary air temperature • burning zone temperature
	♦ oxygen concentration	• flame shape • ignition point • burner momentum • excess air • recirculation • reducing zone • primary air component • swirl intensity
	♦ residence time	• flame shape • kiln diameter
fuel NO	♦ fuel nitrogen content	•
	♦ oxygen concentration	• flame shape
	♦ volatiles concentration in the fuel	• excess air • recirculation • reducing zone • primary air • swirl intensity
	♦ residence time	• flame shape • gas speed

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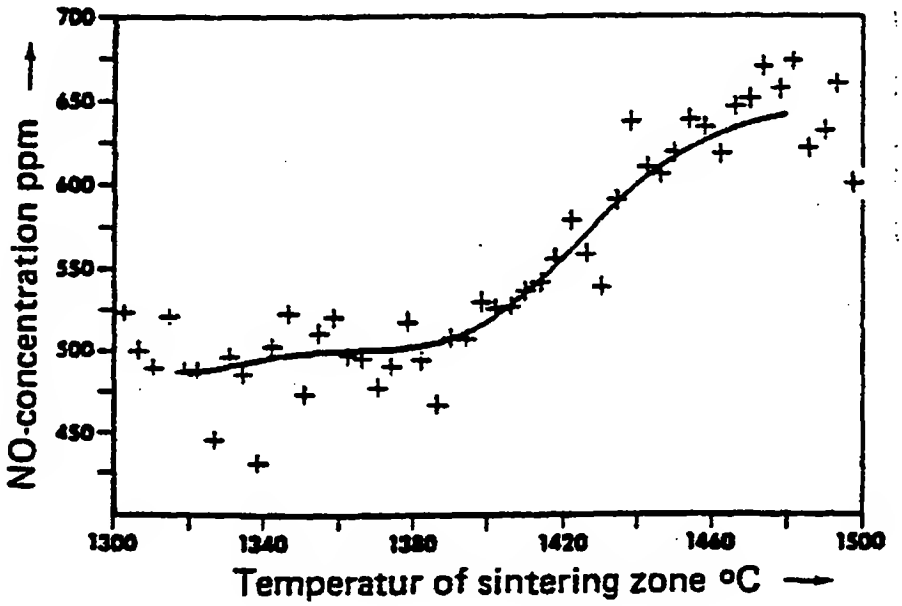
5.3.1 Temperatures

In Chapter 3.1 it is shown that the NO forming reaction is accelerated exponentially with the temperature. The temperature of the combustion gas is defined by heat generated in the flame and the heat radiation from the burning zone. Temperature has a major influence on NO formation.

Figure 8: Influence of the sintering zone temperature on the NO concentration in the waste gas



5.3.2 Temperature Peaks [6]

Fig. 9 shows the increase in thermal NO formation due to temperature peaks in an ideal chemical reactor for different amplitudes of a given temperature behavior. For calculating the curves, first the NO concentrations for the temperatures $\overline{T} + \Delta\overline{T}$ and $\overline{T} - \Delta T$ were calculated, averaged and then referred to the NO concentration for $\overline{T}$.

The subscript "L" in the diagram characterizes the combustion air temperature before the start of combustion. It appears from the diagram that the increase in NO formation becomes greater with increasing amplitude DT and decreasing average combustion air temperature. For practical purposes it is important that NO formation can increase by as much as 20 - 40 % even for very small amplitudes of the gas temperature, e.g. 100 K.

Temperature peaks arise more particularly if, in a given combustion chamber, the momentum and angle of exit of the swirl air and axial air of the rotary kiln burner cannot be optimally adjusted to each other or if short-term fluctuations in the fuel feed occur. Modern burners should therefore offer the greatest possible scope for varying their settings.

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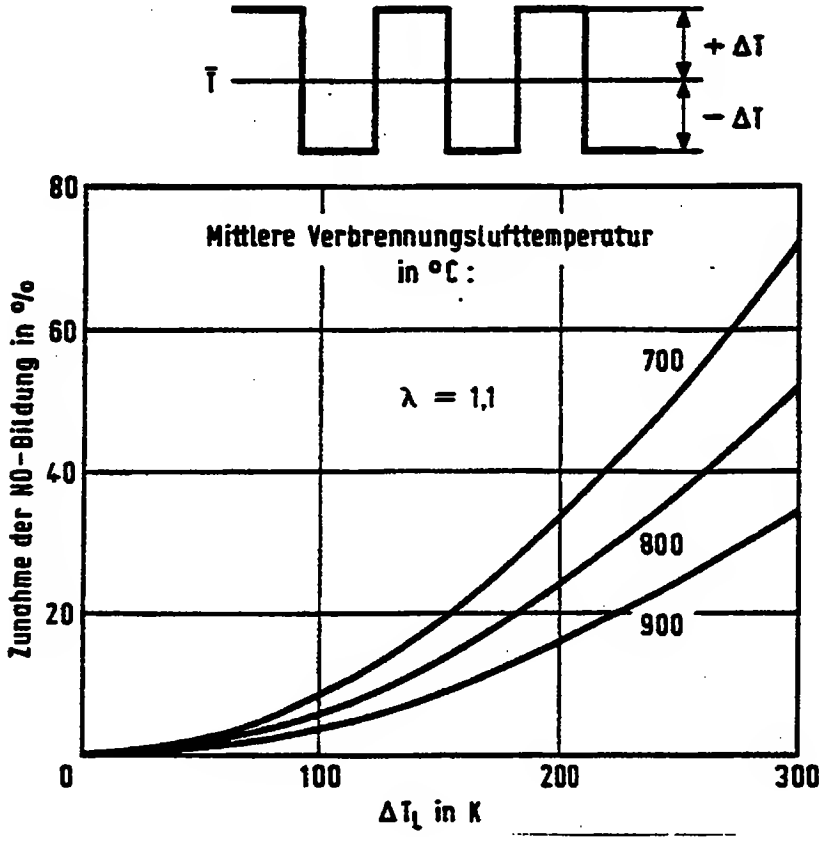
Figure 9: Calculated increase in NO formation based on local or time-dependent temperature differences with the average combustion air temperature as parameter



5.3.3 Excess Air

Fig. 10 shows the behavior of NO concentration in the waste gas. The graph can be subdivided into two areas:

A) Air excess factor < 1.2. The NO concentration increases exponentially with the O_2 concentration. This is according to the explanation in chapter 5.1 NO formation.

B) Air excess factor > 1.2. The NO concentration is not anymore a function of O_2. The high amount of excess air is cooling the flame, reducing the residence time of the gas in the kiln and diluting the exhaust gas. These 3 factors are reducing NO generation and concentration in the exhaust gas.

For most kiln systems the excess air factor is below 1.2. Therefore, oxygen concentration has a major influence on NO formation.

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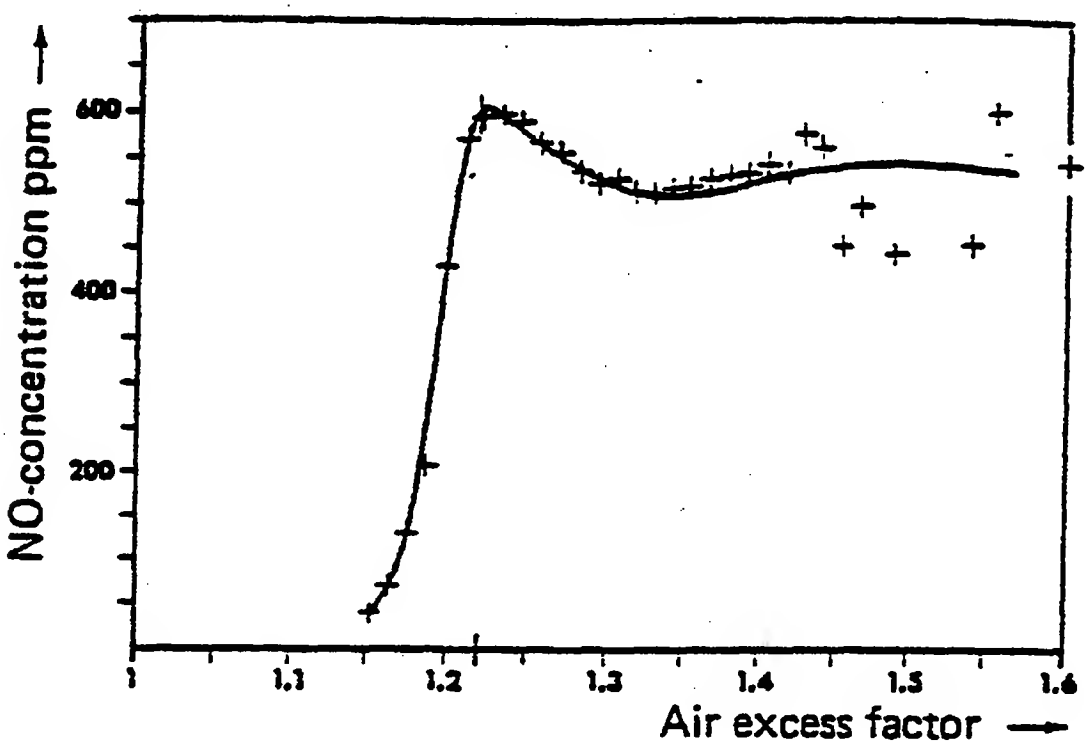
Figure 10: Influence of excess air on the NO concentration in the waste gas



5.3.4 Retention Time

The longer the combustion gas remains in the very hot part of the kiln (burning zone), the higher is the NO formation yield.

5.3.5 Burner Operating Parameters

The following burner operating parameters are influencing the flame characteristics (see [13]).

For specific information about formulas and definitions of burner aerodynamics see chapter 6 report PT 96/14078/E.

5.3.5.1 In-flame Air Level at Ignition

A minimum of air shall be entrained into the flame prior to ignition. This can be achieved by a low primary air input and the optimum use of an internal recirculation zone generated by a bluff body and/or swirl.

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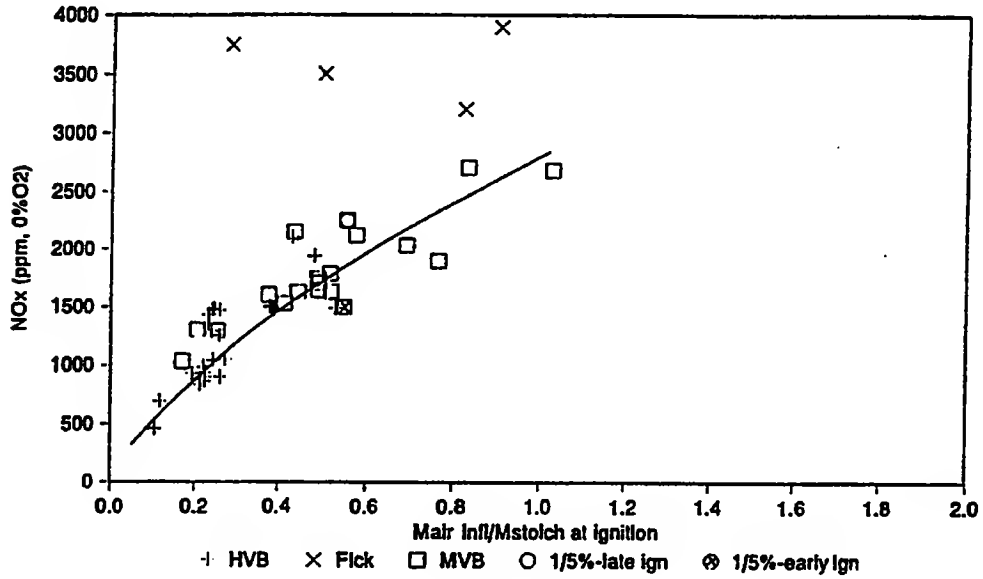
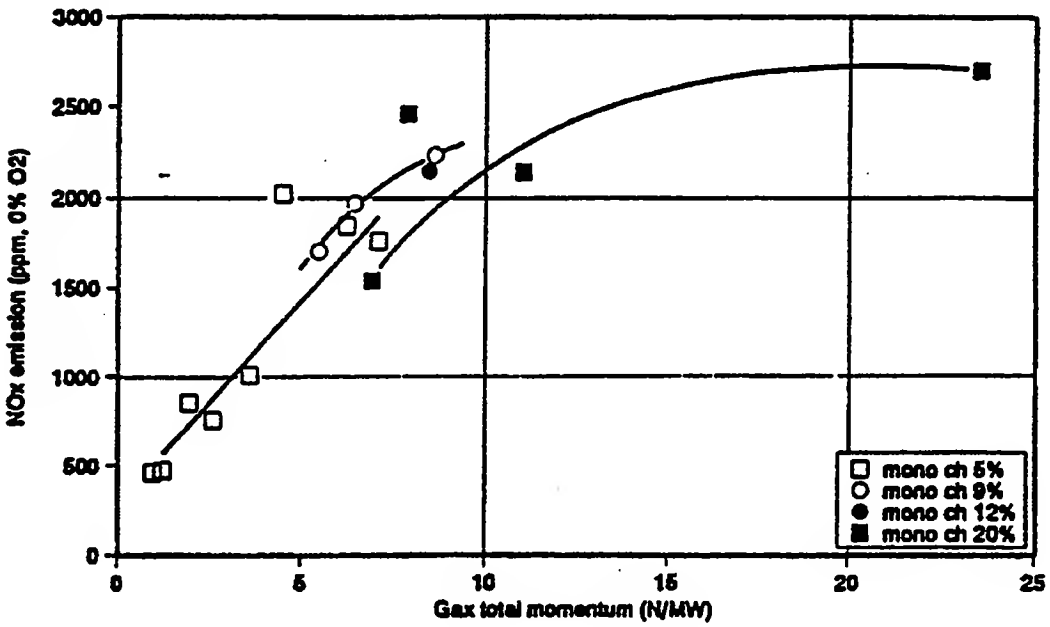
Figure 11:



5.3.5.2 Total Axial Momentum

Total axial momentum affects the overall entrainment into the flame jet. In general higher axial momentum result in enhanced mixing and higher NOx emission levels. For mono channel firing the axial momentum can be observed as a single parameter but for multi channel types a change in axial momentum also influences the tangential momentum.

Figure 12: Influence of Total Axial Momentum on NOx for Mono Channel Burners



From the Cemflame I research study it was finally concluded, that the total axial momentum should be in the range of 3 up to max. 7 N/MW.

This matter of fact however can be confirmed by the old rule of thumb, which states that the kinetic energy of the primary air jet should be kept constant within certain limits.

Applied for a typical kiln system, this formula illustrates as follows:

Figure 13: Basic Data: Production 3000t/d, Heat consumption 3140kJ/kgc, CV Coal 27.2MJ/kg

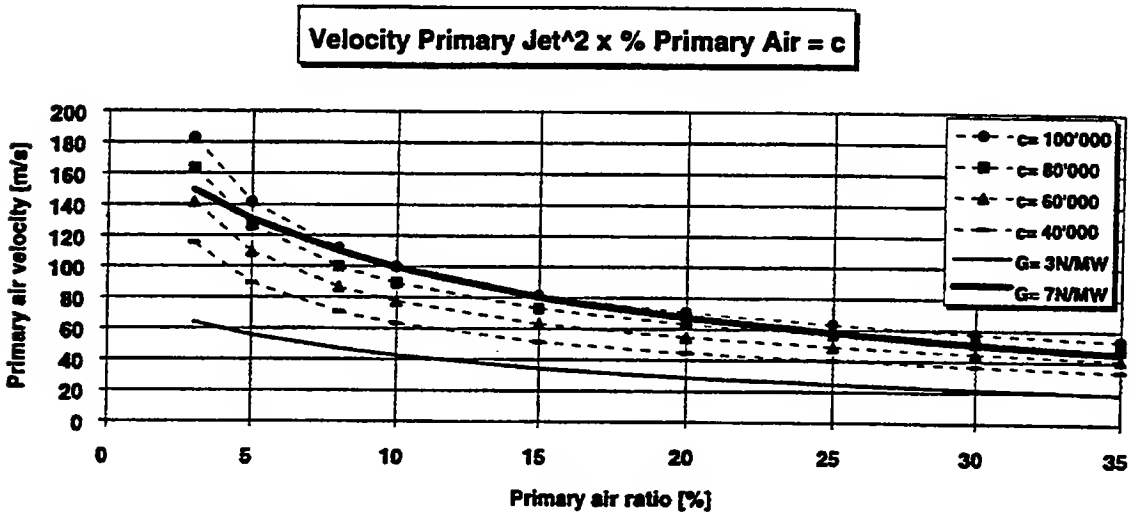
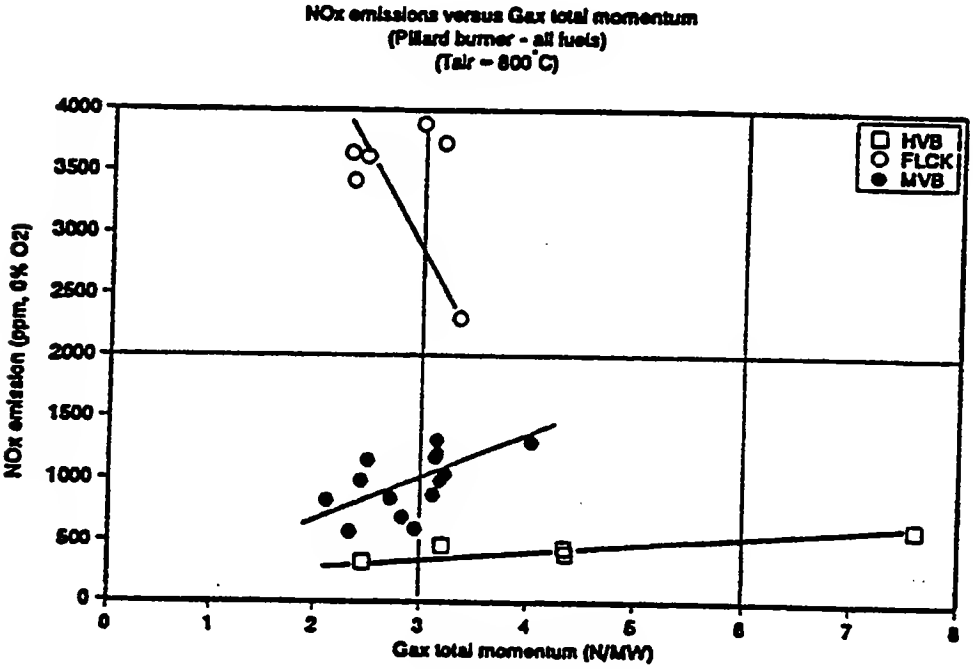


Figure 14: Influence of Total Axial Momentum on NOx for a Pillard Multi Channel Burner



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5.3.5.3 Tangential Momentum and Swirl Level on the Swirling Channel

The second parameter affecting the entrainment into the flame is the tangential momentum. In general higher tangential momentum results in more rapid heat release in the near burner zone and higher NOx emission levels.

5.3.5.4 Swirling and Axial Air Amount, Distribution and Velocity

One of the main parameters affecting the tangential momentum is the swirling air and axial air distribution and injection velocities. In combination the product massflow (kg/s) times velocity (m/s) forms the momenta (N) on the different channels. It can be noticed that both swirling air velocity and amount may have a different influence on the flame characteristics.

5.3.6 Evaluation of Characteristical Burner Data (CETIC Working Group)

During the meetings 1995/96 of the CETIC working group focusing on kiln burners a database containing a total of 42 kiln burners could be established. In particular it was one of the main topics to investigate the NOx behavior of the individual kiln burner systems.

Due to the heterogeneity of the kiln and cooler systems as well as the different fuels used, direct comparison and correlation of the existing data is limited. However certain operational guidelines and tendencies can be given.

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5.3.6.1 Burner Settings / Operational Figures / Correlations

Mono Channel Burner

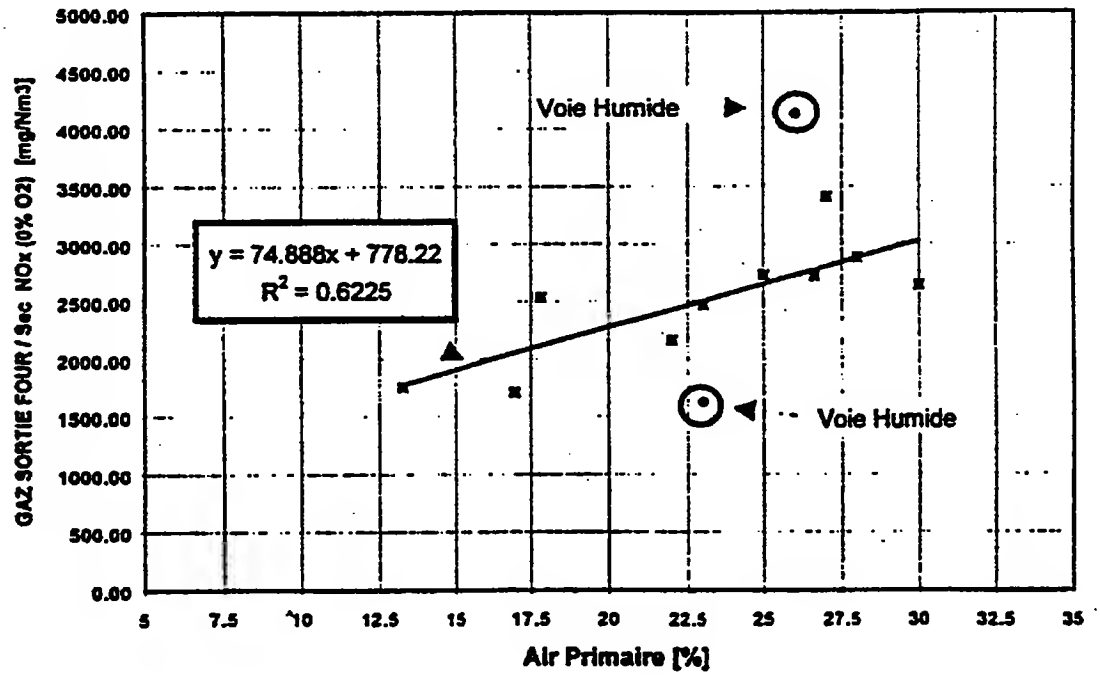
The evaluation of a total of 12 industrial mono channel burners is showing the following application range:

Primary air ratio	% Amin.	13 - 30	[%]
Injection velocity	v	60 -100	[m/s]
Specific axial momentum	G_{ax}	4.5 - 8.5	[N/MW]
Percentage of petcoke burned	% Petcoke	0 - 100	[%]
Kiln inlet NOx at 0% O_2	NO_x @ 0% O_2	1750 - 3400	[mg/Nm³]
Secondary air temperature	T_{sec}	550 - 1000	[°C]

Primary air ratio

High primary air ratios must be avoided. This has negative effects both on kiln heat consumption and on NOx emissions. The CETIC data correlation for mono channel burners, primary air ratio vs NOx emission, clearly showed a rise in NOx at higher primary air ratios.

Figure 15:



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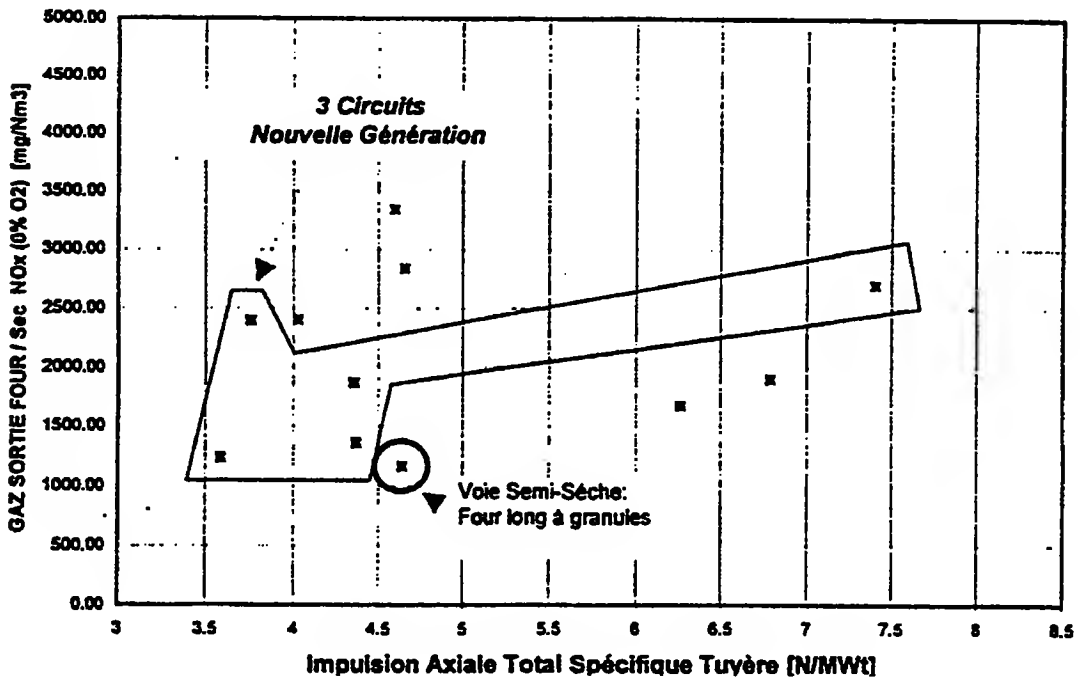
Multi Channel Burner

Primary air ratio	% Amin.	6 - 19	[%]
Percentage axial air		1.9 - 7.2	[%]
Percentage radial air		1.25 - 9.8	[%]
Percentage transport air		2.3 - 5.3	[%]
Injection velocity axial air	v_{ax}	90 - 300	[m/s]
Injection velocity radial air	v_{rad}	60 - 130	[m/s]
Injection velocity transport air	v_{tr}	14 - 38	[m/s]
Specific axial momentum	G_{ax}	3.6 - 7.4	[N/MW]
Percentage of petcoke burned	% Petcoke	0 - 100	[%]
Kiln inlet NOx at 0% O_2	NO_x @ 0% O_2	1160 - 3350	[mg/Nm³]
Secondary air temperature	T_{sec}	800 - 1010	[°C]

Specific axial momentum

For multi channel burners the impact of the specific axial momentum on NO_x emissions could be confirmed (see following graph). The specific axial momentum G_{ax} [N/MW] is in the range of 3.6N/MW up to 7.4 N/MW - very good coincidence with the 3 - 7 N/MW indicated in the Cemflame 1 research report.

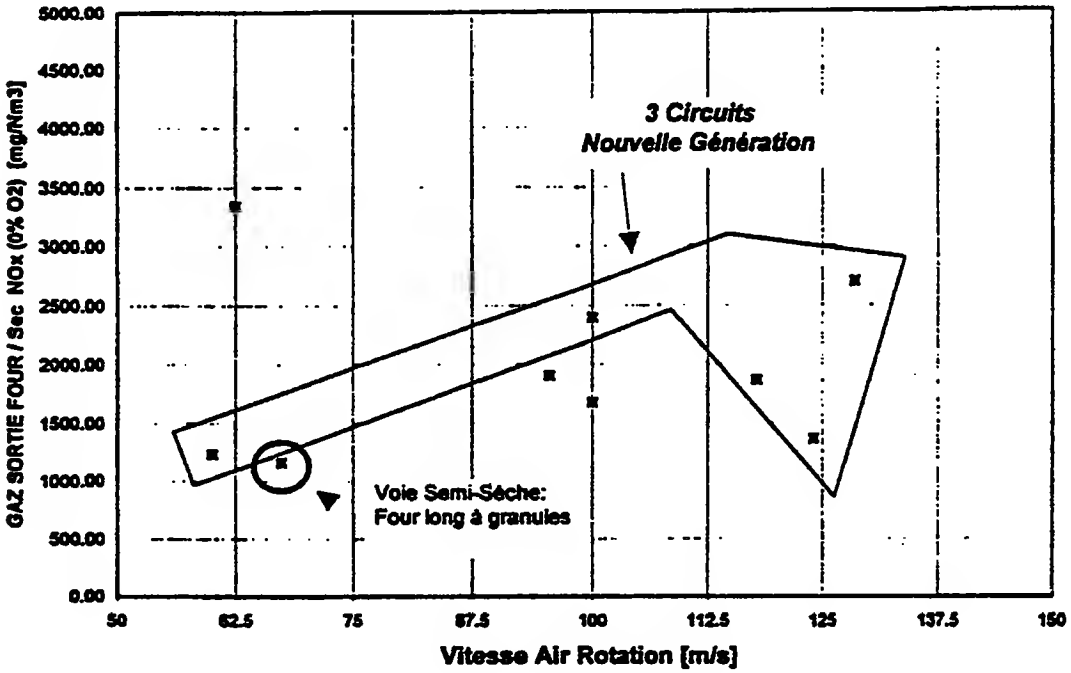
Figure 16:



Radial air velocity

From the CETIC test results further can be observed, that with increased radial velocity NOx emissions tend to be higher. This can be explained with a more rapid heat release in the near burner zone due to a higher tangential momentum.

Figure 17:



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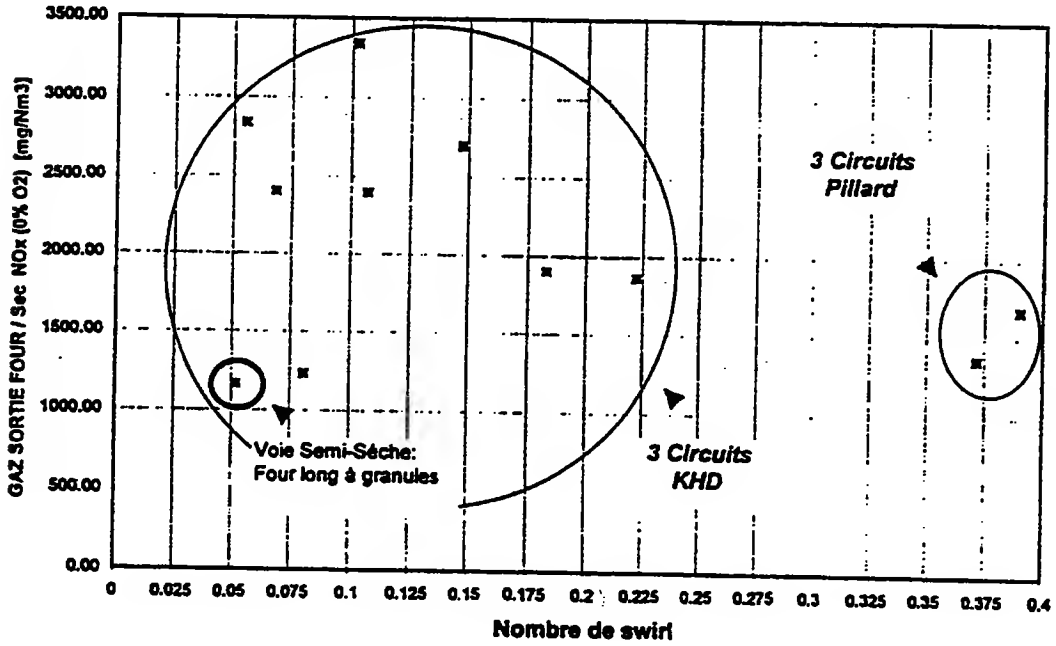
Swirl number

The CETIC attempt to correlate the swirl number with respective NOx emissions was not very successful. It is however noticeable, that the Pillard burners showed by far the highest swirl numbers.

Figure 18:



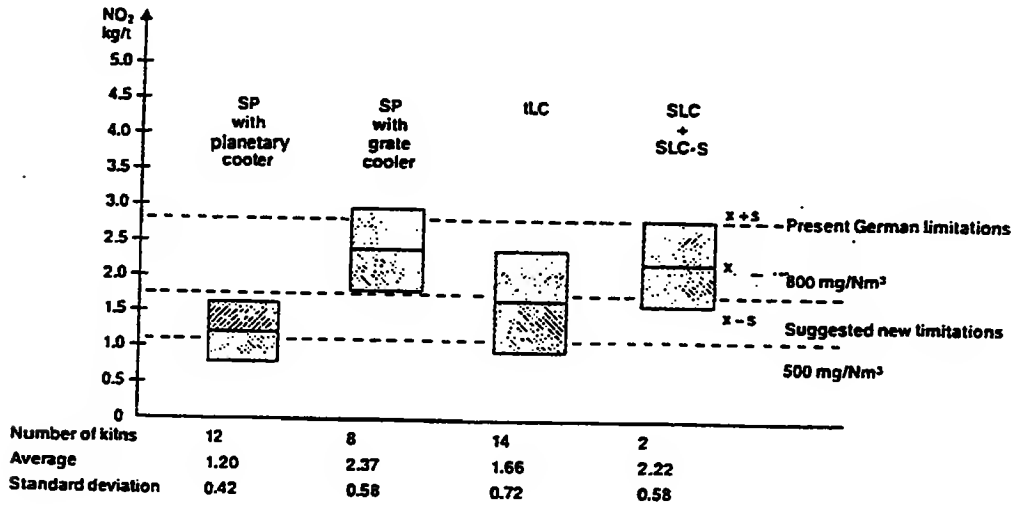
5.3.6.2 Influence of Cooler System on NOx Emissions

The following graph from reference [14] is showing the important effect of the cooler system on NOx emissions.

The grate cooler generating a hot, comparatively clean secondary air has considerable higher NOx output than the planetary cooler with its dusty secondary air, supposing identical kiln system for both cooler systems.

Mean value and standard deviation of measured NOx emission from different types of kiln systems.

Figure 19:



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6. NO EMISSION REDUCTION POSSIBILITIES

To reduce the NO emission from a pyroprocessing system to a certain controlled level, three basically different methods are available:

- Maintain the existing process while reducing the nitrogen input into the system
- Modify the existing process (primary reduction measures)
- Maintain the existing process while adding a separate gas cleaning unit for the exhaust gas (secondary reduction measures)

6.1 Reduction of Nitrogen Input

It is not economical to reduce the nitrogen content in the combustion air. However, a reduction of fuel-N for the secondary or precalciner firing may be feasible under certain conditions.

6.2 Primary Measures

Primary measures are all actions which directly influence the burning process.

6.2.1 Kiln / Clinker Cooler

- The criteria for a low NO generation are:
- constant fuel and kiln charge flow (short and long term)
- constant fuel and kiln charge composition
- constant secondary air flow and temperature
- lower burning zone temperature i.e. higher free lime
- lower LSF i.e. lower combinability (sintering) temperature
- lower flame temperature by
  - dust injection
  - water injection
  - alternative fuels with high H_2O content
- flame front closer to the burner by
  - bluff body
  - higher volatile coal
- low excess air factor
- lower primary air i.e. indirect firing, low primary air burner
- optimum distribution of primary air i.e. customized multi channel burner
- short retention time of the exhaust gas in the flame and the kiln
- minimum temperature fluctuations in kiln and cooler

To satisfy the first point an accurate dosing system and fuel transport is required.

A constant fuel quality is difficult to achieve, especially if waste fuel is used. Therefore, additional effort is needed for homogenization, preparation and analysis of the fuels.

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The amount of heat from the secondary air depends mainly on operating characteristics of the clinker cooler. The new grate plates (for the recuperating zone) developed during the last years, help to stabilize the cooler operation and the heat input into the kiln.

A constant kiln and cooler operation is a prerequisite for optimizing of the excess air factor without reducing conditions for the clinker burning. The excess air factor (l) for the main burner is always above 1.

NOx reduction measures at the burner should achieve a reduction of temperature peaks and low oxygen content in the flame. Temperature peaks can occur if at a multi channel burner air volume and exit angle of swirl and axial air are not adjusted properly and if the fuel flow is fluctuating. Therefore, the burner should have a wide

adjusting range. The optimum operating point has to be determined with systematic long time tests.

The most important factor for NO generation is the ignition distance (= distance between burner exit and ignition of the fuel). An extension of the distance is increasing the NO generation. The reason for this behavior is the degree of total air (O_2) mixed with the fuel which is higher for longer distances.

To reduce the total air mixed into the fuel, primary air flow and total burner momentum should be kept as low as possible.

The operation of LINKman is at most effective when burners are operated near their optimum momentum and can in fact lead to lower NO levels, due to lower burning temperatures via better control.

To reduce the temperature of the secondary air the tertiary air should be extracted from the cooler at the kiln hood.

To shorten the retention time of the exhaust gas in the hot kiln, the gas speed in the kiln should be as high as possible. The limits for the gas speed are dust circulations and mechanical stress for the refractory materials. The upper limit for the specific fuel heat input is 7 MW / m^2 which is about equal to a specific exhaust gas flow of 2.5 kg/m^2s.

The minimum NO concentration with an optimized low NOx burner which can be achieved today is about 800 - 1000 mg NO_2 / Nm^3.

6.2.2 Secondary Firing / Multi-Stage Combustion

Mainly fuel-N is contributing to the NO formation (fuel-NO). It is possible to design the secondary firing in such a way that only little NO is produced and that a portion of the NO from the kiln is reduced. This can be achieved with a multi stage combustion [8].

Multi-stage combustion has turned out to be a suitable method for conventional and also for highly sophisticated clinker burning processes. In the first combustion stage which extends over the sintering zone and the transition zone in the rotary kiln, combustion takes place in an oxidizing atmosphere (excess air coefficient > 1) to ensure good clinker quality. The nitrogen oxides which are inevitable formed in this high temperature zone are partially decomposed in a second combustion stage which has, at least locally, a reducing atmosphere (excess air coefficient < 1).

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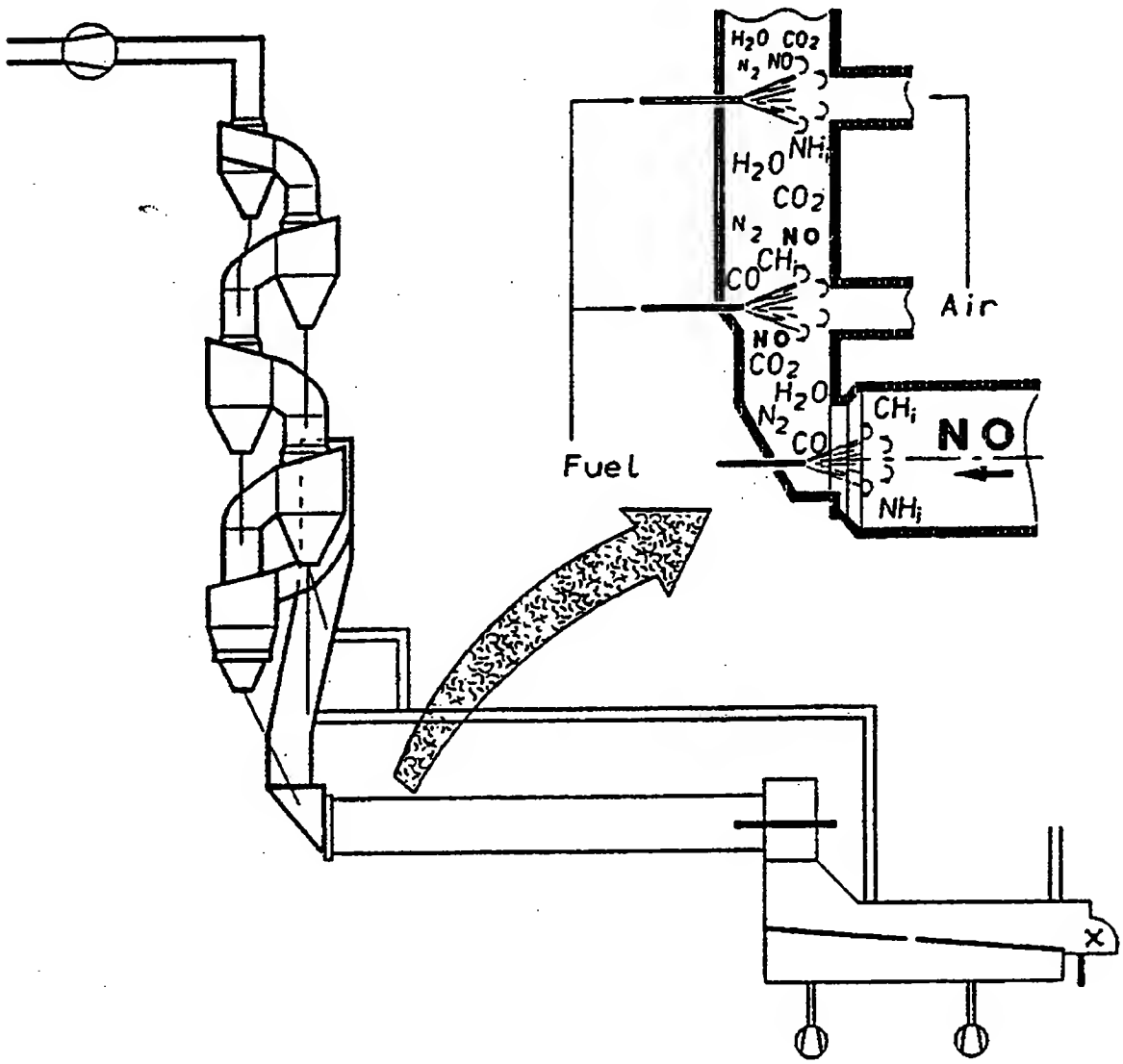
Figure 20: NOx reduction by Multi Stage Combustion



By introducing fuel through a burner in the kiln inlet zone a reducing environment is set up in this second combustion stage. The resulting intermediate products from the consecutive reactions of combustion act as reducing agents for NO created in the sintering zone and at the same time prevent the formation of more NO. This reaction pattern is shown diagramatically in Fig. 12 for a precalciner plant of the PREPOL-AS type. Multi-stage combustion is logical in precalciner plants in order to avoid renewed formation of nitrogen oxides in the calciner from nitrogen in the fuel.

A part of the precalcining fuel is introduced in the kiln inlet to form a reducing zone by arranging an air deficiency (< 1). The rest of the precalcining fuel is also supplied to the calciner, at least partially, in a reducing zone. This fuel burns here under sub- stoichiometric conditions (excess air coefficient < 1) and thus suppresses formation of NOx from the nitrogen in the fuel. In a fourth stage the unburned flue gas constitutients from the reducing zone are fully oxidized by supplying hot combustion air, whereby a turbulence generating gas flow in the calciner serves to enhance the burnout process.

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6.2.3 Possible Negative Side Effects of Primary Measures

Kiln burner

Temperature reduction:

- reduction of production capacity
- higher power and heat consumption
- clinker quality

Oxygen (excess air, primary air) reduction:

- clinker quality
- additional CO formation
- additional SO_2 volatilization
- additional TOC emissions

Secondary combustion

Lower fuel-N concentration:

- higher fuel cost

Staged combustion:

- additional CO formation
- additional TOC emissions
- difficult to control

6.3 Secondary Measures

Secondary measures are exhaust gas treatments mainly SNCR (selective non-catalytic reduction) or SCR (selective catalytic reduction).

In some cases combustion of coarse fuel (tires) in the kiln inlet is also considered as a secondary measure. Because of the reducing condition of the waste fuel combustion, NO is reduced to N_2. This reaction is called NSNCR (non-selective non-catalytic reduction).

The most efficient secondary measure which is already in operation in several kilns is the SNCR with NH_3.

6.3.1 The Selective Non-Catalytic Reduction

From tests in cement works it is known that ammonia, when injected into certain temperature zones of the cement kiln is able to reduce a large proportion of the nitrogen monoxide.

In this chapter the NH_3-injection is described.

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The Reaction Mechanism

The desired process of reducing nitrogen monoxide (NO) by ammonia (NH_3) is initiated by dissociation of the injected ammonia. At room temperature ammonia is stable, but at roughly 600 to 800°C a marked thermal decomposition is initiated by the reaction:

$$NH^{\bullet}_2 + O\,H^{\bullet} \rightarrow NH_2 + H_2O \qquad (13)$$

As the reaction proceeds, the NH_2 radical is responsible for the reaction of nitrogen monoxide (NO).

$$N\,H^{\bullet}_2 + NO \rightarrow N_2 + H_2O \qquad (14)$$

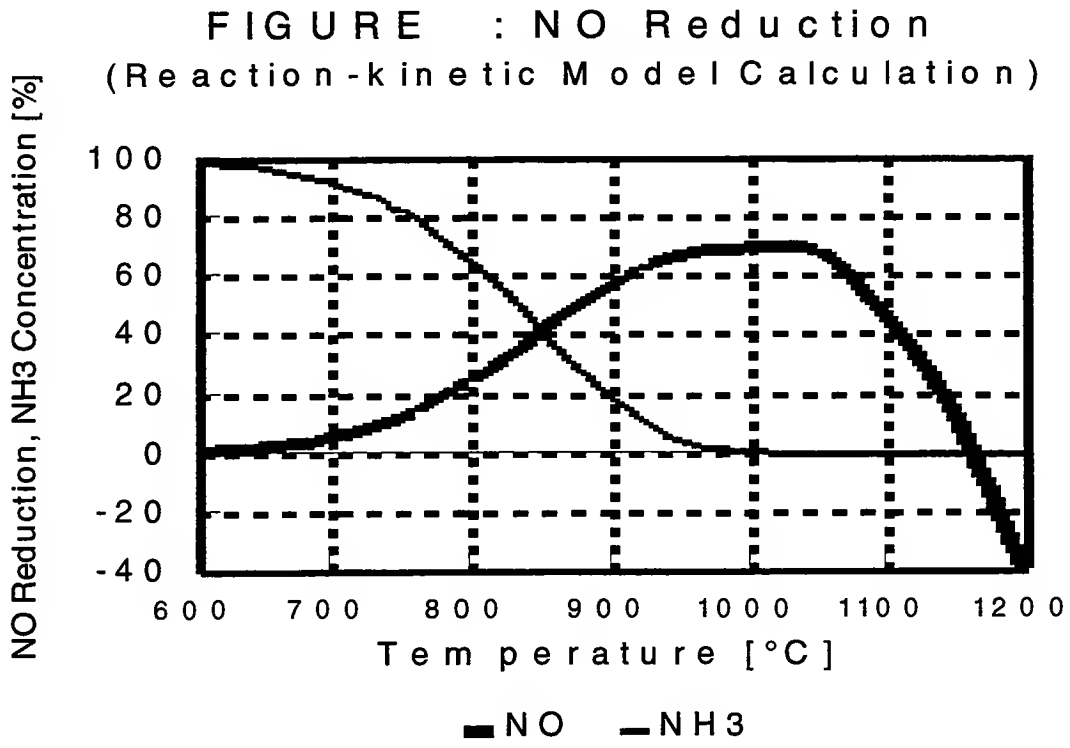
The Influencing Factors

Temperature Window

Reduction of nitrogen monoxide (NO), defined as the decrease in NO relative to the original NO, is a function of the temperature where the ammonia is injected and the reaction is made. As Fig. 13 shows, a high NO reduction is only achieved in a narrow temperature band. This is generally referred to as the "temperature window". In a normal flue gas from a cement kiln the maximum reduction of nitrogen monoxide (NO) is attained at about 950°C.

In a cement kiln at the most the temperature zone between 900 and 1100°C is technically "accessible". This means that the temperature zone is located in the preheater and this is where the ammonia can be injected. For this injection only the "lower" (left-hand) part of the NO reduction curve in Fig. 13 can be used.

Figure 21: NO-Reduction (Reaction-kinetic Model Calculation)



Dwell Time

The formation of the NH_2 radical from the ammonia (equation [?]) is dependent on the time. When ammonia is injected in to the cement process, this dwell time is fixed by the process. In any case the injection points in a cement kiln must be chosen that this dwell time is as long as possible.

Ammonia Slip

The proportion of injected ammonia that does not decompose to NH_2 radicals is called the NH_3 slip. This proportion leaves the reaction zone and travels with the flue gas to colder regions, during which a small proportion can decompose to NH_2 radicals. The

greater part of the slip, though, will remain stable as ammonia and, following various adsorption processes in the preheater, the cooling tower, the raw mill, or the electrostatic precipitator, will finally be emitted through the chimney as gaseous ammonia or adsorbed by aerosols.

Because only the "lower" (left-hand) part of the NO reduction curve in Fig. 2 can be used, in any case a noticeable NH_3 slip in this temperature zone occurs.

Injection Rate

The nitrogen monoxide reduction rate rises with increased NH_3 injection rate, but work tests have shown that a greater injection rate will produce a greater ammonia slip. In cement kilns ammonia should only be injected up to a mol ratio of 1.5 at the most, or in extreme cases 2.

$[NH_3]/[NO_0] < 1.5$ $\qquad$ $NO_0 \rightarrow N\approx$ without reduction

Form of Injected Ammonia

Pure Ammonia

Ammonia is already fluid at low pressures (15°, 7.28 bar). This substance can therefore be stored in a pressurized tank and pumped as a liquid. The entire ammonia system must be dimensioned for a pressure of 20 to 30 bar, which is certainly not optimal and can cause problems with the permission of such a tank in a cement plant.

Ammonia Water

For an ammonia injection in a cement kiln ammonia water (NH_4OH solution) should be used. In this solution the greater part of the ammonia is present in hydrated form as NH_3H_2O and only a very small proportion has dissociated to ammonium (NH_4^+) and hydroxide ions (OH^-). When heated rapidly from ambient temperature to the reaction temperature, the solution decomposes into water (H_2O) and ammonia (NH_3). Thus ammonia water has the same effect as pure ammonia but the problems with transport, storage, handling, etc. are much less.

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Alternatives to Ammonia

It is postulated and proved by works tests that the reduction of the nitrogen monoxide (NO) is effected via the NH_2 radical. This means that all substances capable of supplying an NH_2 radical may be used for reduction of NO.

A component of this kind is, for instance, urea $CO(NH_2)_2$. This substance decomposes approximately in the same temperature zone, thereby producing NH_2 radicals but unfortunately also a $CONH_2$-radicals. That means urea decomposes according to reaction [4] and not according to reaction.

$$CO(NH_2)_2 \rightarrow CO + 2(NH^{\bullet}_2) \quad (15)$$

$$CO(NH_2)_2 \rightarrow CO + NH^{\bullet}_2 + NCONH^{\bullet}_2 \quad (16)$$

About the behavior of the $CONH_2$-radical up to now nothing is know, but it is possible that cyanide compounds are produced in the kiln system.

6.4 Effectiveness of NOx Reduction Measures [9]

Figure 22: Comparison of NOx Control Technologies for Cement Kilns

Control Technology	Technical Feasibility	Potential NOx Reduction	Relative Cost	Effect on Clinker Quality	Effect on Other Emissions
CM	Medium (both kiln types)	15 - 30%	Low	May be adverse	CO, THC, SO_2 may increase
LNB	High (both kiln types)	15 - 30%	Low	• Improved • May vary with installation	• CO, SO_2 may increase • may vary with installation
SAC	High (precalciner kilns)	20 - 50%	Low	None	CO, THC may increase
SNCR	Medium (precalciner kilns)	40 - 70%	Medium	None	Potential for NH_3, PM_{10} emissions
NSNCR	Medium (all kilns)	20 - 30%	Low	None	CO, THC, SO_2 may increase

CM = Combustion Modifications
LNB = Low NOx Burner
SAC = Staged Air Combustion
SNCR = Selective Non-Catalytic Reduction
NSNCR = Non-Selective Non-Catalytic Reduction

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7. EXAMPLES OF NOX EMISSION REDUCTION

A collection of short descriptions of plants and their measures against NOx emissions is enclosed. Everybody is welcome to contribute to this collection.

NO_x emission reduction

Plant:	Siggenthal (SG)
Capacity:	2000 t/d (design), 1900 t/d (standard)
Fuel type	Fuel oil, dried sewage sludge, complete tires [1]
Plant description:	Kiln with 4-stage preheater
Supplier/equipment:	Polysius kiln and preheater (Dopol), Fuller grate cooler
Initial NO_x emission problem:	Very high NO_x emissions of up to 2800 mg/Nm^3_{dry} (in direct operation) were reported in 1987. (LRV-limit = 1500 mg/Nm^3_{dry}, since 1991 LRV-limit = 800 mg/Nm^3_{dry}) [LRV = Swiss clean air act]
Solutions investigated:	In 1988 trials to reduce the NO_x emissions by injection of Ammonia-water in the raiser duct were carried out. Reductions of up to 46% were reached. The problem of this reduction method was the high NH_3 emission due to the NH_3 in the raw meal, which was already higher than the emission limit. So any small additional NH_3 amount would not be acceptable. [3] Using a Pyrojet low-NO_x burner, the emissions in 1990 were around 1500 mg/Nm^3_{dry}. [2] SG had a project to bum sewage sludge containing Hg and at the same time the NO_x and SO_2 emissions where much too high. To solve these problems, a 3-stage waste gas cleaning system was foreseen. In 1990 the system was successfully tested in a pilot plant with an Ammonia-water injection into the riser duct as well as an active coke filter. Using this SNCR, NO_x could be reduced to N_2 and H_2O with ammonia injection. In these trials a reduction of the NO_x emissions by more than 60% and hence below 800 mg/Nm^3_{dry} was reached.
Solution realised:	Based on the experiences of the pilot plant, a denitrification system, based on SNCR was built. This system is followed by the kiln EP and a POLVITEC (Polysius Environmental Technology) active coke scrubber, which is mainly filtering out SO_2 and Mercury but also other pollutants like heavy metals, eventually occurring NH_3 from the denitrification and organics. The whole system was successfully commissioned in 1994. Investment cost: 30'000'000 CHF Operating cost: 3.3 CHF/t cli
Emissions reached:	< 800 mg/Nm^3_{dry} [4]
Literature:	[1] ATR, Annual Technical Report 1994,"Holderbank" [2] Stenger, Dr. R., Informationsveranstaltung der PCW Siggenthal zum Thema Klärschlamm-Verbrennung / Abgas -reinigung, HMB Bericht MA 92/93/D, VA-Dok: SG, Reg. 2 [3] Waltisberg, J., Die Reduktion der NO_x-Emission durch Eindüsung von Ammoniak in den Vorwärmer, HMB Bericht MA 88/10686/D, VA-Dok: SG, Reg. 2 [4] Effektive Emissionsminderung bei Einsatz von Sekundärbrennstoffen im Zementdrehofen von "HCB Siggenthal, Dr. D. Rose, Dr. L. Brentrup, Krupp Polysius

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Figure 22: Integration of a POLVITEC-Filter in a cement plant

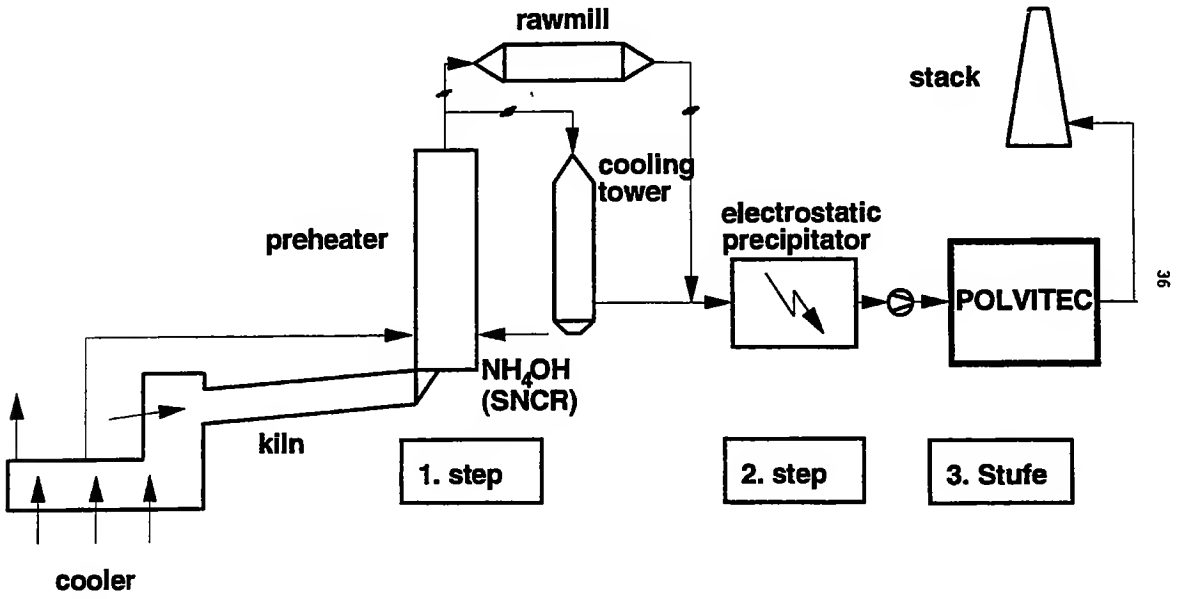
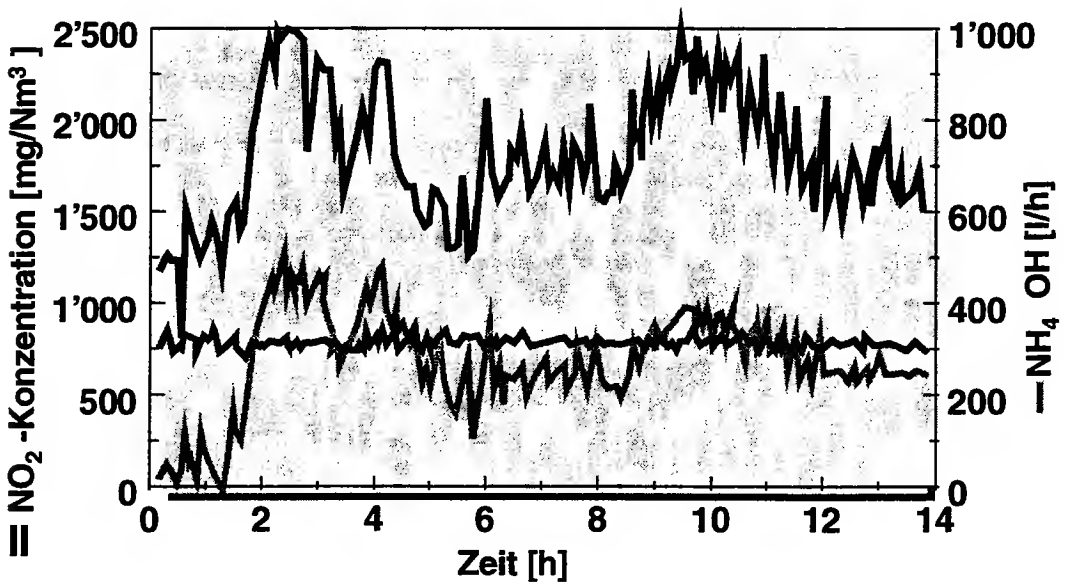


Figure. 23: Effect of SNCR of NOx at the Siggenthal plant. The NOx emission is controlled to below 800 mg/Nm$^3_{dry}$



NO_x emission reduction

Plant:	Untervaz (UV)
Capacity:	1900 t/d (standard)
Fuel type:	Coal, fuel oil; alternative fuels as dried sewage sludge, waste oil, distillation residue, plastics [1]
Plant description:	Kiln with 4-stage-suspension preheater and planetary cooler (kiln III)
Supplier/equipment:	KHD kiln, preheater and cooler
Initial NO_x emission problem:	In January 1985 NO_x emissions of 770 mg/Nm^3_{dry} were measured. After changing the original Pillard burner to a new Pyrojet burner, and additional changings on the lining (dam ring), in June 1985 a NO_x emission 360 mg/Nm^3_{dry} was reported. [2]. In this time, only coal was used as fuel. Later, using a mixture of coal and oil, the flame was not any more as stable as before, and the NO_x emission became higher. During measurements in 1989 for the office of environment the following NO_x emissions were measured: [3] 880 mg/Nm^3_{dry} in compound operation 950 mg/Nm^3_{dry} in direct operation (LRV-limit = 1500 mg/Nm^3_{dry}, since 1991 LRV-limit = 800 mg/Nm^3_{dry}) [LRV = Swiss clean air act]
Solutions investigated:	In 1990 the reduction of NO by injection of Ammonia-water or urea was tested at different locations in the preheater tower. By injection of Ammonia-water in the raiser duct on stage 4 of the heatexchanger, a NO reduction of up to 42 % of the initial 840 mg/Nm^3_{dry} could be achieved [4]. After these short trials, a further test to add Ammonia-water to the preheater has been carried out in 1991. The achieved reduction was about 50 %, depending also on the ratio NH_3/NO, i.e. approx. to 400 - 500 mg/Nm^3_{dry}. Compared to the addition of urea, the achievable reduction with Ammonia-water was twice as high.[5] The influence of water in the waste oil was investigated in 1992. Without negatively influencing the temperature of the sintering zone, a maximum of 5.7 g H_2O/kg clinker could be injected, which resulted in a NO-reduction of 20 - 25%. [6] In 1993 a test to reduce NO selective and non-catalytic with pure hydrogen was carried out at temperatures between 600 and 1600 °C, without success. [7]
Solution realised:	Actually the NO_x emissions are kept at a low level due to the low-NO_x-burner and the water contained in the waste oil and in distillation residues. Additionally the Linkman control as well as a rather high CO concentration and the dusty atmosphere in the burning zone due to the planetary cooler contribute to a low NO-level.
Emissions reached:	In spring 1993 the following NO emissions of the kiln (calculated as NO_2) have been measured: [8] 540 mg/Nm^3_{dry} in compound operation 600 mg/Nm^3_{dry} in direct operation The averaged values of 1994, measured after the heatexchanger, were in the same range.
Further Optimisation:	Is actually not necessary. If ever it would be necessary, the possibility of adding Ammonia-water to the preheater could be envisaged again.

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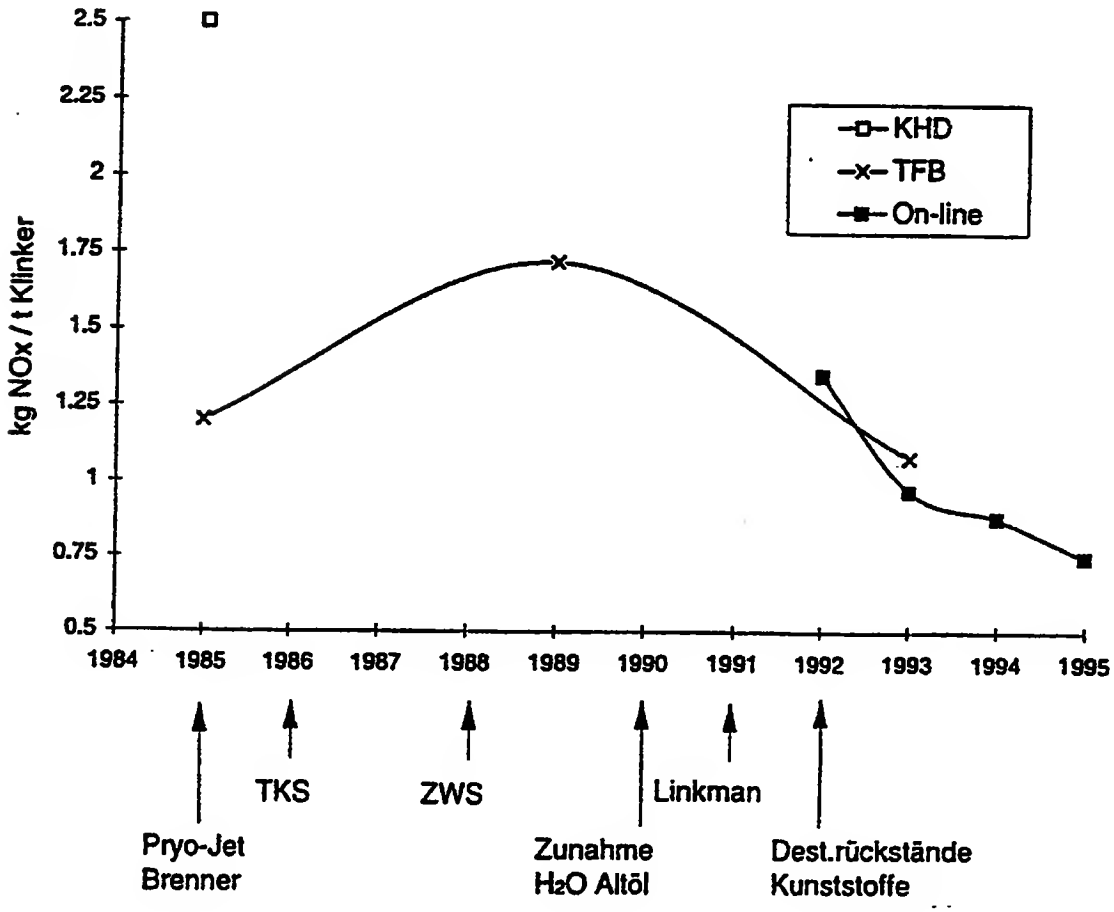
Figure 24: NOx-Emissions BCU (kiln #1)



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NO_x emission reduction

Plant:	Rekingen (RK)
Capacity:	2'200 mtpd (design), 2130 mtpd (standard)
Fuel type:	Coal, fuel oil, waste timber
Plant description:	4-stage-suspension preheater kiln
Supplier/equipment:	Polysius kiln and preheater
Initial NO_x emission problem:	NO_x emissions used to be very high in the past, and have not frequently been measured. In about 1984, the systematic observations and measurements of these emissions was started. As an example, in 1985 NO_x emissions of about 1800 mg/Nm^3 (based on 3 % O_2) were measured. [1] (LRV-limit = 1500 mg/Nm^3_{dry}, since 1991 LRV-limit = 800 mg/Nm^3_{dry}) [LRV = Swiss clean air act]
Solutions investigated:	Initially the influences of different parameters on the existing plant have been investigated, as for example the temperature of the burning zone, the amount of primary- and secondary air, coal as well as raw meal [1]. In 1990 the actual NO_x emissions were reported as 1'200 mg/Nm^3_{dry} in direct operation and 950 mg/Nm^3_{dry} in compound operation. These values have been below the actual valid LRV-limit of 1'500 mg/Nm^3_{dry}. Due to a regulation of the state of Aargau, a further reduction of the total NO_x-emissions from 1200 t/year to about 300 t/y was required. It was expected to reach this reduction by installing a new Pyrojet burner (awaited REDUCTION OF ABOUT 30 - 40 %) and a additional denitrification based on SNCR (awaited reduction of about 60%) [2]. Through the installation of a new Pyrojet low-NO_x-burner in 1990, the NO_x-emissions were reduced to around 800 mg/Nm^3_{dry}. The denitrification has then first been investigated in trials, by adding liquid ammonia or urea to the preheater. The best results in the trials have been achieved, adding liquid ammonia into the raiser duct between preheater stage 3 and 4. Depending on the NH_3/NO ratio NO_x-reductions of up to around 50% could be detected [3] Due to unstable burning characteristics (design problem) which lead to higher NO_x emissions, the Pyrojet low-NO_x burner was exchanged in 1993 again to a Pillard 3 -channel burner (same type as earlier).[4] With this burner NO_x emissions of 720 mg/Nm^3_{dry} could be reached.
Solution realised:	In 1994 a 4-channel Rotaflam burner was installed, mainly to be able to use waste timber as alternative fuel.
Emissions reached:	With the Rotaflam burner the NO_x emissions were reduced by 10 20 % to an average value of 650 mg/Nm^3_{dry}.

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NO_x emission reduction

Plant:	Hardegsen (HD)
Capacity:	1'086 mtpd (standard)
Fuel type:	Coal (>98%), fuel oil, natural gas, landfill gas [1]
Plant description:	4-stage-suspension preheater kiln with grate cooler
Supplier/equipment:	MIAG kiln and Fuller cooler
Initial NO_x emission problem:	Until 1989 the NO_x emissions using a Pillard VR-K3 three chanel burner were in the range of 1300 mg/Nm^3_{dry} (based on 10 % O_2) [2,3]. (TA Luft-limit = 1800 mg/Nm^3_{dry}, since 1991 the limit is 800 mg/Nm^3_{dry} for old plants and 500 mg/Nm^3_{dry} for new plants.)
Solutions investigated:	Different low-NO_x burners were investigated. Finally the decision for a Pillard Rotaflam burner was influenced by the lower investment costs, the use of the existing pipe lines and the entire burner suspension system and the fact, that one of the existing fans could be reused.
Solution realised:	In 1990 a Rotaflam low-NO_x burner was installed and is operated since then with good experience.
Emissions reached:	The NOx emissions could be reduced by about 22% under comparable conditions to 1004 mg/Nm^3_{dry} in the acceptance tests.

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J. Waltisberg, HMB, Verfahrenstechnische Abteilung; "Holderbank" NEWS 2/91; 1991

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Sources and Reduction of SO_2-Emissions

P. Kutschera

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Summary:

The SO_2 emission of a cement plant depends on the

- type of the kiln
- quality of the raw material and fuel
- operating conditions of the kiln and the raw mill system
- secondary SO_2 reduction measures

To reduce the SO_2 emission from a pyroprocessing system, three basically different methods are available:

- Maintain the existing process while reducing the sulfur input into the system
- Modify the existing process (primary reduction measures)
- Maintain the existing process while adding a separate gas cleaning unit for the exhaust gas (secondary reduction measures)

With today's technology it is possible to reduce the SO_2 emission to an acceptable level. The main methods are SO_2 absorption with $Ca(OH)_2$ in the preheater or in a separate scrubber and adjustments of the raw material.

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1. INTRODUCTION

SO_2 was one of the first air pollutant with which the public made very negative experiences. Mainly SO_2 but also other pollutants from coal fires in the city of London were responsible for the death of 4000 people between December 5 and 9, 1952.

Today, the SO_2 emission is known to be responsible for a part of the acid rain which is attacking plants and animals, for increased pH of lakes and for corrosion and decomposition of buildings.

Today, the total worldwide SO_2 emission is declining because of reduced sulfur content in the fuels and scrubbers for large SO_2 sources like coal or fuel fired power stations.

The SO_2 emission of some cement plants is still on the high side and should be reduced in the future. The formation of SO_2 in kiln systems and methods to reduce them are presented hereafter.

2. CHEMICAL AND PHYSICAL PROPERTIES AND ENVIRONMENTAL ASPECTS OF SOME SULFUR COMPOUNDS

Sulfur (S)

Sulfur is the 15th most common terrestrial element and the 9th most abundant element in the universe. It is widely distributed in nature in different compounds as well as in pure form. The main sulfur content of the lithosphere is estimated to be approx. 0.05%.

Physical Properties [1]

The melting / solidification point of normal sulfur is 119.3 - 114.5°C, depending on the mode of heating.

Density of solid at 20°C, kg/m^3
Rhombic α 2070

Melting point, °C, Natural
Rhombic α 110.2

Density of liquid, kg/m^3
115°C 1808
445°C 1614

Boiling point, °C 444.6

Density of vapor /
density of air, 470°C 1.837

Chemical Properties

Molecular weight: 32,1 g/mol

Sulfur is one of the most reactive elements; it reacts directly with most elements except iodine, gold, platinum and the noble gases. In humid air it is weakly oxidized, forming traces of sulfur dioxide and sulfurous acid. At approx. 250°C, sulfur ignites in air and burns with a blue flame but the presence of sulfur dioxide significantly raises the ignition point.

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Pyrite / Marcasite (FeS_2)

Common and important iron and sulfur mineral

Physical Properties [13], [2]

	Pyrite	Marcasite
Cristalline form	cub.	rhomb.
Density t/m^3	5	4.87
Melting point °C	1'171	450
Solvent	HNO_3	HNO_3

Chemical Properties [2]

	Pyrite	Marcasite
Molecular weight g/mol	119.98	119.98
Ignition temperature °C	350 - 550	
Complete combustion °C	850 - 940	

Four main reaction steps (Eqs. 3 - 6) make up the overall pyrite roasting reaction (represented by Eq. 7). The specified enthalpy values [7] refer to 298 K:

$2\ FeS_2$	$\rightarrow$	$2\ FeS + S_2\ (g)$	ΔH	=	+ 293 kJ/mol	(1)
$S_2\ (g) + 2\ O_2$	$\rightarrow$	$2\ SO_2$	ΔH	=	- 723 kJ/mol	(2)
$2\ FeS + 3\ O_2$	$\rightarrow$	$2\ FeO + 2\ SO_2$	ΔH	=	- 948 kJ/mol	(3)
$2\ FeO + 0.5\ O_2$	$\rightarrow$	Fe_2O_3	ΔH	=	- 282 kJ/mol	(4)
$2\ FeS_2 + 5.5\ O_2$	$\rightarrow$	$Fe_2O_3 + 4\ SO_2$	ΔH	=	- 1660 kJ/mol	(5)

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Sulfur Dioxide (SO_2)

Sulfur dioxide is produced industrially in greater quantities than any other single sulfur compound. It is generated as the first stage in the manufacture of virtually all the sulfuric acid used by industry, irrespective of the basic raw material.

Physical Properties [1]

Sulfur dioxide, SO_2, is a colorless, non-flammable, toxic gas with a characteristic pungent smell and acid taste.

Melting point (101.3 kPa)	- 75.5°C
Dynamic viscosity at 0°C	368 Pas
Critical temperature	157.5°C
Boiling point (101.3 kPa)	- 10.0°C
Latent heat of vaporization (at *bp*)	402 J/g
Standard density at 0°C (101.3 kPa)	2.93 kg/m^3
Standard enthalpy of formation	- 4636 J/g

Specific heat capacity c_p (101.3 kPa):

0°C 586 J kg $^{-1}$ K $^{-1}$

500°C 816 J kg $^{-1}$ K $^{-1}$

Chemical Properties [1]

Molecular weight: 64.06 g/mol

Sulfur dioxide is very stable; thermal dissociation becomes significant only above 2'000°C. It can be decomposed by shock waves, irradiation with ultraviolet or X rays or by electric discharge.

The reaction of sulfur dioxide with oxygen to form sulfur trioxide is industrially the most significant of all its reactions because of its importance in sulfuric acid production. In the gas phase, it only takes place at elevated temperatures and, for a satisfactory yield of sulfur trioxide, it requires the presence of a catalyst. In aqueous solution, sulfur dioxide is oxidized to sulfuric acid at low temperature by air in the presence of activated coke or nitrous gases, or by oxidizing agents such a hydrogen peroxide.

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Environmental Aspects and Toxicology [1]

A substantially larger amount of sulfur dioxide than utilized industrially is produced by the combustion of sulfurous fossil fuels and is discharged into the atmosphere with the flue gases. The calculated total emission of sulfur dioxide from power stations, traffic, households, industry and trade in the Federal Republic of Germany in 1975 amounted to approx. 3.6×10^6 t. In contrast, only about 2.7×10^6 were used for that year's sulfuric acid production of approx. 4.2×10^6 t H_2SO_4.

In the last years the SO_2 emissions from power plants in Germany were reduced continuously with the installation of flue gas desulfurization plant and with the utilization of low-sulfur fuels, but emissions from traffic, households, industry, etc. in 1990 still account for 620×10^3 t/a. Emissions from power stations amounted to 320×10^3 t/a.

For worldwide atmospheric emissions, only rough estimates are available. For 1970, while global emissions were estimated at 157×10^6 t SO_2, only about 61×10^6 t were consumed in the production of approx. 94×10^6 t of sulfuric acid. The worldwide cement production (1985) is estimated to about 700×10^6 t/a. Based on an average SO_2 concentration of about 1000 mg/Nm^3, the portion of SO_2 emitted by cement plants compared to the total anthropogenetic SO_2 emission (200×10^6 t/a) is about 0.7% [5].

Large-scale emission of sulfur dioxide close to ground level has indisputably been the cause of some spectacular environmental problems in the past. A large area around Sudbury, Ontario, was completely sterilized by sulfur dioxide from primitive ore roasting operations around the turn of the century and much of it is still barren as a result of the ensuing soil erosion. Calamitous sulfuric acid fogs occurred in Donora, Pennsylvania, and London, England, in 1948 and 1952, when adverse climatic conditions prevented sulfur dioxide from industrial sources and domestic coal fires from dispersing.

The causes of these episodes were correctly identified and the action taken to control **ground level smoke** and sulfur dioxide concentrations has effectively prevented them from recurring. In the case of industrial sources, however, this has often been achieved by merely building taller chimneys to disperse the sulfur dioxide over a wider area and as the consumption of fuels has increased over the years it has been argued that the problem has merely been shifted from the location where the sulfur dioxide is produced to other locations downwind.

Considerable publicity has been given to the apparently increasing problem of **acid rain** and there does, indeed, seem to be a correlation between the incidence of acid rainfall and the atmospheric transport of pollutants from major industrial locations.

Establishing in any degree of certainty to what extent sulfur emissions are responsible for acid rain is, however, complicated by incomplete knowledge of the magnitude of sulfur emissions from natural sources, the atmospheric chemistry of sulfur and the importance of other potential acidulates such a nitrogen oxides.

Nevertheless, the evidence linking sulfur dioxide pollution to acid precipitation has been mounting gradually. A recent report by the U.S. National Academy of Sciences, for example, has stated that there appears to be a direct proportionate relationship between sulfur dioxide pollution and the amount of acidic sulfates in precipitation.

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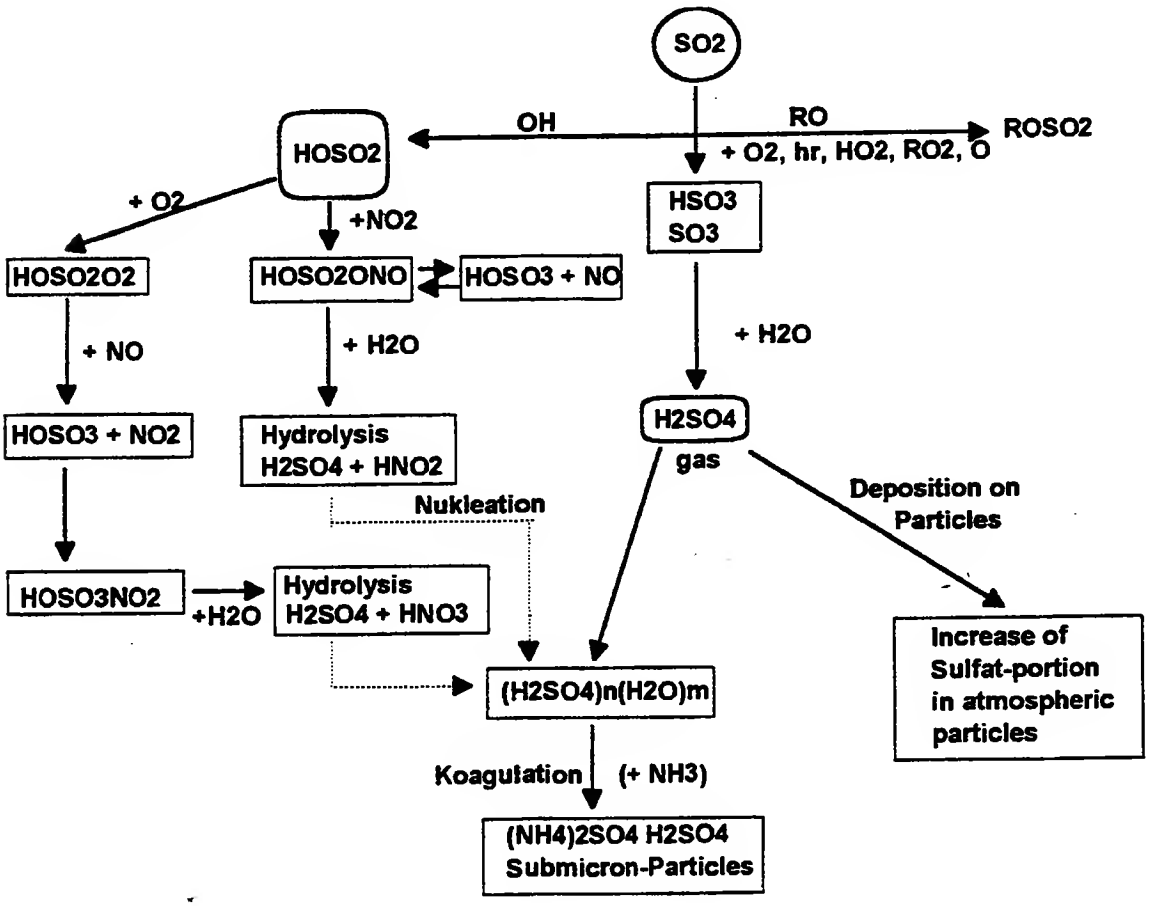
Ill-effects of sulfur dioxide on humans and animals are mainly related to irritation and damage of the moist mucous membranes by the formation of sulfurous acid. The odor threshold of sulfur dioxide in air is between 0.3 and 2.5 ppm. In most human beings, concentrations of 5 - 10 ppm will lead to irritation of the respiratory tract; in sensitive people they may produce spasms of the bronchi. Higher concentrations will cause heavy irritative coughing, while breathing sulfur dioxide at concentrations above 400 - 500 ppm, even for only a few minutes, is dangerous to life. The maximum workplace concentration is defined as 5 ppm (14 mg/m^3).

Long-term exposure may possible lead to bronchopneumonia and, in extreme cases, to toxic pulmonary edema with dyspnoea, cyanosis and cardiac-circulatory failure.

Sulfurous acid absorbed by the body is converted to sulfate and discharged in the urine, as evidenced by an increase in urine acidity.

Plants are impaired by sulfur dioxide at even lower concentrations than human beings. Sulfur dioxide levels of 1 - 2 ppm are enough to cause acute damage to the leaves of plants in only a few hours, followed by necrosis, probably resulting from impaired photosynthesis.

Atmospheric Chemistry of SO_2 [3]



Sulfuric Acid

Physical Properties

Pure sulfuric acid H_2SO_4 is a colorless, water-white, slightly viscous liquid, *mp* 10.4 C, *bp* 279.6°C. It can be mixed with water in any ratio.

Chemical Properties

Molecular weight: 98.08 g/mol

Sulfuric acid is a strong acid with characteristic hygroscopic and oxidizing properties. Sulfuric acid, like the sulfate ion, is chemically and thermally very stable.

$SO_2 + 1/2\ O_2$	$\rightarrow$	SO_3	$\Delta H°$	=	- 99.0 kJ	(6)
SO_3 (g) + H_2O(l)	$\rightarrow$	H_2SO_4 (l)	$\Delta H°$	=	-132.5 kJ	(7)

Environmental Aspects and Toxicology [1]

Sulfuric acid has a highly corrosive effect on the eyes, the mucous membranes and the skin, even in low concentrations. Because it completely destroys living tissue, concentrated sulfuric acid causes burns that penetrate deeply and heal only slowly. Swallowing sulfuric acid produces extreme pain in the digestive tract, vomiting and shock and there is a danger of perforation.

Sulfuric acid vapors or mists irritate the eyes and the mucous membranes of the nose, pharynx and respiratory tract, causing heavy coughing and breathlessness.

Sulfuric acid releases are highly deleterious with respect to ground and surface waters. The substance is toxic to both fish and algae, both directly and as a result of reaction with other materials in the water. Any concentration ≥ 1.2 mg/L is considered lethal to fish: 6.3 mg/L or more causes death within 24 h.

Sulfur Trioxide (SO_2)

Physical Properties

Gaseous SO_3

Nominal density, g/L (0°C, 1013 mbar) 3.57

Specific heat Cp, kJ m^{-3} K^{-1}
100°C 2.543
500°C 3.191

Liquid SO_3

Density, g/cm^3 (25°C) 1.9

bp, °C (1013 mbar) 44.8

Heat of evaporation (boiling point), J/g 538

Vapor pressure, bar
20°C 0.26
100°C 8

Critical temperature, °C 217.7

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3. SO_2 EMISSION LIMITS / NORMAL EMISSION

Emission Limits

Germany	400 mg/Nm^3 dry (no waste fuels)
Switzerland	500 mg/Nm^3 dry
Europe	100 - 2400 mg/Nm^3

Normal Emission

There is no normal SO_2 emission level for cement kilns. The SO_2 emission depends mainly on quantity and quality of "S" input, kiln system and SO_2 reduction systems. It is known that SO_2 emission of cement kilns can be between very low values of less than 50 mg/Nm^3 and very high values up to 3500 mg/Nm^3.

Figure 1 Average SO_2 Emission according to a "Holderbank" Survey

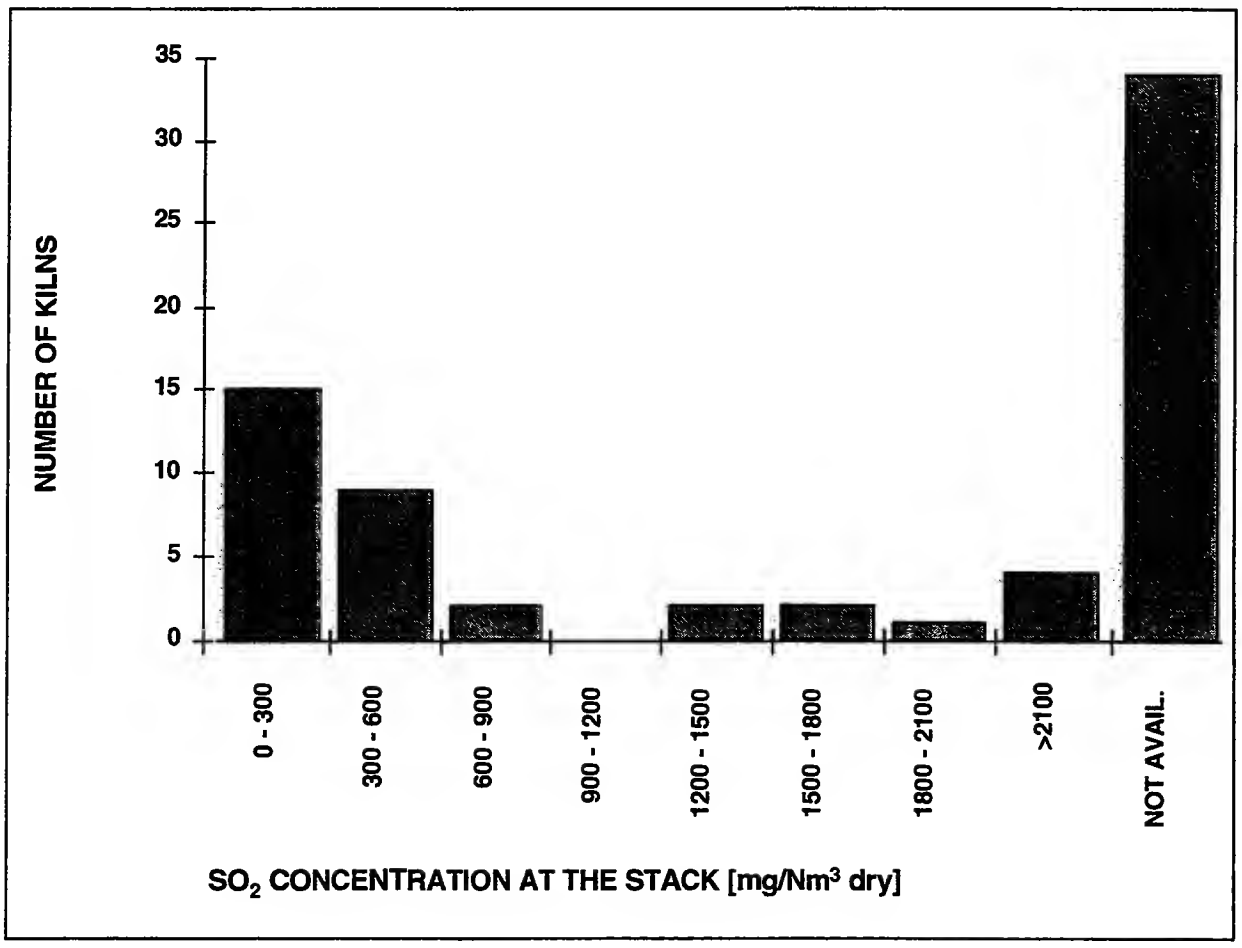
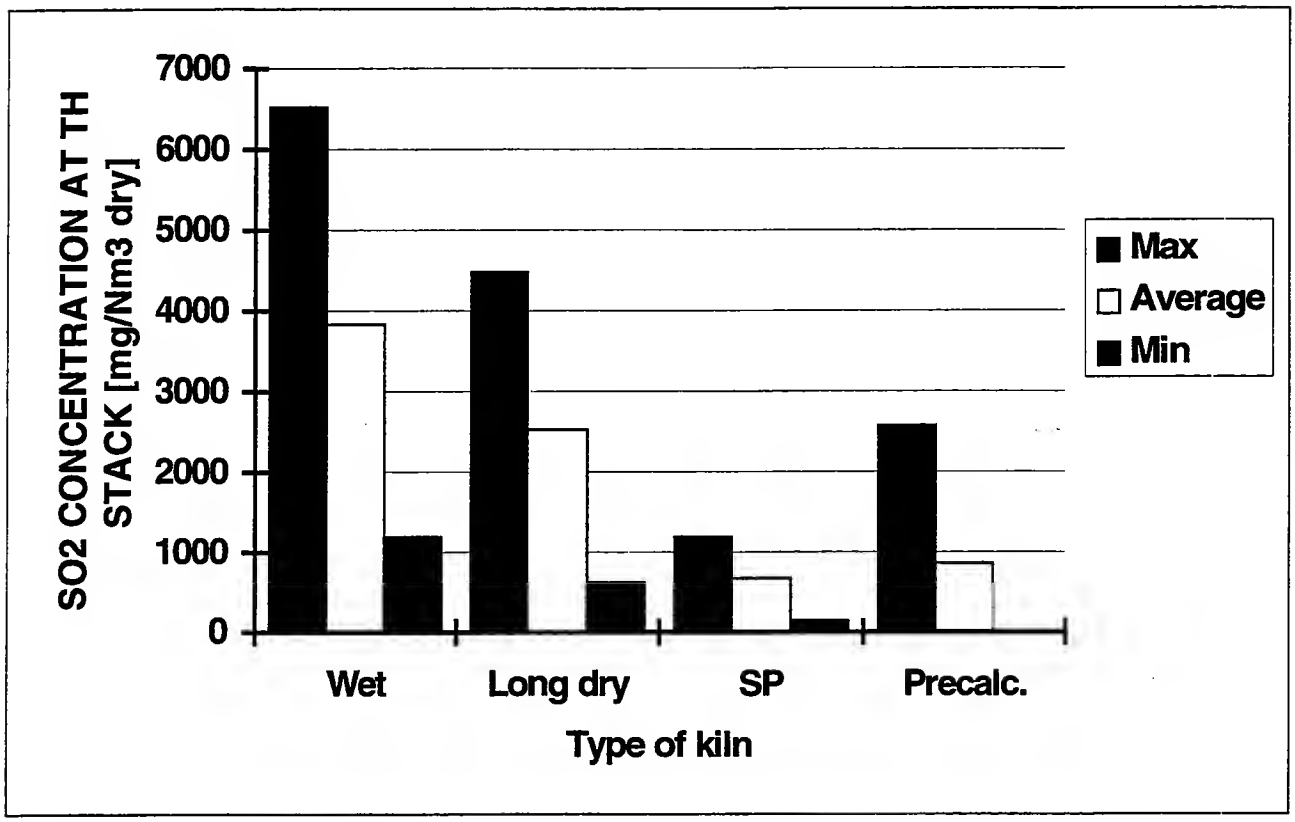


Figure 2 Result of PCA SO_2 emission survey 1982 [4]



Assumption: wet: 2.1 Nm^3 dry / kg cli (18 kilns)
long dry: 1.5 Nm^3 dry / kg cli (12 kilns)
SP: 1.4 Nm^3 dry / kg cli (5 kilns)
Precalciner: 1.4 Nm^3 dry / kg cli (12 kilns)

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4. SULFUR INPUT INTO KILN SYSTEM [4] [6]

Contained in the raw materials (mainly as FeS_2) and the fuel, sulfur enters the process mainly in the form of sulfates (gypsum $CaSO_4$, 2 H_2O), sulfides (Pyrit: FeS_2) and organic sulfur compounds. In the process, the sulfur compounds may either be reduced or oxidized to form gaseous SO_2.

Table 1: Sulfur content of kiln feed and fuels of the "Holderbank" Plants in 1994

	% SO_3		
	Average	Minimum	Maximum
Kiln Feed	0.46	0	1.93
	% S		
Fuels	Average	Minimum	Maximum
Coal	0.86	0.02	3.51
Pet Coke	4.22	1.01	8.3
Diesel Oil	2.61	0.02	3.5
Heavy Fuel Oil	2.86	0.12	11.6
Alternative Fuels	Average	Minimum	Maximum
Liquid	0.56	0.11	2.01
Solid	0.92	0	3.9
Natural Gas	0.52	0	3.17
Tires	1.63	0.8	3.9
Waste Oil	1.16	0.5	2

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5. BEHAVIOR OF S-COMPOUNDS IN THE PROCESS

Wet Kilns

A schematic representation of the sulfur circulation in a wet kiln is shown in fig. 3. Depending on the raw materials, the conditions in the kiln burning zone and the internal circulation, 30 - 80% of the sulfate compounds entering the kiln burning zone may decompose and form SO_2 which leaves the kiln burning zone with the exhaust gas together with the SO_2 formed by oxidation of the organic sulfur in the fuel fired into the kiln.

On passing through the calcining zone, the preheating and the drying sections of the kiln, some of the SO_2 will be reabsorbed in the raw materials but due to the poor contact between the kiln gases and the raw materials, especially in the calcining zone, 40 - 60% of the SO_2 from the kiln burning zone will be emitted from the kiln.

In the preheating section of the kiln, the SO_2 concentration in the gas may even be increased if the raw materials contain sulfides which will burn here in accordance with equation:

$$\overset{+II\,-I}{4\,FeS_2} + 11\,O_2 \xrightarrow{>550°C} \overset{III-II}{2\,Fe_2O_3} + \overset{IV-II}{8\,SO_2} \qquad (8)$$

Having left the kiln, the exhaust gas usually only passes through a precipitator before being emitted to the atmosphere through the stack.

Depending on the type of precipitator between 10 and 30% of the SO_2 in the exhaust gas may be absorbed in the raw materials at this stage. Taking into account internal circulation and the evaporation and absorption rates mentioned, about 30% of the sulfur entering the kiln system will be emitted as SO_2. The emission will increase roughly in proportion to the total input of sulfur compounds in raw materials and fuel.

Since the specific heat consumption of wet kilns is high, SO_2 emission depends very much on the sulfur content in the fuel. If the excess air is reduced below a certain level, a sharp rise in SO_2 emission from the kiln may occur, as local reducing conditions will increase SO_2 formation in the burning zone and make SO_2 reabsorption in the kiln back-end more difficult.

Long dry kilns

Except for a possible slurry preheating section, the design of the long dry kiln is virtually the same as that of a wet kiln.

Consequently, the SO_2 absorption factors are comparable to those of wet kilns as mentioned above.

However, with the same type of raw material and fuel the SO_2 emission from the long dry kiln system will be lower than that from the wet kiln. This is due to the lower specific heat consumption and the fact that the exhaust gases from the long dry kiln are often used for drying in the raw mill in which 20% - 50% of the SO_2 might be absorbed by intimate contact with freshly ground raw meal particles.

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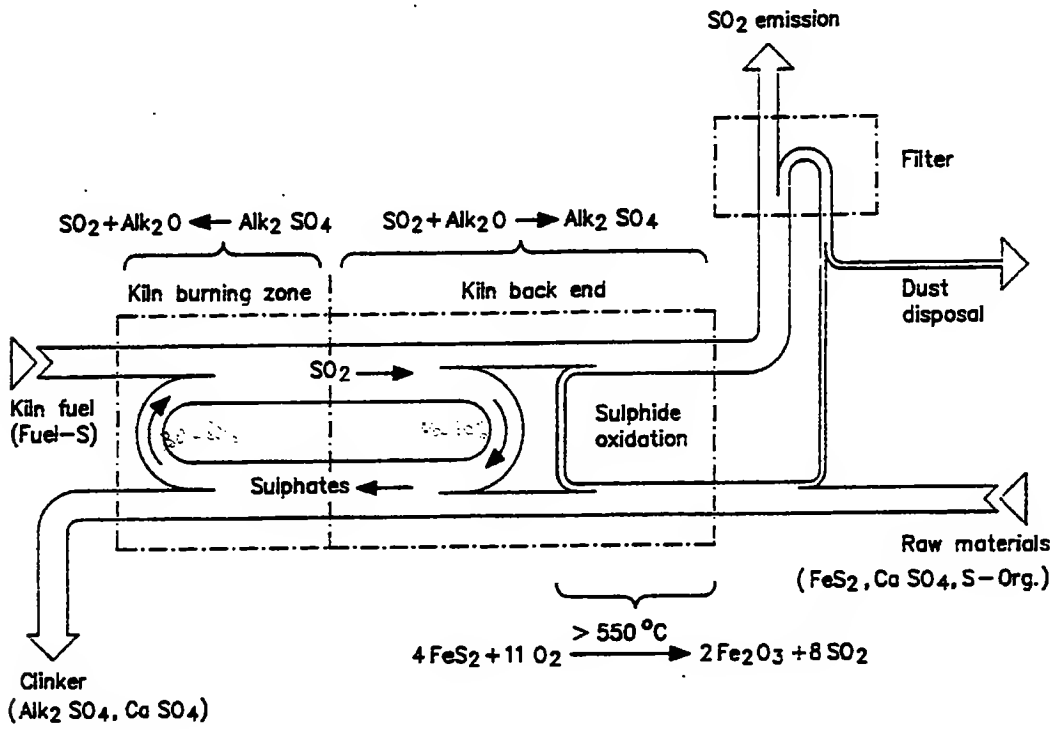
Figure 3 Behavior of sulfur in a wet or long dry kiln system without raw mill



Preheater kilns

A schematic representation of the sulfur circulation in a dry-process preheater or precalciner kiln with bypass is shown in fig. 4.

In 4 and 5-stage cyclone preheater kilns complete preheating of the raw meal takes place in intimate contact with the exhaust gas from the kiln and in the lower stages the temperature reaches 850°C at which point part of the raw meal starts to calcine.

The SO_2 coming from the kiln is thus brought into contact with free CaO and $CaCO_3$ at a temperature at which the following reactions proceed relatively fast:

$$\underset{\text{II-II}}{CaO} + \underset{\text{IV-II}}{SO_2} + \tfrac{1}{2}O_2 \longrightarrow \underset{\text{II IV-II}}{CaSO_4} \qquad (9)$$

$$\underset{\text{II IV-II}}{CaCO_3} + \underset{\text{IV-II}}{SO_2} \longrightarrow \underset{\text{II IV-II}}{CaSO_3} + \underset{\text{IV-II}}{CO_2} \qquad (10)$$

In this way, nearly all SO_2 formed in the kiln is absorbed by the hot meal and reintroduced into the kiln and bound into the clinker. Only if the sulfur circulation between the kiln and the lower preheater stage reaches extreme levels or in the case of local reducing condition in the kiln back-end and the riser duct SO_2 may escape via the lower stage. This will often be the case if coarse waste fuel (e.g. rubber tires or pieces) is fired into the riser duct or kiln inlet.

Except for this situation, considerable SO_2 emission from preheater kilns (without by-pass) will only occur in case the raw materials contain non-sulfates such as pyrites which form SO_2 already in the upper stage cyclones where the temperature and the concentration of free lime and alkaline materials are too low to ensure complete reabsorption of the SO_2 formed, according to above two reactions (9, 10).

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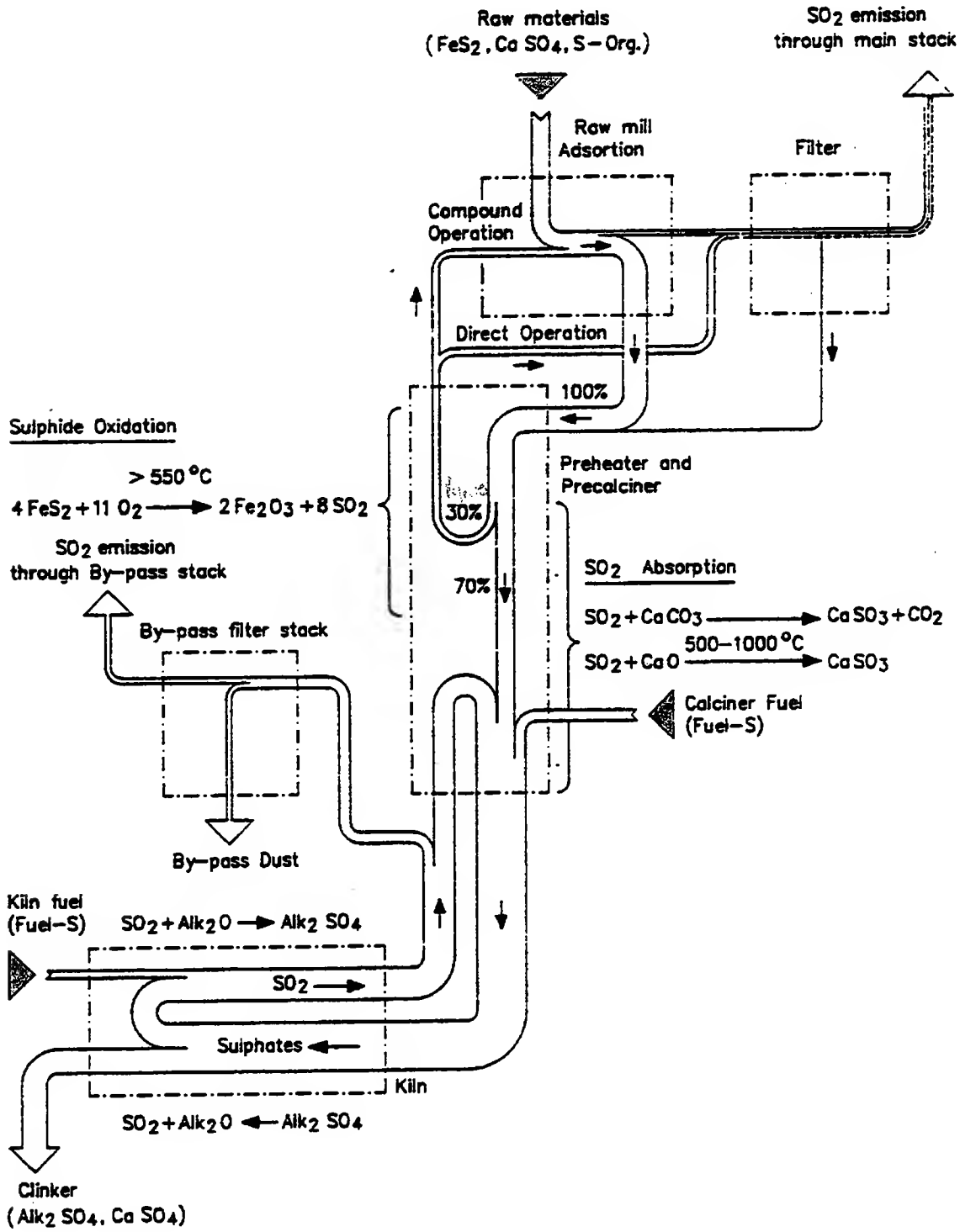
Normally, 30 - 50% of the sulfur present in the raw meal in the form of pyrites will leave the preheater as SO_2. Part of this is absorbed in the raw mill and the precipitator. Still, SO_2 emission from the stack will amount to 15 - 30% of the sulfur entering with the raw materials in the form of non-sulfates.

This means an SO_2 emission of 500 - 1000 mg / kg cli for each per mil sulfur present as non-sulfate in the raw materials.

Figure 4 Behavior of sulfur in a preheater kiln system



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Precalciner kilns

The precalciner kiln system in which the kiln gases pass the precalciner offers ideal conditions for absorption of SO_2 from the kiln due to the high amount of free CaO and a temperature of approx. 900°C in the calciner.

But also other precalcining systems where kiln gases enter into contact with sufficient free CaO in the kiln riser duct and the lower cyclones absorb SO_2 from the kiln.

The only source of SO_2 emission from the preheater of a precalcining kiln system is therefore any content of non-sulfates in the raw meal which will generate SO_2 in the upper preheater stages as explained above. Many precalciner kilns in the USA have a kiln gas bypass. On leaving the kiln riser duct the kiln gas is cooled by air dilution and/or water injection to 400 to 230°C (depending on the filter system). The bypass outlet is placed to ensure the lowest possible dust concentration in the bypass gas.

This means limited possibility of absorption of the SO_2 extracted with the kiln gases in the bypass duct. Very often, more than 50% of the SO_2 contained in the by-passed kiln gases will thus enter the atmosphere via the stack. This might explain why the average SO_2 emission from the precalciner kiln systems is higher than that from the SP kiln systems.

It should be borne in mind that the evaporation factor for sulfur components in the kiln and consequently the SO_2 emission from the bypass may increase drastically if coal combustion in the precalciner is incomplete. In this case, residual carbon from the calciner will enter the kiln inlet where it will create reducing conditions, resulting in greatly increased sulfur "evaporation". Accordingly, improving the combustion in the precalciner is a mean of reducing the SO_2 emission from a system with bypass.

6. SO_2-EMISSION REDUCTION POSSIBILITIES

To reduce the SO_2 emission from a pyroprocessing system to a certain controlled level, three basically different methods are available:

- Maintain the existing process while reducing the sulfur input into the system
- Modify the existing process (primary reduction measures)
- Maintain the existing process while adding a separate gas cleaning unit for the exhaust gas (secondary reduction measures)

6.1 Reduction of Sulfur Input into the System

Raw Material

A reduction of the total sulfur input or very important for preheater kilns the total input of sulfides is often possible. A sulfur-selective quarrying may not only reduce the total input of sulfur but also reduce its fluctuations. The maximum SO_2 retention capacity of a kiln system as described above is reached at the lowest SO_2 fluctuation. Because at very low sulfur input the absorption capacity of $CaCO_3$ or $Ca(OH)_2$ is not used completely whereas during very high sulfur input the SO_2 generation is over the absorption capacity of the raw meal. In some cases the SO_2 emission stays below limit if the sulfur content in the raw material is not fluctuating too much.

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In the case of the Höver plant in Germany, the desulfurization with $Ca(OH)_2$ injection into the preheater is working at its limit because of its very high sulfide input. As explained below, this method is only working up to an SO_2 emission of about 1000 mg/Nm^3 dry. If Höver would not have a very homogenous sulfur content in their raw meal it would not be possible to stay below the emission limit. Not even with $Ca(OH)_2$ injection.

The costs of selective quarrying or opening a new quarry should be compared with the costs of the installation and operation of a system for secondary reduction measures.

A selective elimination of sulfur in the raw material is not yet developed.

Fuel

Reduction of the sulfur input via the fuel can usually only be attained by accepting a considerable increase in fuel cost. Except for the case where the fuel can be replaced by alternative fuels.

A reduction of the fuel-sulfur will under normal conditions not reduce SO_2-emission from preheater kilns without bypass.

6.2 Modification of the Existing Process

Non-Preheater System → Preheater System

As documented and explained above, preheater systems do have the lowest SO_2- emission of all clinker production systems.

This modification is reducing the SO_2 emission significantly if:

- the SO_2 emission is not mainly caused by non-sulfates input via the raw material
- the total input of circulating elements (S, alkali, Cl) is low enough to avoid a kiln bypass

This modification is reducing the SO_2-emission because of:

- retainment of sulfates and fuel-sulfur by embedding sulfur in the clinker
- SO_2-adsorbtion during compound operation in the raw mill
- lower heat consumption

Modification of fuel preparation and feed [8]

A reducing atmosphere in the kiln is increasing the formation of CO and the volatility of "SO_2".

Therefore, the SO_2 emission of preheater kilns with bypass, wet and long dry kilns and to a lesser degree also of normal preheater kilns is increased if a reducing atmosphere is present in the kiln:

$$2\,CaSO_4 + C \xrightarrow{>1000°C} 2\,CaO + CO_2 + 2\,SO_2 \qquad (11)$$

This can be avoided by better fuel preparation and feed as well as with an increased O_2 concentration in the kiln.

Coal, Coke:

- no oversized particles (residues on 90 µm sieve = 0,5 x % volatiles)
- continuous feed (feeder, transport)

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Oil:

- good atomization: burner nozzle e.g. Pillard, Unitherm, temperature, pressure

Alternative fuels:

- solid: no oversized lumps: e.g. chipped instead of whole tires, small bundles
- liquid: see "oil" above

Besides fuel preparation and feed also the O_2 concentration in the kiln should be optimized to reduce the sulfur volatility. To control the O_2 in the kiln, a kiln inlet probe is recommended. O_2 control with gas extraction after the preheater is also possible but because of fluctuation of the false air intake into the preheater it is less accurate.

In most cases, a compromise in the O_2 concentration (reducing atmosphere) has to be found, because the NO_x-emission, fuel and power consumption can be increased by increasing the O_2-concentration in the kiln.

SO_2 adsorption in the raw mill

The exhaust gas from preheaters and long dry kilns are usually used in the raw mills to dry the raw meal. The freshly ground raw meal in the raw mill has a large and very active surface where SO_2 can be adsorbed. The highest degree of adsorption (100%) can be found at very low SO_2 concentration whereas at very high concentrations (2500 - 3000 mg/Nm^3 dry) like at the Untervaz plant only an adsorption of about 30% is possible. In systems with an emission of about 1000 to 1500 mg/Nm^3 dry an adsorbtion of 50% is normal.

The degree of adsorption in the raw mill is also influenced by the temperature and the humidity. A lower temperature or a higher humidity is increasing the adsorption capacity.

Raising the proportion of compound to direct operation, time is reducing the time of high SO_2 emission and therefore also the effort of secondary measures (e.g. amount of $Ca(OH)_2$ added to the kiln feed).

6.3 Secondary Reduction Measures

Catalytic or non-catalytic elimination of SO_2 (as it is known for NO_x) is not reasonable because oxidation would lead to SO_3 and reduction to SH_2, both would be more harmful than SO_2.

Therefore, only absorption or adsorption of SO_2 can be used as secondary reduction measures.

SO_2 absorption with $Ca(OH)_2$ (slaked lime) [9]

The most common measure against SO_2 emission is SO_2 absorption with $Ca(OH)_2$.

$$Ca(OH)_2 + SO_2 \longrightarrow CaSO_3 + H_2O \qquad (12)$$

It can be added to the exhaust gas:

- at the upper cyclone stage of the preheater together with the kiln feed
- in a scrubber after the kiln system

The addition of $Ca(OH)_2$ to the kiln feed is very successfully used in three "Holderbank" plants with preheaters (RK, HV, UV) (see chapter 7). However, this technology has its limits. For instance only concentrations not exceeding about 1000 mg SO_2/Nm^3 can be reduced. At the same time, the emission limit value should be less than 400 mg SO_2/Nm^3.

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If the normal SO_2 emission is very high (> 1000 mg/Nm^3) as it used to be in the Untervaz (UV) plant in Switzerland, a separate SO_2 scrubber after the preheater has to be considered. The only slaked lime scrubber for a cement plant was built in the UV plant. There, the kiln exhaust gas flows through a Venturi reactor with an expanded fluidized bed formed by the absorbent which consists of a blend of hydrated lime (slaked lime) and raw meal [9]. A more detailed description of the UV reactor is attached (see chapter 7).

Experience showed that the temperature for SO_2 absorption with slaked lime is very important. Efficient absorption was only observed at temperatures above 350°C and below 70°C. Therefore, systems with $Ca(OH)_2$ injection into the conditioning tower or $Ca(OH)_2$ in bagfilters do not have a sufficient SO_2 absorption efficiency [10].

Replacement of $Ca(OH)_2$ power by other Ca-compounds

To save on the high cost of the fine slaked lime, other desulfurization agents were tested at the Höver plant of Nordcement for their suitability. These were coarser slaked lime, powdered chalk and partly calcined kiln meal. However, the results of these experiments were mostly unsatisfactory. The effectiveness of the slaked lime currently in use was in no case anywhere near achieved. Powdered chalk has the lowest effectiveness. The reactivity of the coarse slaked lime was also barely detectable due to the small surface area. In the case of the partly calcined raw meal which had been pneumatically withdrawn from the lowest cyclone stage of a preheater, a slight desulfurization effect could be discerned. However, the technical problems in handling were so great that its further use was abandoned.

Tests with unslaked lime (CaO) were not carried out since on account of its lower reactivity compared with slaked lime, the difference in cost does not justify the increased quantities required. Optimization, e.g. also for absorption of larger quantities of SO_2, is only likely to be achieved by selecting a finer slaked lime. Thus a material with a BET specific surface of 18 or 36 m^2/g could certainly lead to fixation of larger quantities of SO_2.

Untervaz started to use a $Ca(OH)_2$-mud from a former dump of a carbide manufacturer. The mud has a negative price and can be added to the raw material. This is reducing the $Ca(OH)_2$-feed to the scrubber.

At the Fort Collins plant precalcined material (CaO) from the precalciner is added to the bypass exhaust gas to reduce SO_2-emission. In addition to that, bypass dust is recirculated in the bypass to increase the concentration of SO_2 absorbed in the dust. These measures are sufficient to stay below the SO_2-limit.

Activated Coke Absorber

Activated coke is under presence of water a very efficient SO_2 absorber. The only filter of this kind is installed in the Siggenthal (SG) plant of "HCB. The SO_2 emission is reduced by the filter from above 500 to below 50 mg/Nm^3.

However, this type of absorber is very pricy. The Polvitec in SG is only economical because of the contribution from the village of Zurich for burning their sewage sludge.

Sodiumbicarbonate Absorber

Obourg.

Passamaquoddy Absorber

SO_2 scrubbing is only a side effect of the Passamaquoddy scrubber. As described in chapter 7 the main purpose is to recover CKD to avoid dust dumping. The SO_2 is first absorbed by the CaO in the CKD and reacts afterwards with the alkali oxides in the CKD to form an alkali salt. The alkali salt is extracted and if possible sold as a fertiliser.

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7. EXAMPLES OF SO_2 EMISSION REDUCTION

Below a collection of short descriptions of plants and their measures against SO_2 emission. Everybody is welcome to contribute to this collection.

SO_2-emission reduction

Plant:		**Fort Collins (FC)**
Capacity:		1500 t/d (design) / 1345 t/d (standard)
Raw material:		Containing kerogen and sulfur (~10% as Pyrite)
Fuel type:		Nat. gas, coal (0.44 % S), pet coke (4 to 5% S)
Plant description:		1-stage-preheater with precalciner
Supplier/equipment:		Polysius preheater, Allis-Chalmers kiln
Initial SO_2-emission problem:		The SO_2-limit of 150 ppm (i.e.~ 320 mg/Nm^3) can not
	[1], [2]	always be maintained, (sometimes up to 300 ppm) and limits the kiln output. The SO_2 emissions originate from the bypass and not from the precalciner. Kerogen in the raw material is burned in the calciner, combusted into CO2 and SO_2. At the same time CaO reacts with SO_2 if sufficient Oxygen is available to form $CaSO_4$ (SO_2 scrubbing). The bypass is the control valve for the sulfur level in the clinker. The bypass rate is 100 %!
Solutions investigated:	[2]	The reason for the emission from the bypass is believed to be due to a limited quantity of CaO being available in the bypass dust to combine with SO_2 from the kiln. So adding some dust to the bypass-duct would help. Since a lot of sulfur input into the system is from coal, the specific fuel input has to be reduced. The sulfur content of the kiln feed is rather high with 3.6%. In order to control the SO_2-emission, a splitter gate should be installed, in order to deviate a portion of return dust to the bypass scrubber.
Solution realized:	[4]	Dust return (mainly precalciner dust) from the bypass / precalciner bagfilter into the bypass duct has been installed before 1994, however the return dust system needs further modifications.
Emissions reached:	[3]	The necessary SO_2-limit has been reached. Most of the time, no additional scrubber dust is used for stack sulfur control. Consequently the low stack SO_2 (average 40 - 50 ppm) would allow a lower waste dust quantity.
Further Optimization:	[3],[4]	The plant capacity should be increased up to 1600 t/d with several modifications (and reduction of losses), without increasing the emissions. A modification of the kiln feed inlet is planned for 1995.

Literature:

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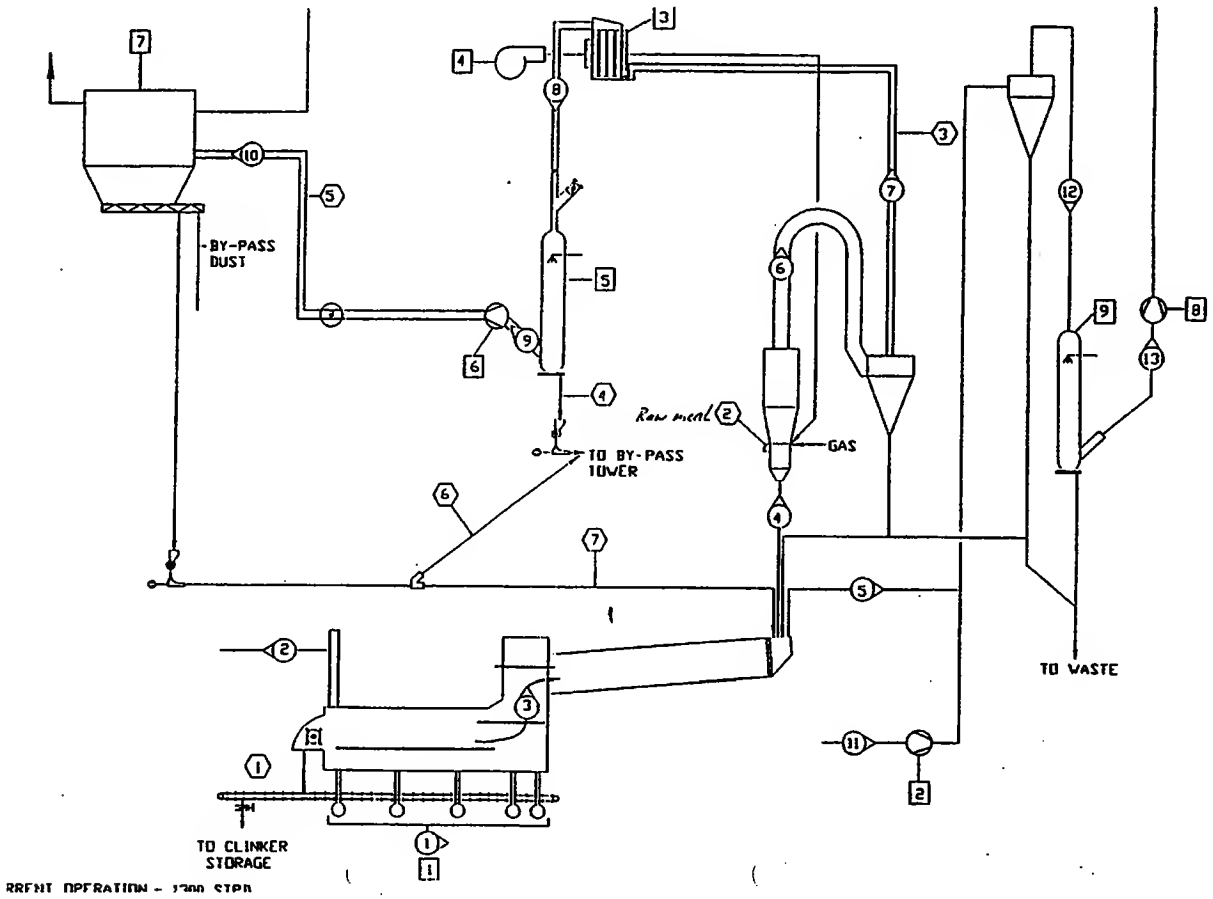
[1] POLYSIUS: Telefax (1989), VA-Dok: FC, Reg. 2D

[2] Kupper, A.K.: HMC Report VA 91/5866/E, VA-Dok: FC, Reg.2A

[3] Bachmann, R.: HMC Report VA 94/4263/E, VA-Dok: FC, Reg.2A

[4] Kupper, A.K. and Bürki, Dr. P.: HMC Report VA 95/4288/E, VA-Dok: FC, Reg.2A



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SO_2-emission reduction

Plant:	Höver (HV)	
Capacity:	3'000 t/d (design), 2'900 t/d (standard)	
Raw material:	Sulfur components: Pyrite 0.05 - 0.45 %w/w	
Fuel type:	Coal (0.4%S), fuel oil, nat. gas; alternative fuels: whole tires (max. 10% of total heat input)	
Plant description:	Kiln with 4-stage twin line suspension preheater and planetary cooler	
Supplier/equipment:	KHD kiln and preheater	
Initial SO_2-emission problem:	SO_2 emissions of up to 1300 mg/Nm^3	
	[1],[2]	(TA Luft = 400 mg/Nm^3) In direct operation, up to 900 mg/Nm^3 in compound operation
Solutions investigated:	The following investigations were proposed:	
	[3],[4],[5]	Reduction of SO_2 emissions by lowering the exhaust gas temperature.
		Lowering the temperature in the raw mill in compound operation.
		Desulfurization with $Ca(OH)_2$: different trials have been carried out. To reach an emission < 400 mg/Nm^3, the SO_2 concentration in the stack should be held at about 350 mg/Nm^3. The calculated amount (based on trial results) of $Ca(OH)_2$ needed to stay below the limit would have been 2.4 kg/t clinker in 1990 and 6.4 kg/t clinker in 1991.
Solution realized:	Desulfurization with $Ca(OH)_2$ addition to the kiln feed (bucket elevator). The $Ca(OH)_2$ is proportioned by speed control rotary valves with three independent pneumatic injection pipes, which can be used simultaneously when required. The $Ca(OH)_2$ addition was started up in November 1991.	
Emissions reached:	[1]	The addition of $Ca(OH)_2$ is controlled by the measured SO_2-concentration at the stack, in order to stay below the TA-Luft limit of 400 mg/Nm^3.
Further Optimization:	[1]	Since the costs for slaked lime are quite high, different other desulfurization agents have been tested:
		powdered chalk: low absorption efficiency
		coarse slaked lime: barely detectable effect due to small surface area
		partly calcined raw meal: slight desulfurization effect, but additional technical problems.
		Tests with unslaked lime (CaO) have not been carried out, due to its low reactivity compared with $Ca(OH)_2$. A further optimisation could be reached, using a finer slaked lime (higher surface area).

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[1] Boes, K.-H.: Measures to reduce the SO2 emission during clinker burning at Nordcement AG's Höver works.
Zement, Kalk, Gips (1993), P.514-518

[2] Waltisberg, J.: HMB Aktennotiz VA 86/66/D,
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[3] Hasler, R.: HMB letter to Nordcement AG,
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[4] Waltisberg, J.: HMB Bericht VA 87/5437/D,
VA-Dok: HV, Reg.18

[5] Waltisberg, J.: HMB Bericht VA 91/5904/D,
VA-Dok: HV, Reg. 1

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Figure A **Slaked lime chemical and physical data [1]**

"Blütenweiss slaked white lime from Fels-Werke GmbH
Chemical and physical data

CaO	71.99%
MgO	0.65%
SiO_2	2.00%
Fe_2O_3	0.45%
Al_2O_3	0.47%
SO_3	0.27%
CO_2	1.23%
Combined H_2O	21.60%
Moisture	0.90%
BET	approx. 10 m^2/g
$Ca(OH)_2$	90.0%
Fineness:	
R > 0.063 mm	4.0%

"Blütenweiss slaked white lime from Fels-Werke GmbH

Chemical and physical data

CaO	71.99%
MgO	0.65%
SiO_2	2.00%
Fe_2O_3	0.45%
Al_2O_3	0.47%
SO_3	0.27%
CO_2	1.23%
Combined H_2O	21.60%
Moisture	0.90%
BET	approx. 10 m^2/g
$Ca(OH)_2$	90.0%
Fineness:	
R > 0.063 mm	4.0%

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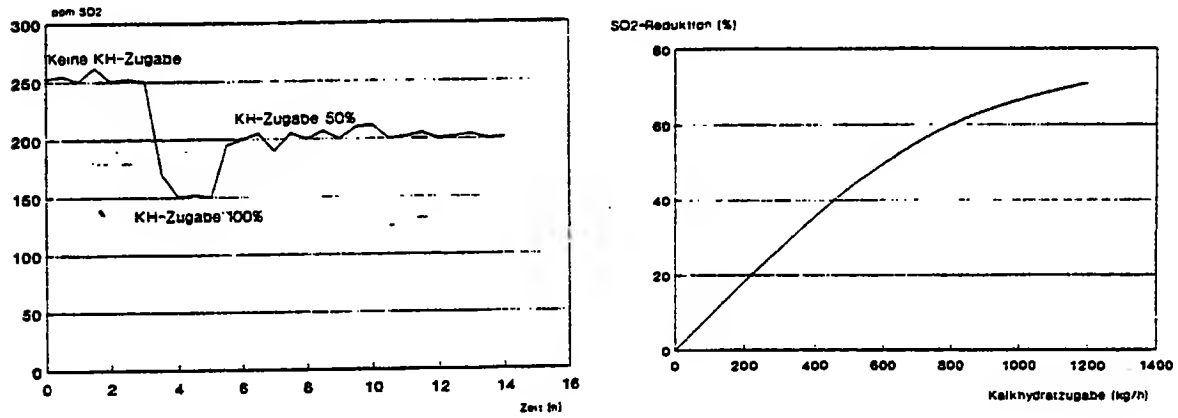
Figure B SO_2 reduction efficiency with slaked lime addition



Keine KH-Zugabe	=	no slaked lime feed
KH-Zugabe	=	slaked lime feed
Zeit	=	time
SO_2-Reduktion	=	SO_2 reduction
Kalkhydratzugabe	=	slaked lime feed rate

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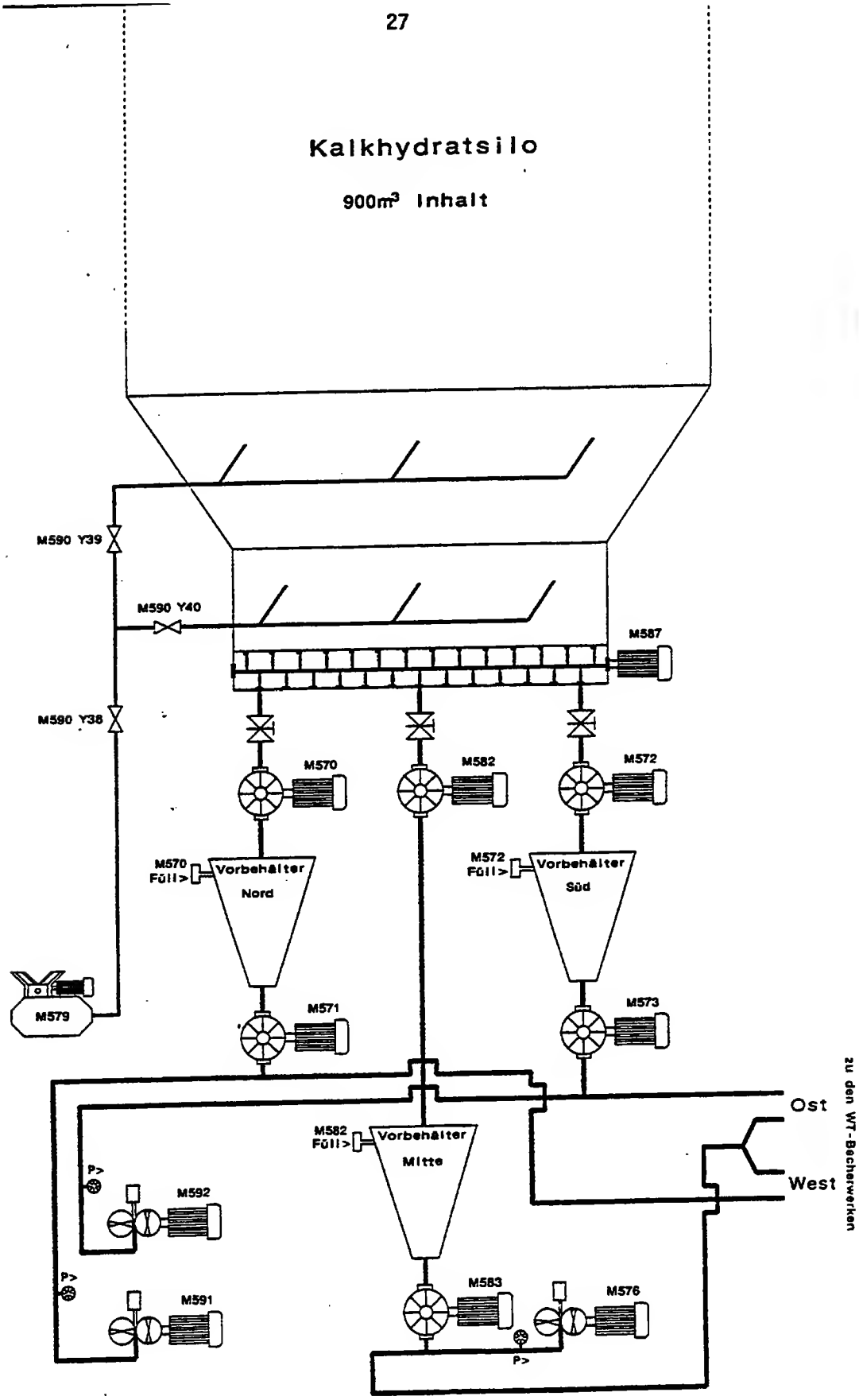
Figure C Slaked lime silo and dosing



SO_2-emission reduction

Plant:		Rekingen (RK)
Capacity:		2'200 mtpd (design), 2130 mtpd (standard)
Raw material:		Sulfur component: 0.55 % Pyrite
Fuel type:		Coal 0.4%S, fuel oil 0.8%S, waste timber
Plant description:		4-stage-suspension preheater kiln
Supplier/equipment:		Polysius kiln and preheater
Initial SO_2-emission problem:		SO_2 emissions of up to 1200 mg/Nm^3 (LRV-limit = 500 mg/Nm^3) in direct operation [LRV = Swiss clean air act]
Solutions investigated:		Reduction of SO_2 emissions by lowering the exhaust gas
	[1],[2]	temperature. Reduction from 180°C to 150°C results in a reduction of 50 mg/Nm^3, the same reduction is obtained by further lowering the temperature to 140°C.
		Desulphurization with $Ca(OH)_2$, added to the kiln feed only during direct operation leads to a reduction of 400 - 450 mg/Nm^3 at an initial emission of 750 - 1200 mg/Nm^3
Solution realized:	[3]	Desulphurization with $Ca(OH)_2$ addition to the kiln feed
		3 kg/tcli (max. 17kg/tcli) during direct operation (20% of the operating time)
		Costs are about 60'000 CHF/year
Emissions reached:		The addition of $Ca(OH)_2$ is controlled by measuring the SO_2 concentration in the stack, in order to reach the limiting values of LRV, i.e. below 500 mg/m^3.

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[2] Scheuch, J.: Entschwefelungsversuche mit Ca(OH)2 im Ofendirektbetrieb, Versuchsbericht (1986), VA Dok: RK, Reg.17

[3] "Holderbank" Cement Course: Source and reduction of emissions gaseous, VA 91/5882/E, P.7

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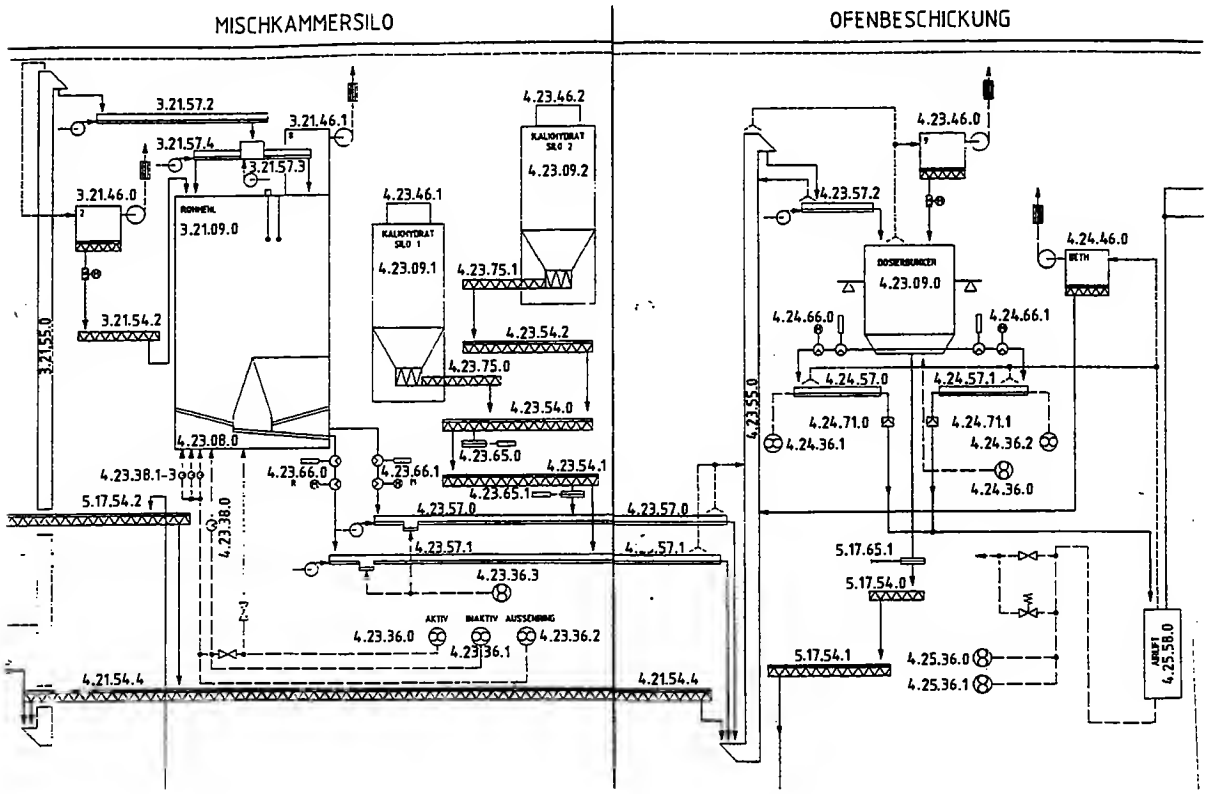
Figure A Dosing of slaked lime to the kiln feed



SO_2 -emission reduction

Plant:		Siggenthal (SG)
Capacity:		2000 t/d (design), 1900 t/d (standard)
Raw material:	[2]	Sulfur: 0.7% S
Fuel type:	[2]	Fuel oil (2.3%S), dried sewage sludge, complete tires (1.3%S)
Plant description:		Kiln with 4-stage preheater
Supplier/equipment:		Polysius kiln and preheater (Dopol), Fuller grate cooler
Initial SO_2 emission problem:		SO_2 emissions of up to 1500 mg/Nm^3 in direct operation,
	[1]	up to 900 mg/Nm^3 in compound operation (LRV-limit = 500 mg/Nm^3) [LRV = Swiss clean air act]
Solutions investigated:		SG had a project to burn sewage sludge containing Hg and at the same time the SO_2 and NOx emissions where too high. A pilot plant with an active coke filter as well as an ammonia injection into the riser duct was tested successfully. Mercury, SO_2 and other pollutants were absorbed on the active coke. NOx was reduced to N2 and H_2O with ammonia injection.
Solution realized:		A POLVITEC (Polysius Environmental Technology) active coke scrubber behind the kiln EP was successfully commissioned in 1994. Beside SO2 and Mercury it is also filtering out other pollutants like heavy metals, NH3, organic compounds and dust. The loaded coke is injected into the kiln as a fuel substitution.
		Investment cost: 30'000'000 CHF
		Operating cost: 3.3 CHF/t cli
		Emissions reached: [3] < 12 mg/Nm^3 dry

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[1] Dr. R. Stenger, : HMB Bericht MA 92/93/D, VA Dok: SG, Reg. 2

[2] ATR, Annual Technical Report 1994,"Holderbank"

[3] Effektive Emissionsminderung bei Einsatz von Sekundärbrennstoffen im Zementdrehofen von "HCB Siggenthal, Dr. 1§D. Rose, Dr. L. Brentrup, Krupp Polysius

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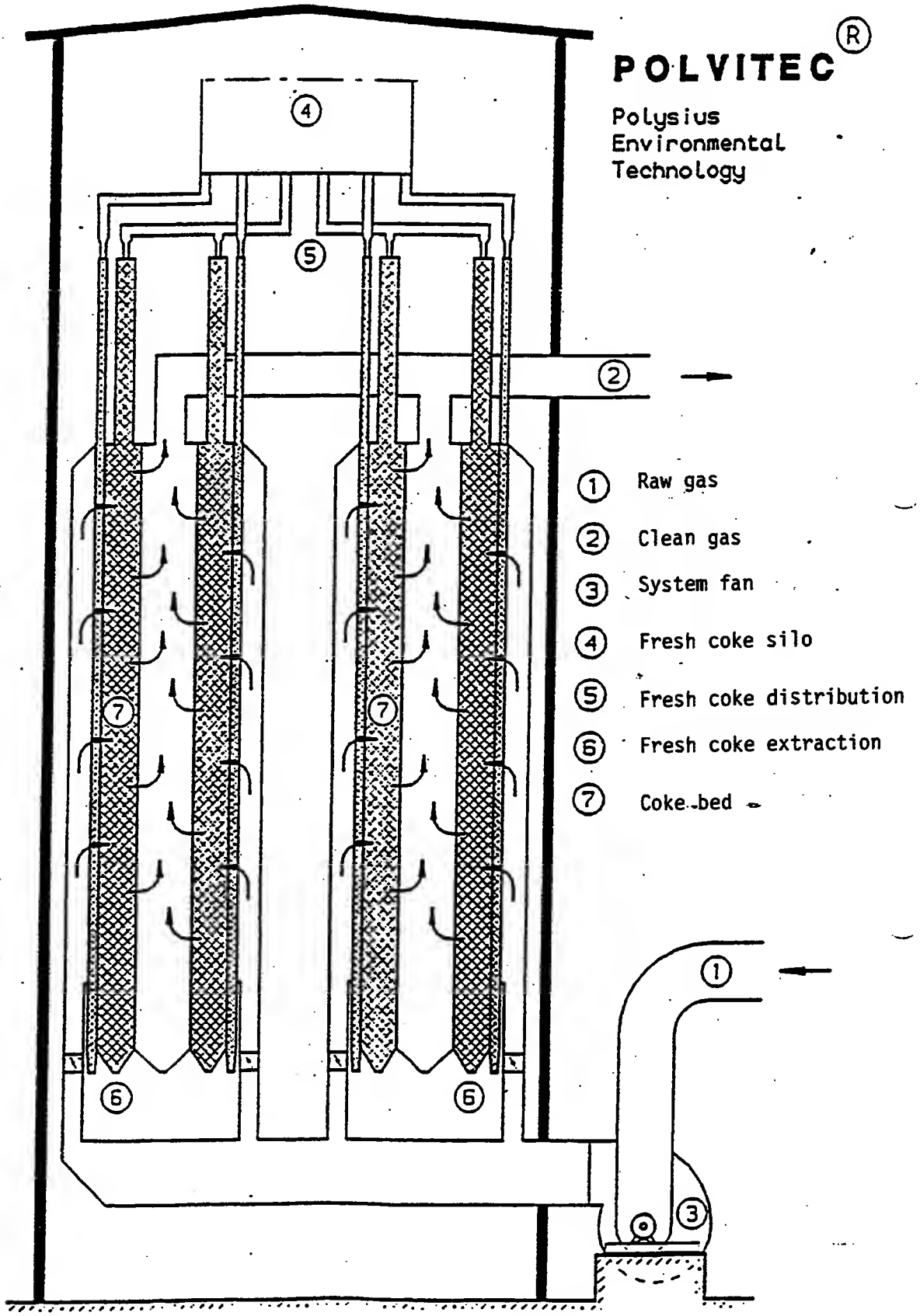

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SO_2-emission reduction

Plant:		Untervaz (UV)
Capacity:		1900 t/d (standard)
Raw material:		Sulfur components: Pyrite and Markasit, usually 10 - 16 g SO_3/kg clinker
Fuel type:	[7]	Coal (0.07%S), fuel oil; alternative fuels as dried sewage sludge, waste oil (0.6%S), destillation residue (0.35%S), plastic (0.02%)
Plant description:		Kiln with 4-stage-suspension preheater and planetary cooler (kiln III)
Supplier/equipment:		KHD kiln, preheater and cooler; Lurgi SO_2 -scrubber
Initial SO_2-emission problem:		SO2 emissions of up to 3600 mg/m3 (LRV-limit = 500 mg/Nm^3) in direct operation [LRV = Swiss clean air act]
Solutions investigated:		In 1984/85 trials to lower the SO_2 emissions by changing
	[1],[2],[3],[4]	the operating parameters have been carried out. Addition of Ca(OH)2 to the raw meal: Reduction of up to 50% of SO_2, still insufficient.
		Because no possibility of lowering the SO_2 emission of the Plant to the required 500 mg/Nm^3 by simple means as the one explained above could be found, the use of a scrubber had to be considered. Because a wet SO_2 absorber (washer) was too expensive (investment and operating costs) a dry absorber was chosen.
Solution realized:		A system consisting of a circulating fluidized bed supplied
	[1],[2]	by Lurgi, Germany, was started up in 1988. It works as follows:
		The kiln exhaust gas flow through a venturi reactor with an expanded fluidized bed formed by the absorbent that consists of a blend of hydrated lime (93% $Ca(OH)_2$) and raw meal (0.8-1.0t/h $Ca(OH)_2$ and 2.5t/h raw meal). The latter is mainly added to prevent caking of the very fine hydrated lime. Water is injected to operate the reactor as close as possible to the water dew point and to promote the reaction with the SO_2. Due to the intensive contact of the exhaust gas with the absorbent in the fluidized bed at a temperature (approx. 65°C) close to the water due point (approx. 58 to 61°C), the SO_2 becomes very effectively combined with the absorbent, whereby predominately calcium sulfite ($CaSO_3$) is generated. The absorbent then passes with the exhaust gas into an electrostatic precipitator. The bulk of the precipitated absorbent is returned to the venturi reactor (approx. 175t/h), while the remainder is discharged and either returned to the kiln feed or added to the cement.
		The stoichiometric factor of $Ca(OH)_2$ to SO_2 is about 2.3 to 2.6.
		The pressure loss in the system is 2200 Pa. This results

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		in a power demand for the fan of 315 kW. With the addition of 155 kW for the electrostatic precipitator and the transport systems, the total power consumption of the system amounts to 470 kW. Adding all the material to the kiln feed results in an increase of the SO3 content in the clinker from 1.0 to 1.3%. This is not desirable, but so far not of much concern, because it has turned out that the additional sulfates in the clinker act as a gypsum substitute, so that the SO3 content of the cement can be kept at 2.7% SO3 as before. The investment for the whole desulfurization plant, including auxiliary installations, amount to 15 million CHF. The operating costs are about 2.2 CHF/t cli (Slaked lime 1.6, electrical power 0.4, maintenance 0.5, gypsum substitution 0.3 CHF/t cli).
Emissions reached:	[5]	During the acceptance measurements in May 1989, the following values have been reached: Direct operation: 431 mg/m^3 Compound operation: 418 mg/m^3
Further Optimization:	[6]	In 1990 the circulating fluidized bed was operated with 1 t/h hydr. lime and 2.5 t/h raw meal. It has been investigated to use hot meal instead of hydr. lime. Due to analyses of hot meal, it seemed not to be possible to replace hydr. lime completely, but it should be tested whether a part of the hydr. lime could be replaced

Literature:

[1] Bonn, W., Hasler, R.: Verfahren und Erfahrung einer roh- stoffbedingten SO2-Emission im Werk Untervaz der Bündner Cementwerke. Zement, Kalk, Gips (1990), P. 139-143

[2] "Holderbank" Cement Course: Source and reduction of emissions gaseous, VA 91/5882/E, P. 5 - 6

[3] Hasler, R., Wickert, Dr.H.: HMB Bericht VA 86/5281/D, VA-Dok: UV, Reg. 17A

[4] Berclaz, Ch.:HMB Bericht VA 85/73/D, VA-Dok: UV, Reg. 17A

[5] Strahm, E., Waltisberg, J.: HMB Bericht VA 89/5665/D, VA-Dok: UV, Reg. 2

[6] Waltisberg, J.: HMB Aktennotiz VA 90/6/D, VA-Dok: UV, Reg. 2

[7] ATR, Annual Technical Report 1994,"Holderbank"

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Figure A **Burning of dried sewage sludge in a cement kiln**

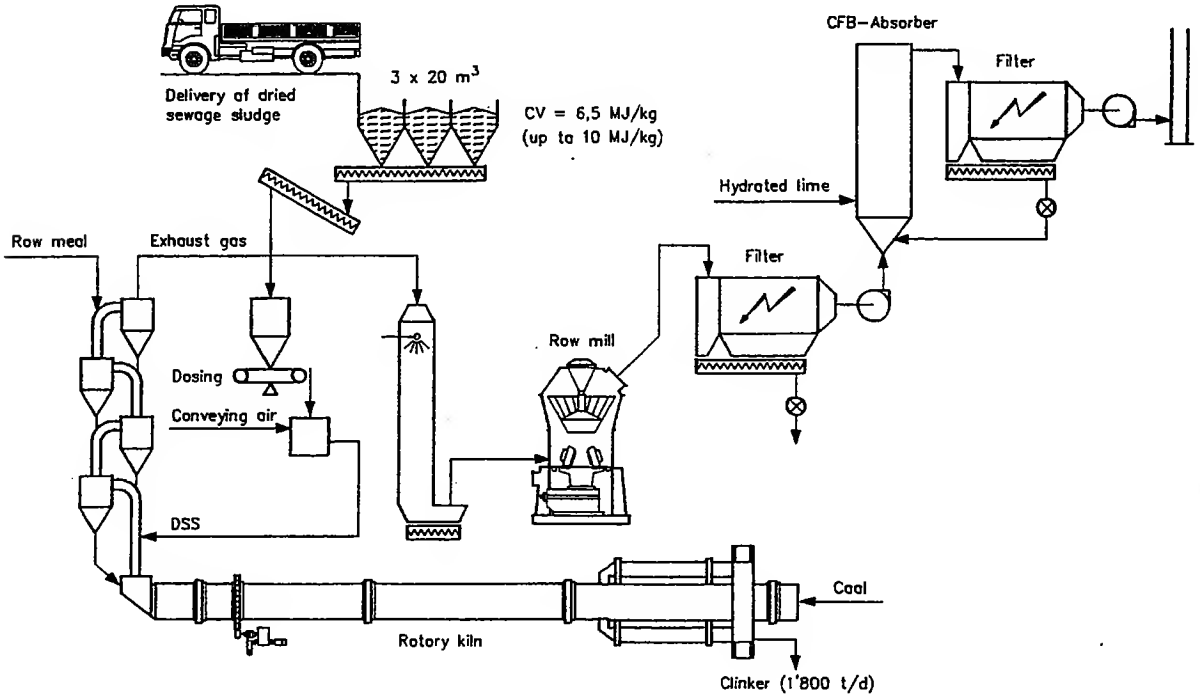
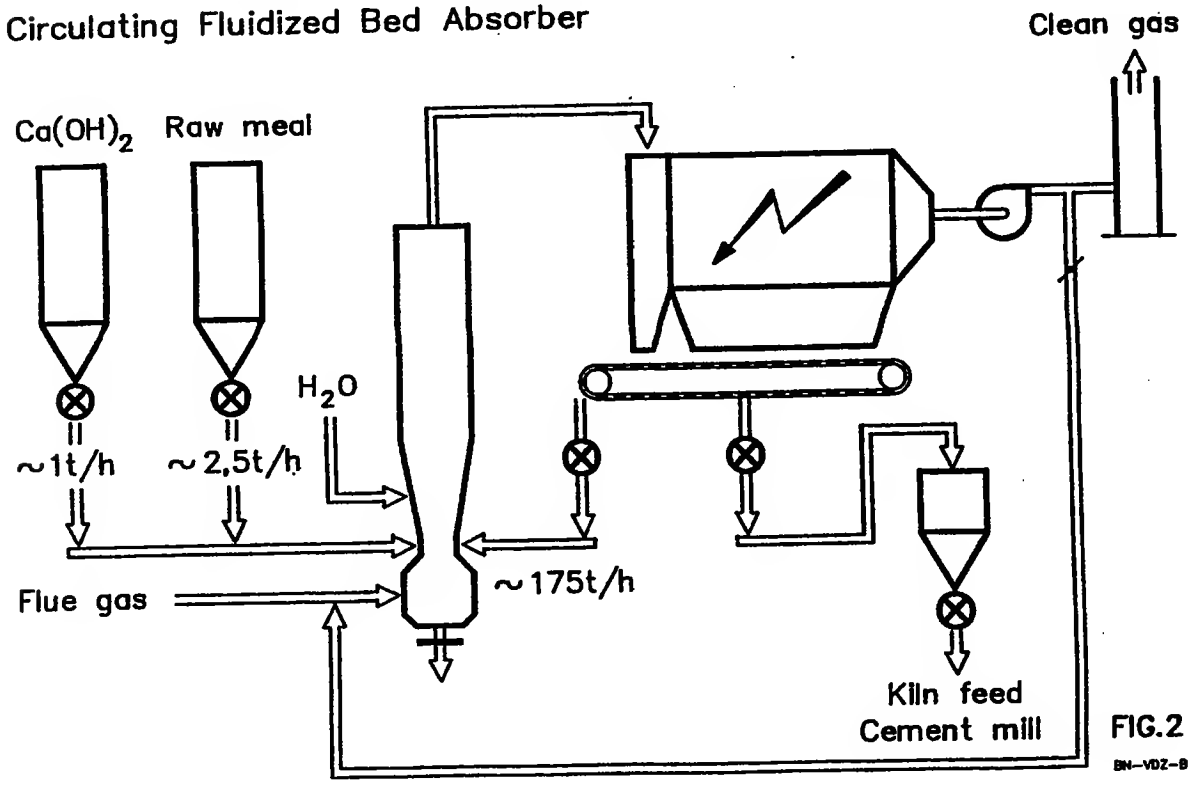


Figure B **Circulating Fluidized Bed Absorber**



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SO_2-emission reduction

Plant:		**Aalborg** Portland (not in Holderbank Group)
Capacity:		0.5 Mio. t/year of white clinker with 4 kilns
Raw material:		
Fuel type:		Fuel oil and pet coke
Plant description:		Wet process plant
Supplier/equipment:		FLS
Initial SO2 emission problem:		Due to the white color of white clinker, and since in the
	[1]	white kilns very little sulfur is retained, only special fuel quality could be used. Most of the sulfur was emitted as SO_2 in concentrations of 500 - 800 ppm.
Solutions investigated:		Since the fuel consumption is generally higher for white clinker and for a wet process, in 1980 a conversion to a semi-wet process was considered. Due to high capital costs and technical risks this investigation was not followed further.
		The recovery of some of the waste heat seemed to be a more attractive solution. In combination with the SO_2-problem, a combined heat recovery and desulfurization process seemed to be very attractive, and it was decided to build a pilot plant. From March 1989 to February 1990 a test program was carried out. The degree of desulfurization was found to be 75 %.
Solution realized:		The heat recovery and SO_2 absorption unit for the four wet kilns consisting of 4 gas/water heat exchangers and 2 scrubber systems was commissioned in 1991.
		The exhaust gas from the kiln passes first the gas/water heat exchanger before it enters the SO2 scrubber at a temperature of about 115°C. In the scrubber SO_2 is absorbed in a slurry loaded with 6 10% solids consisting of 98% $CaSO_4.2H_2O$ and 2% $CaCO_3$. The slurry is sprayed in counter current to the exhaust gas and collected in the recycle tank at the bottom of the scrubber where it is oxidized with air ($CaSO3 + 0.5O_2 \rightarrow CaSO_4$). A part of the slurry is pumped to a centrifuge where water and gypsum are separated. The rest is reinjected through a circulation line into the scrubber. A chalk slurry of 30% moisture is injected into the circulation line before the spray nozzles to replace the used and extracted absorbens ($CaCO_3+SO_2 \rightarrow CaSO_3 + CO_2$). The optimum pH for the circulated slurry has been found to be between 5.4 and 5.6 and is controlled by the amount of chalk slurry added to it.
		The circulated slurry passes on its way from the recycle tank to the nozzles a heat exchanger to

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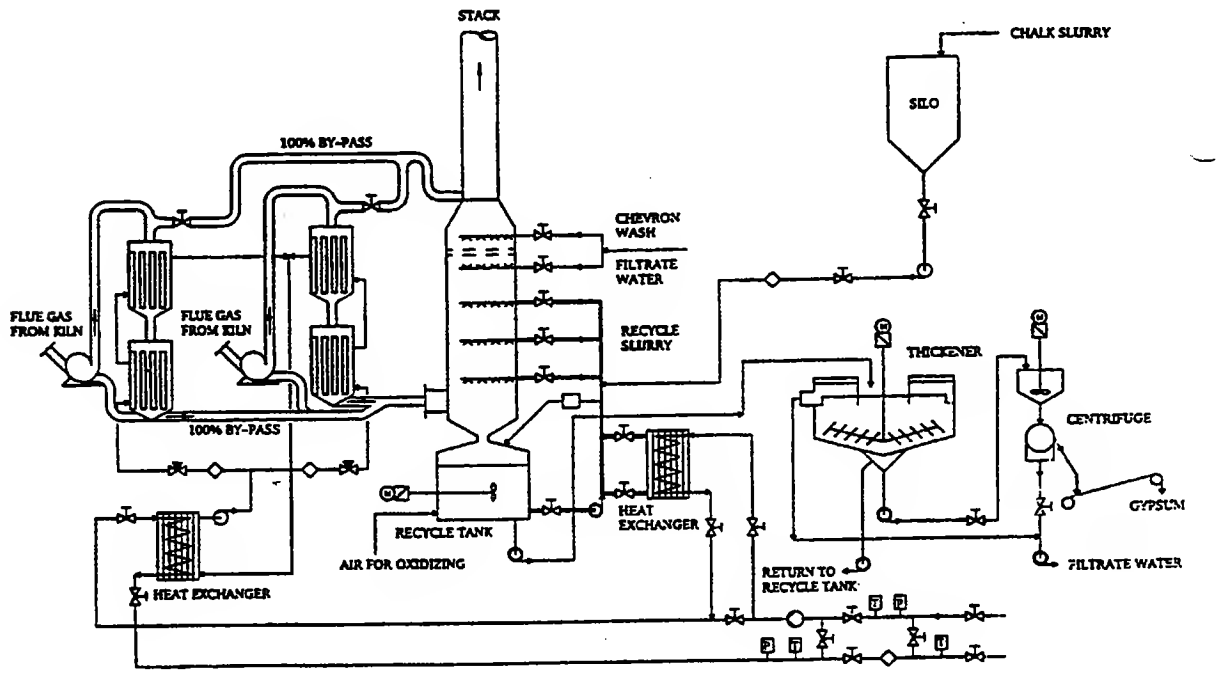
	preheat the water for the gas/water heat exchanger.
	The exhaust gas leaves the scrubber with 75% less SO_2 and a temperature of about 70°C.
Emissions reached:	The scrubber reaches an SO_2 reduction of 75-80%, i.e. the remaining SO_2-concentration is below 160 200 ppm.
Further Optimization:	Further optimization concerning the desulfurization is not foreseen.

Literature:

[1] H.E.Borgholm: A new heat recovery and desulfurization plant for 4 wet kilns in Aalborg Portland. 35th IEEE cement industry technical conference in Toronto, IEEE catalogue (1993), 395 - 409.

Figure A Heat recovery and desulfurization plant of Aalborg Portland / Denmark



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SO2-emission reduction

Plant:		Dragon Products, Thomaston (Maine, USA), (not in "Holderbank" group)
Capacity:	[1]	4000'000 t/year
Raw material:		
Fuel type:		
Plant description:		Wet kiln
Supplier/equipment:		
Initial SO_2 emission problem:		SO_2 emissions never were a problem (500 mg/Nm^3).
	[1]	During the time when Martin Marietta owned the plant (i.e., before 1980), the US EPA designed Thomaston, an upper class coastal residential area, a non attainment area (i.e. an area which had not attained the required level of emissions). This was due to dust emissions from both kiln and quarry (CKD). The Dragon plant discarded dust to produce a lower, but not a low alkali cement.
Solutions investigated:		
Solution realized:	[1],[2]	The Passamaquoddy scrubber was commissioned in 1990. Conversion is accomplished in the Recovery Scrubber by reintroducing into CKD the CO2 that was released during calcining. Exhaust gas from the kiln is cooled in a heat exchanger (A) and bubbled through a reaction tank (B) containing a slurry of CKD and water. CO_2 in the gas reacts with CKD in the slurry to reclaim CKD for kiln feed, while SO_2 reacts with potassium sulfate in the CKD to form potassium sulfate in solution.
		$CaO + SO_2\ 0.5O_2 \rightarrow CaSO4$
		$CaSO_4 + 2KOH + CO_2 \rightarrow CaCO_3 + K_2SO_4 + H_2O$
		Reclaimed CKD is separated in a settling tank (C), rinsed in a secondary settling tank (D) to remove potassium sulfate remnants, and returned to the cement plant as raw feed. Potassium solution is pumped to the crystallizer (E) for recovery as dry potassium sulfate. Heat for the crystallizer is obtained from the exhaust gas heat exchanger (F).
Emissions reached:	[3]	The exhaust gas SO_2 elimination is claimed to be over 90%. Beside SO_2 also HCl as well as ammonia and some less volatile organic compounds are absorbed.

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Literature:

[1] Dust and Other Secondary Materials Management Using the Passamaquoddy Recovery Scrubber

[2] The Recovery Scrubber , Passamaquoddy Technology

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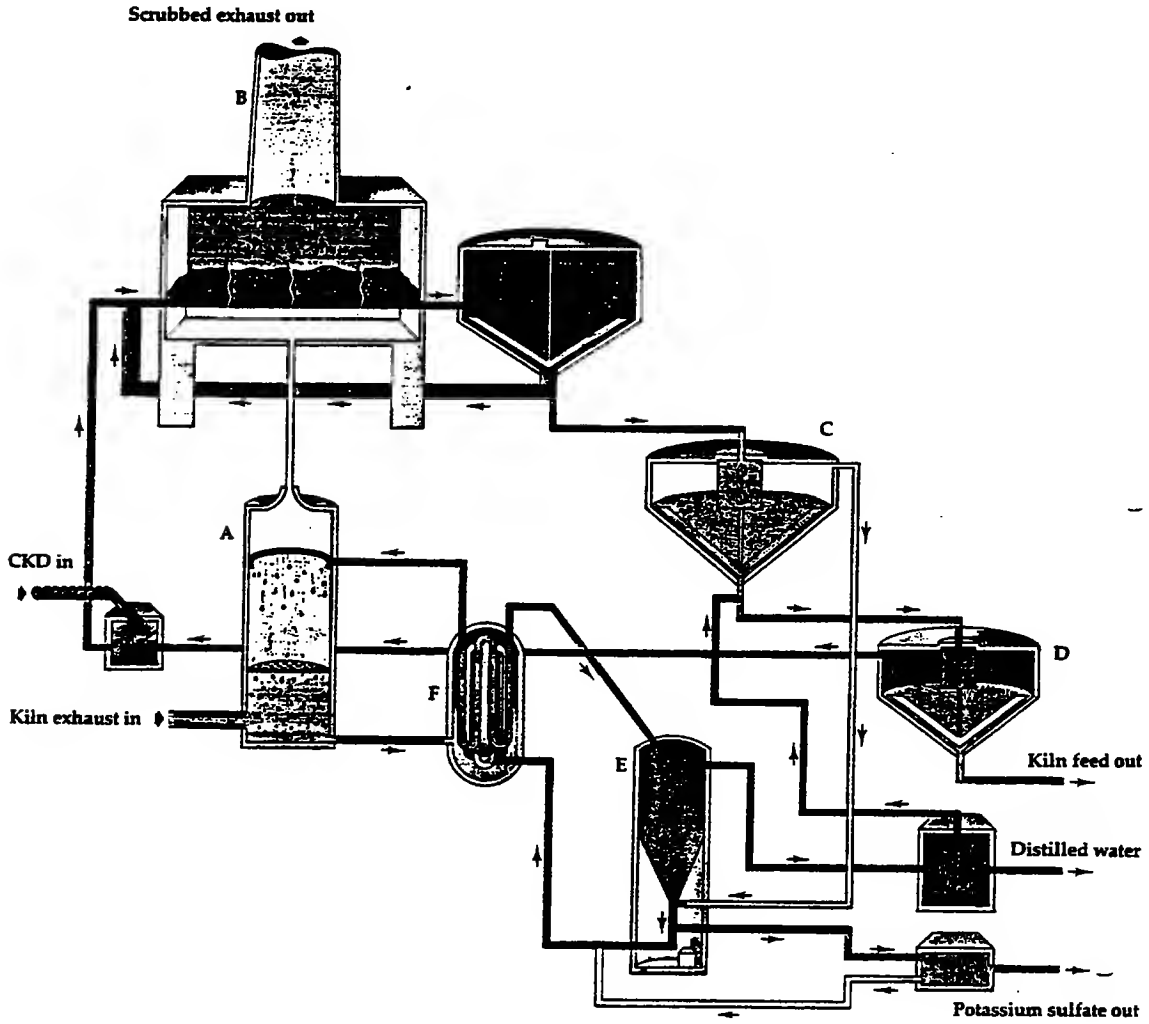
Figure A Recovery scrubber from Passamaquoddy



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[12] Emissions of NO_x and SO_2 from cement clinker burning, V. Johansen, A.H. Egelov, Denmark

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Figure 14 SO_2-Reduction =f(Mol-ratio Ca/S)

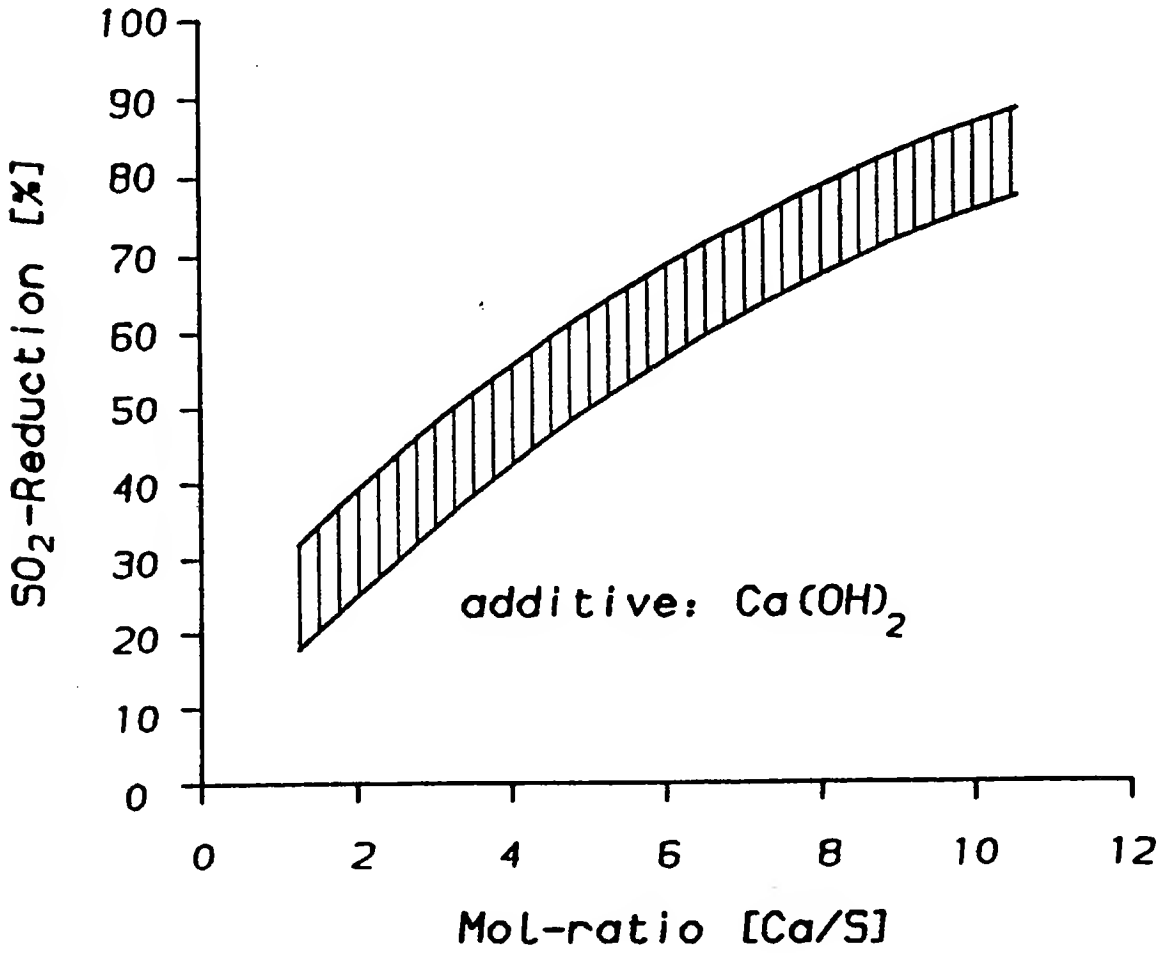

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Figure 15 SO_2 REDUCTION WITH CA $(OH)_2$

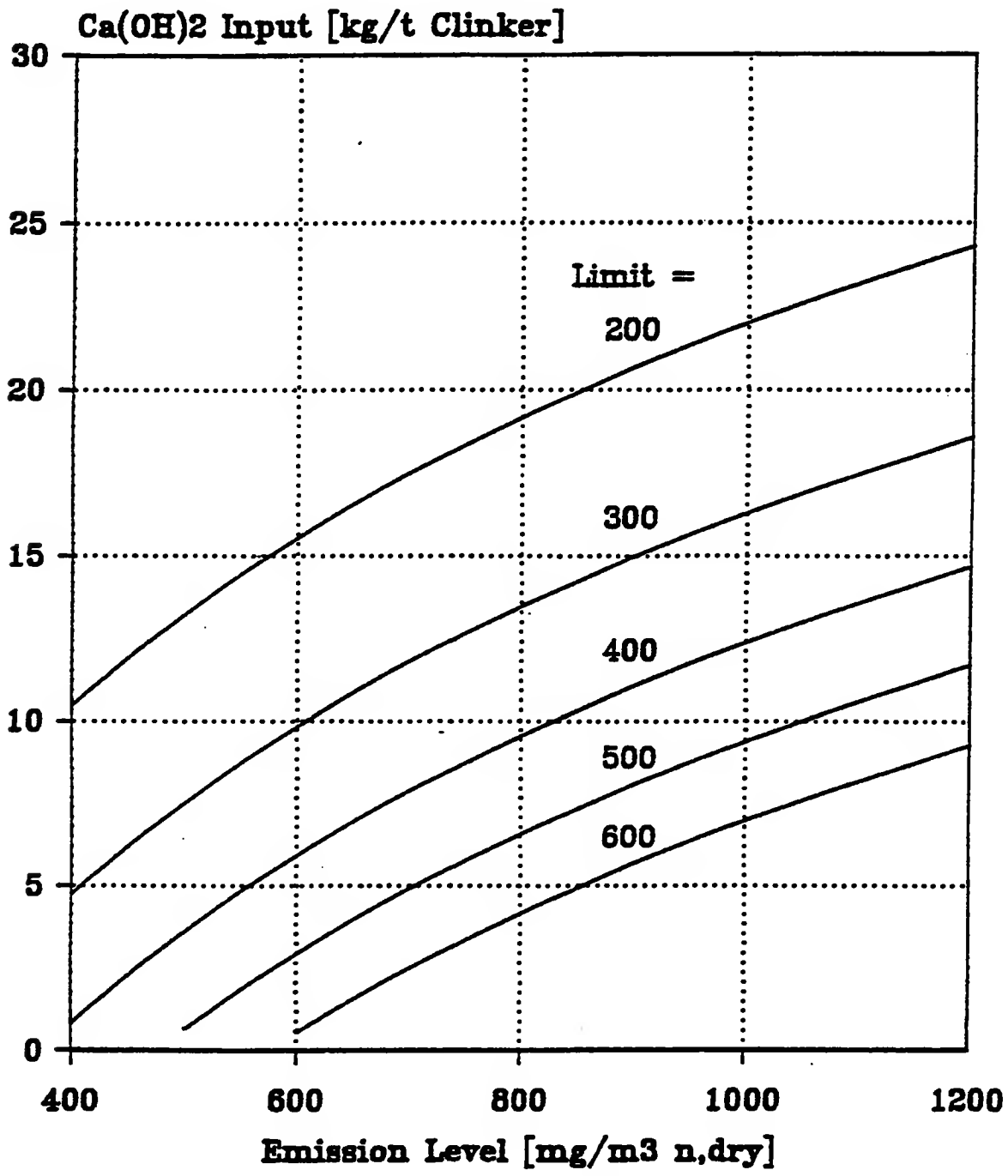
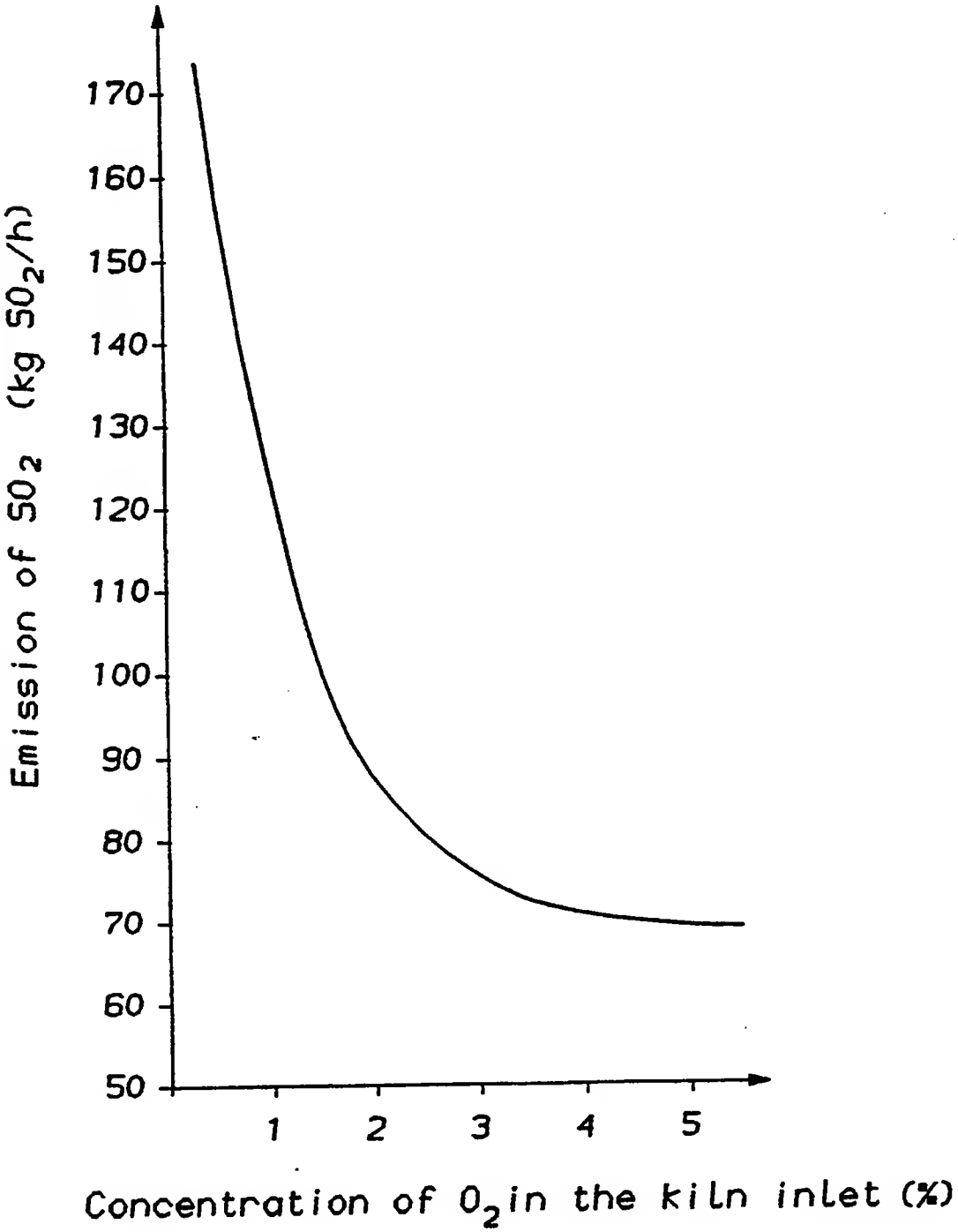


Figure 16 Influence of oxygen content in the kiln gas



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Chapter 2

Dedusting

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4. Dedusting Concepts for Cement Kilns 157

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1. INTRODUCTION

Dust filters were the first secondary emission reduction measure in the cement industry. The motivation for dedusting of exhaust gas and vent air are:

- Compliance with environmental regulations
- Reduction of product loss
- Protection of employees and equipment from harmful dust impacts (irritation plugging, erosion)

This paper is focused on the dedusting of kiln/raw mill exhaust gas and clinker cooler vent air. They are the largest dust filters of the entire clinker production line. Because they have to provide both very high reliability and efficiency under extremely severe conditions those filters have to fulfill the most difficult task among all cement plant dedusting equipment. The basic working principles of the presented filters are also valid for other applications.

2. PRESENT STATE OF CEMENT KILN EMISSION

According to the very much varying dust emission limits in countries where cement plants are operated and the development in the dedusting technology, the average dust emission from cement kilns varies between 10 and 500 mg/Nm^3 dry. Only a few exceptions are above 1000 mg/Nm^3.

3. PRESENT LEGAL SITUATION

Dust was one of the first stack emissions that were regulated and is today still the only emission limit for some plants. The reduction efficiency required is much higher compared to other emissions like SO_2 or NO_x. The dedusting efficiency of modern dust filters is about 99.999% compared to only 95% of very good SO_2 filters.

In Europe, emission limits are expressed as mass of particulates per gas volume: mg/Nm^3 (N: 273 K and 101325 Pa). Usually the gas volume is calculated on dry base. In some countries the gas volume is referred to a certain oxygen concentration, mostly 10% O_2.

Emission Limits

Nm^3_{dry}= m^3 at 273 K, 101300 Pa and 0% water

Europe: 20 - 50 - 500 mg/Nm^3

Germany: 50 mg/Nm^3 $_{dry\ at\ 10\%\ O2}$

In the United States of America not all plants do have a dust emission standard. But all of them do have an opacity limit which is to some degree correlated to the dust emission.

USA (e.g.):	0.3 lb/$t_{feed,\ dry}$	ca. 80 mg/$Nm^3_{2\%O2,\ wet}$
	0.05 - 0.08 gr/dscf	ca. 115 - 180 mg/Nm^{3dry}
	0.015 gr/acf	ca. 60 mg/$Nm^3_{2\%\ O2,\ wet}$
	5.5 lb/hr	ca. 15 mg/$Nm^3_{2\%\ O2,\ wet}$

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Some US plants also have to comply with PM10 limits. PM10 stands for particulates smaller than 10 µm. Particulates smaller 10 µm are small enough to enter and mechanically damage the lungs. To express the limits the same units as discussed above are used.

USA PM10 (e.g.):	0.016 gr/dscf	ca. 37 $mg/Nm^3_{2\% O2, wet}$
	4.7 lb/h	ca. 13 $mg/Nm^3_{2\% O2, wet}$
	0.015 gr/acf	ca. 60 $mg/Nm^3_{2\% O2, wet_t}$

US plants burning hazardous waste are regulated under BIF (Burners and Industrial Furnaces). Other plants do have a state permit defining certain parameters like NO_x, SO_2 CO, particulates (dust) and THC emission. The limits for these emissions are called emission standards. This standards are individually defined for each plant and usually represent the operating situation under certain conditions. Therefore, the US standards are different from the emission limits in Europe where emission limits are valid for a whole state or country.

For comparison reasons all emission limits/standards are indicated in mg/Nm^3. The emission standards in the USA are usually not using mg/Nm^3 but ppm, $lb/t_{dry\ feed}$gr/dscf, $lb/1000\ lb_{gas}$, b/hr, etc. To convert them into mg/Nm^3 certain assumptions were necessary.

All the above explained emission limits do include definitions how and when the compliance tests have to be carried out. It is, e.g., very important whether the emission has to be measured continuously or not. Some dust filters like electrostatic precipitators (EP) are very sensitive on process changes and can have an increased dust emission during transition periods and may not be in compliance during that time.

4. STANDARD TECHNOLOGY (ST) FOR DUST EMISSION REDUCTION

(Technology frequently used in the cement industry)

ST for cement kiln dedusting is the application of bagfilters or electrostatic precipitators. Both are very frequently used and can usually comply with today's dust emission regulations.

Some of the older dedusting technologies like gravel beds or multiclones are still in operation but are not build any more, mainly because they do have difficulties to comply with today's more stringent dust emission regulations.

To reduce the emissions from a pyroprocessing system to a certain controlled level, three basically different methods are available:

- Maintain the existing process while reducing the input of precursor substances into the system
- Modify the existing process (primary reduction measures)
- Maintain the existing process while adding a separate gas cleaning unit for the exhaust gas (secondary reduction measures)

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4.1 Reduction of Precursor Substances Input into Kiln System

Dust input can obviously not be reduced because it is the raw material for our product.

4.2 Process Optimization (Primary Reduction Measures)

Modifications of the process can reduce dust emissions from existing dust filters (see below). However, the dust reduction achieved in the process is far below the efficiency of a dust filter. This means that process conditions can support the dedusting equipment to work properly but the final dust emission is always determined by the filter type and its efficiency (secondary measure).

However, there are two exceptions where process changes can even be more efficient than a dust filter. If a grate clinker cooler with conventional vent air dedusting is converted into a ventless system or a satellite clinker cooler is used instead of a grate cooler, the dust emission of the clinker cooler becomes virtually zero.

4.3 Secondary Reduction Measures

Almost all cement kilns are equipped with dust filters because of economical and ecological reasons. Nowadays, there are basically two different types of dust filters used: bagfilters and electrostatic precipitators. Some time ago also multiclones and gravel bed filters were installed. They are less efficient than modern systems and very often too sensitive on process changes.

5. DUST CHARATERISTICS

The characteristics of dust have a strong influence on the behavior and design of dust filters and on the impact of the dust on its environment. Dust is characterized by

- Size
- Shape
- Hardness
- Chemical composition
- Mineral structure
- Electrical resistivity
- Specific weight
- Angle of repose
- etc.

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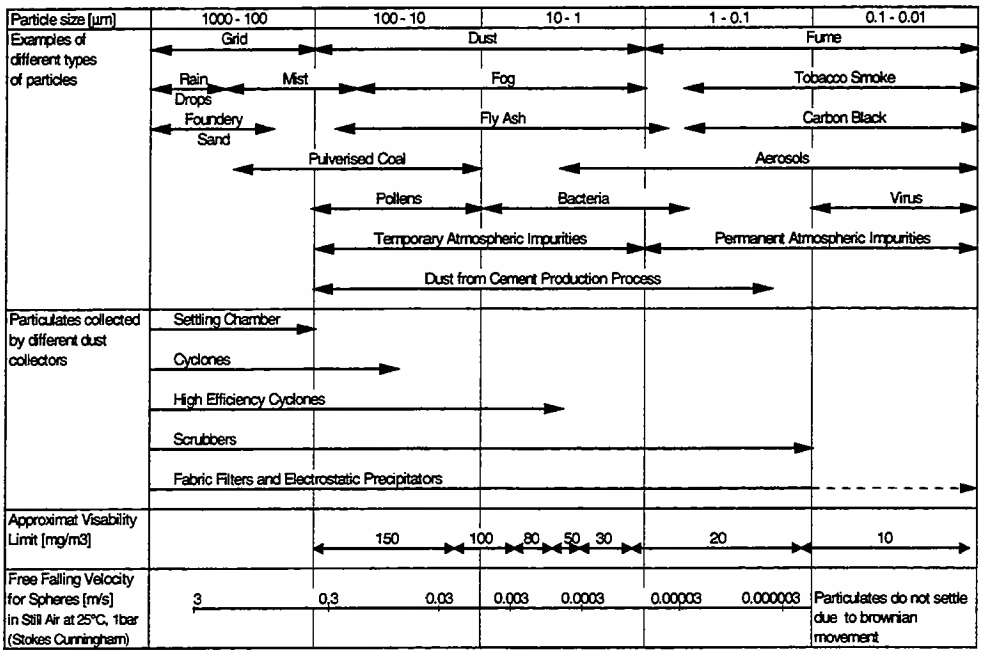
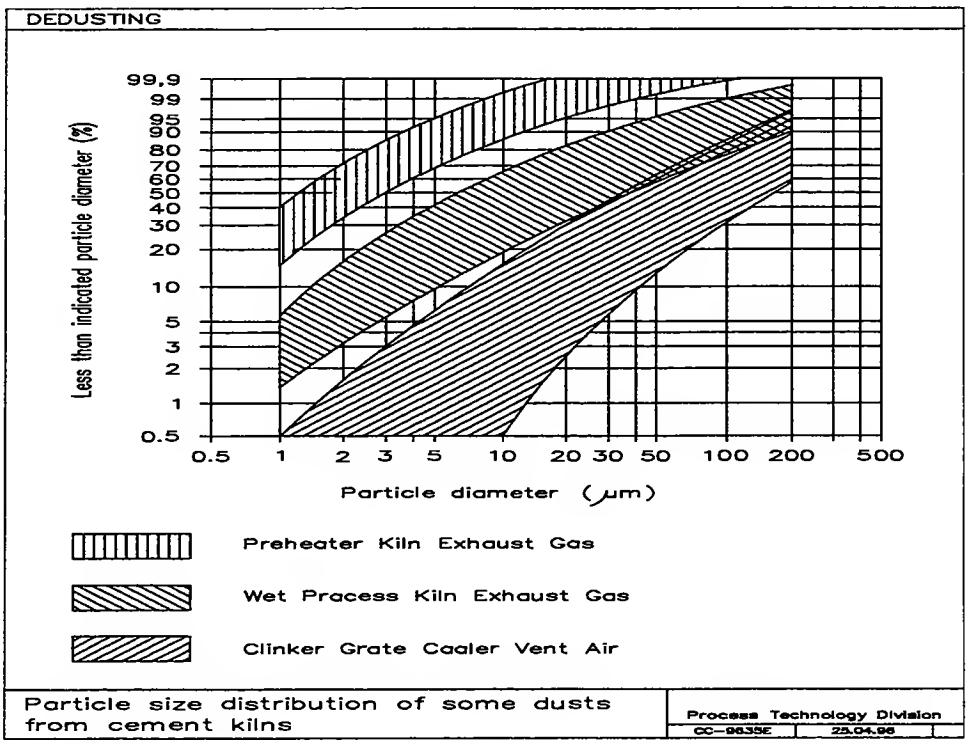
Fig 1. Comparison of sizes and physical characteristics of various dusts



The character of the dust is defined by its origin and the different treatments like grinding, blending, classifying or burning. Dust from a preheater kiln is much finer than dust from a clinker cooler and because of this more difficult to separate.

Fig. 2: Particle size distribution of some dusts from cement kilns



6. DEDUSTING EFFICIENCY

To describe the performance of a filter or to compare different filter systems the non-dimensional dedusting efficiency η is used. It describes the filter performance independently from the filter load.

$$\eta = 1 - \frac{r}{R} \qquad 1)$$

where:

η = dedusting efficiency

r = clean gas dust content

R = raw gas dust content

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Electrostatic Precipitator (EP)

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1.3 How EP Clean Gas Dust Content is determined 100
1.4 HT-Rectifier 121
1.5 Voltage-Current Curves 125

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Our companies, and thousands more throughout the world, will be faced with a substantial environmental challenge over the next few years. Increased governmental regulation and enforcement of clean air laws will require your air pollution control equipment to consistently meet rigid performance standards. And with an electrostatic precipitator that is *always* a challenge.

1. TECHNOLOGY BASICS OF EP

1.1 How does an EP work?

Fig. 3 shows a schematic drawing of an electrostatic precipitator. An industrial precipitator has a number of passages through which the gases pass at a velocity of about 1 m/s. The passages are formed by two parallel rows of vertically mounted collecting plates and a number of discharge electrodes vertically suspended between the collecting plates.

Normally the spacing of the discharge and collecting electrodes varies between 125 and 200 mm and the voltage applied between them is 35 to 110 kV negative DC according to spacing, gas and dust conditions.

The high negative voltage applied to the electrically insulated discharge electrodes creates a strong electrical field between the discharge electrodes and the earthed collecting plates. The highest strength occurs near the discharge electrodes. As the voltage is raised, electrical breakdown of the gas close to the electrode surface takes place. This breakdown, called "corona", appears as a bluish glow extending into the gas a short distance beyond the surface of the discharge electrode.

The corona produces large numbers of gas ions, the positive ions being immediately attracted to the discharge electrodes while the negative ions migrate towards the collecting plates.

Some of the moving ions attach themselves to dust particles suspended in the gas between the electrodes. Dust particles are charged either by bombardment by the ions moving under the influence of the electrical field, or by ion diffusion, both types of charging taking place simultaneously. The particle size determines which type of charging is predominant, ion diffusion being the prevailing mechanism for particle sizes below 1 micron.

Through the influence of the electrical field, the negatively charged particles migrate towards the collecting plate to which they adhere while being electrically discharged.

These particles build up a layer of dust on the plate surface which is dislodged by rapping.

The dislodged particles fall by gravity towards the bottom of the precipitator, ending up in the bottom hopper from where the dust is extracted by either mechanical conveyor (drag chain or screw conveyor) or pneumatic type system.

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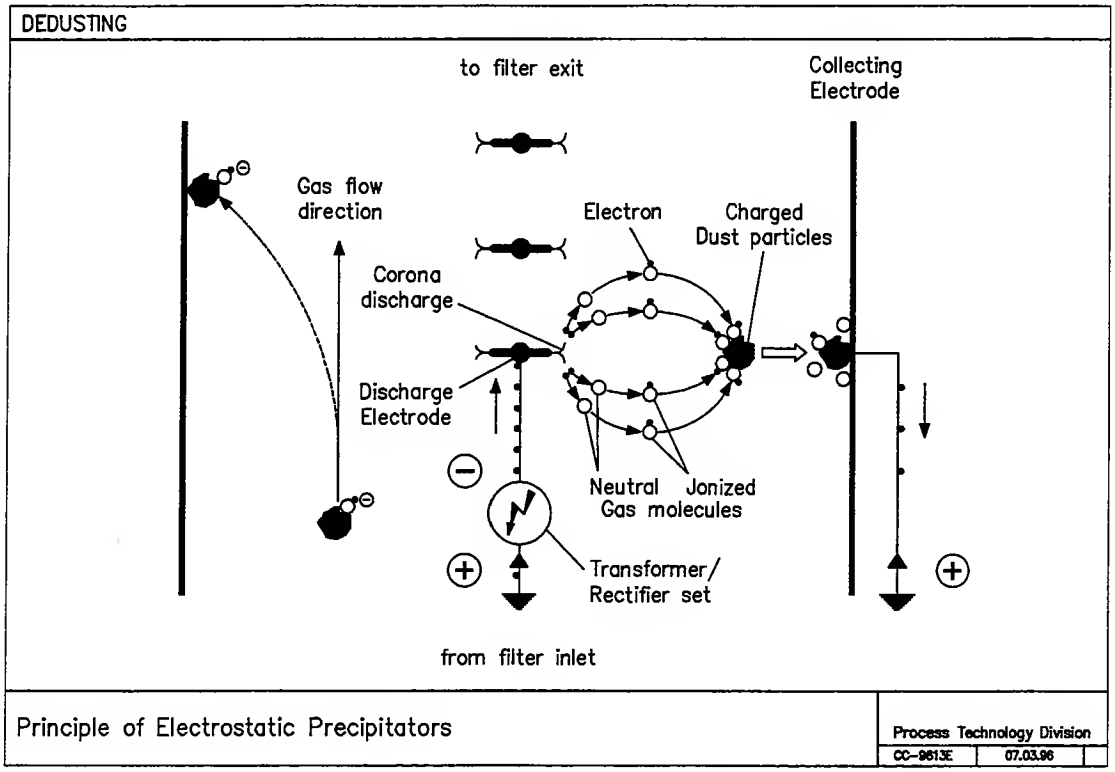
Figure 3 How electrostatic precipitators work



1.2 How EP Efficiency is determined

There have been many attempts over the years to develop satisfactory equations based on fundamental theories in order to enable the efficiency of a precipitator to be forecast. They are contained in a large number of technical papers, which are conveniently summarized in H.E. White's book entitled "Industrial Electrostatic Precipitation". Generally the performance of EPs can be expressed by the following Deutsch formula:

$$\eta_{EP} = 1 - e^{\left(-\frac{\omega \cdot L}{v \cdot s}\right)} \qquad 2)$$

or

$$\eta_{EP} = 1 - e^{\left(-\omega \frac{A}{Q}\right)} \qquad 3)$$

where

η	=	Efficiency of the electrical precipitator
ω	=	Particle migration velocity (m/s)
A	=	Total projected collecting area (m^2)
Q	=	Gas flow (m^3/s)
L	=	Field length (m)
v	=	Gas velocity (m/s)
s	=	Distance between collecting and discharge electrodes (m)

From equation 3 it follows that the dedusting efficiency of a precipitator depends on:

I	the migration velocity w (m/s)
II	the total projected collecting area A (m^2)
III	the gas flow Q (m^3/s)

1.3 How EP Clean Gas Dust Content is determined

If equations 1 and 3 are combined then one can describe the clean gas dust content r in function of

R	the raw gas dust content (g/m^3)
ω	the migration velocity (cm/s)
A	the total projected collecting area (m^2)
Q	the gas flow (m^3/h)

$$r = R \cdot e^{\left(-\frac{A}{Q}\omega\right)} \qquad [mg/m3] \qquad 4)$$

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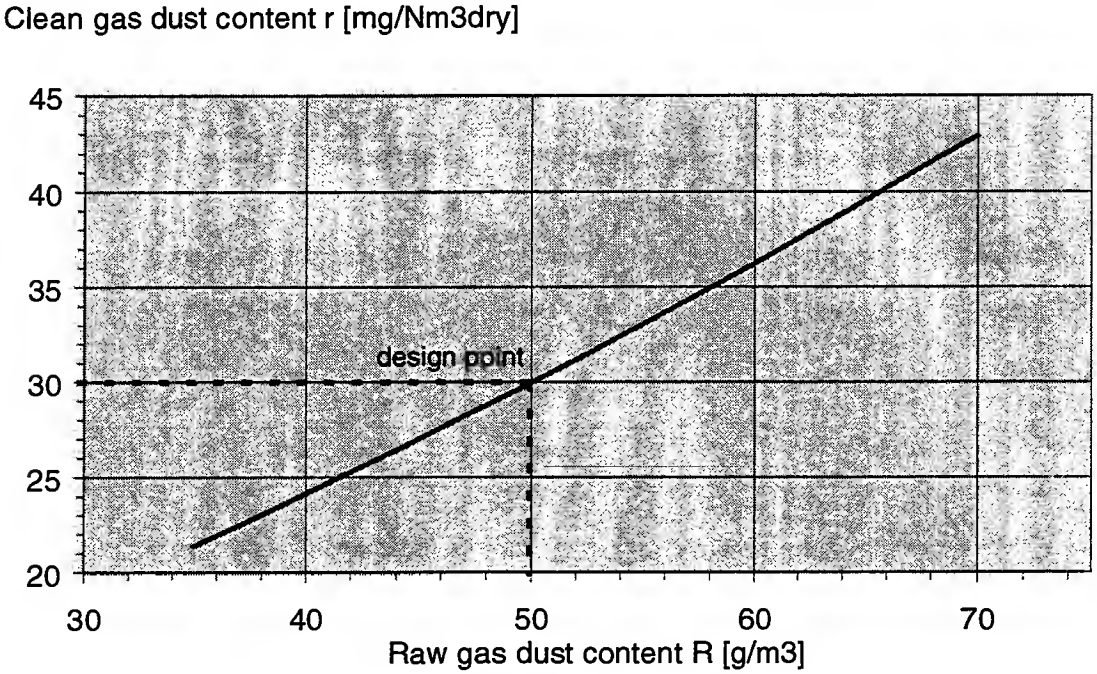
1.3.1 Clean Gas Dust Content (r) in Function of the Raw Gas Dust Content (R)

With respect to equation 4, one would expect that the raw gas dust content (R) is directly proportional to the clean gas dust content (r). However, because the migration velocity is increasing with R, the effect of R on r is much weaker than expected.

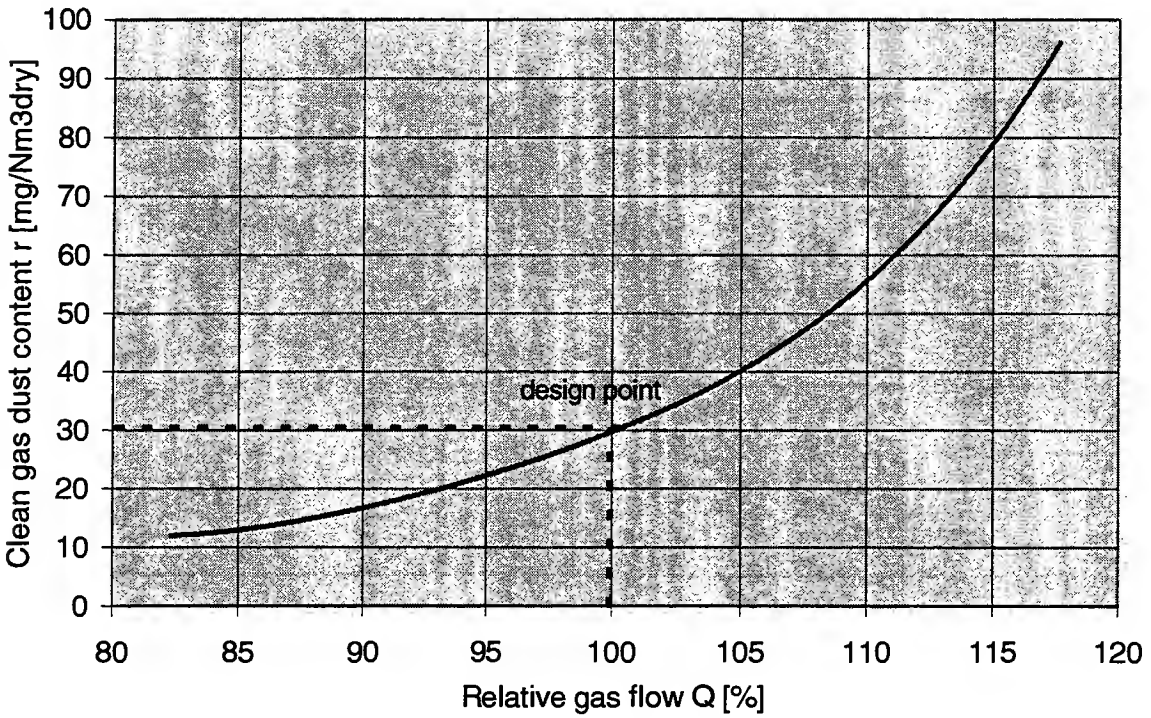
Figure 4 Example of correlation between raw gas dust content R and clean gas dust content r



1.3.2 Clean Gas Dust Content (r) in Function of the Gas Flow (Q)

The equation 4 shows that r is an exponential function of the inverse gas flow (Q). However, the migration velocity (ω) is also influenced by the gas flow. Therefore, the correlation of r and Q is not exactly according to the equation 4 with ω = constant.

Figure 5 Example of a correlation between gas flow Q and clean gas dust content r for a modern kiln EP during compound operation



At relative gas flows above 100%, r is increasing exponentially because of the exponential correlation of r and Q (equation 4) and the amplifiying effect of turbulence and dust re-entrainment from the collecting plates.

The latter is overlaid by other effects mainly based on physical and chemical changes of the particulates caused by the lower clean gas dust content (r).

At this point we already realize that the calculation of r is very complex because, unfortunately, migration velocity ω is not constant but a function of R, Q and other variables.

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1.3.3 Clean Gas Dust Content (r) in Function of the Total Projected Collecting Area (A)

The total projected collecting area (A) is

$$A = l \cdot h \cdot G \cdot F \cdot 2 \qquad [m2] \qquad 5)$$

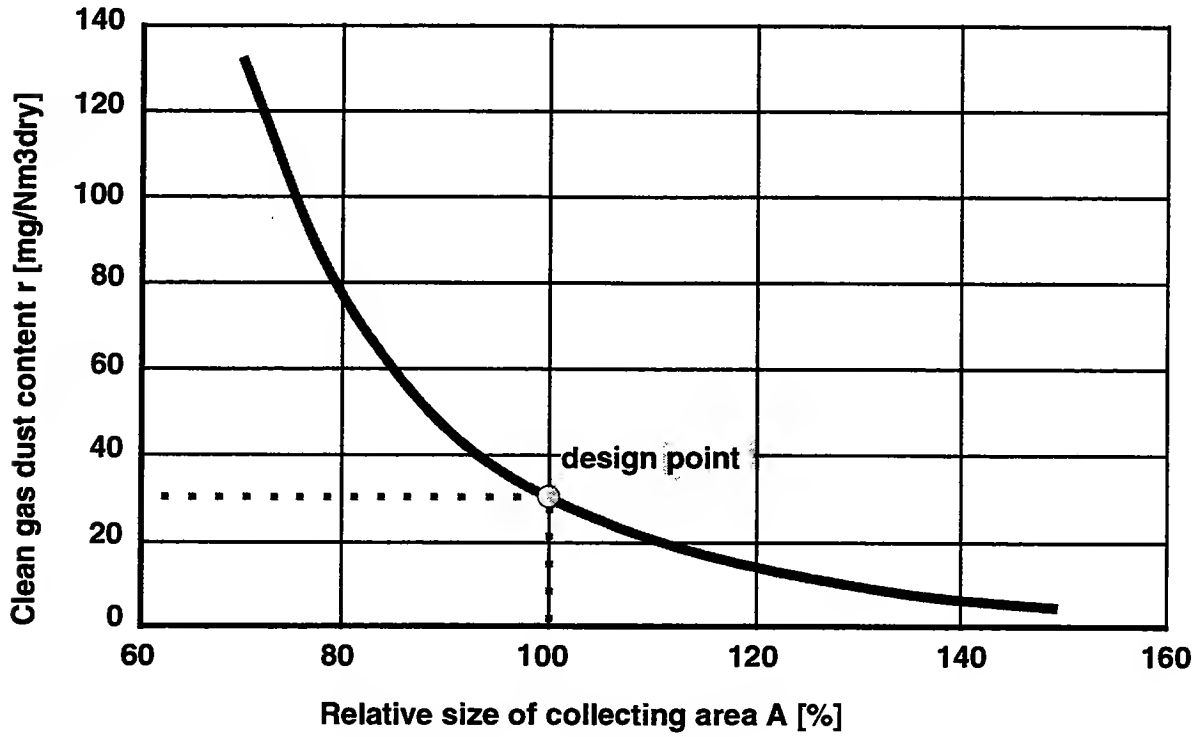
where

l	=	Length of field (m)
h	=	Height of field (m)
G	=	Number of gaps of one field (-)
F	=	Number of fields (-)

The factor of 2 is required because both sides of the collecting electrodes are active during the dust extraction process.

The correlation between A and r is about inverse to the correlation between Q and r (see equation 4). It is important to notice that the required collecting area is increasing exponentially with the reduction of the clean gas dust content.

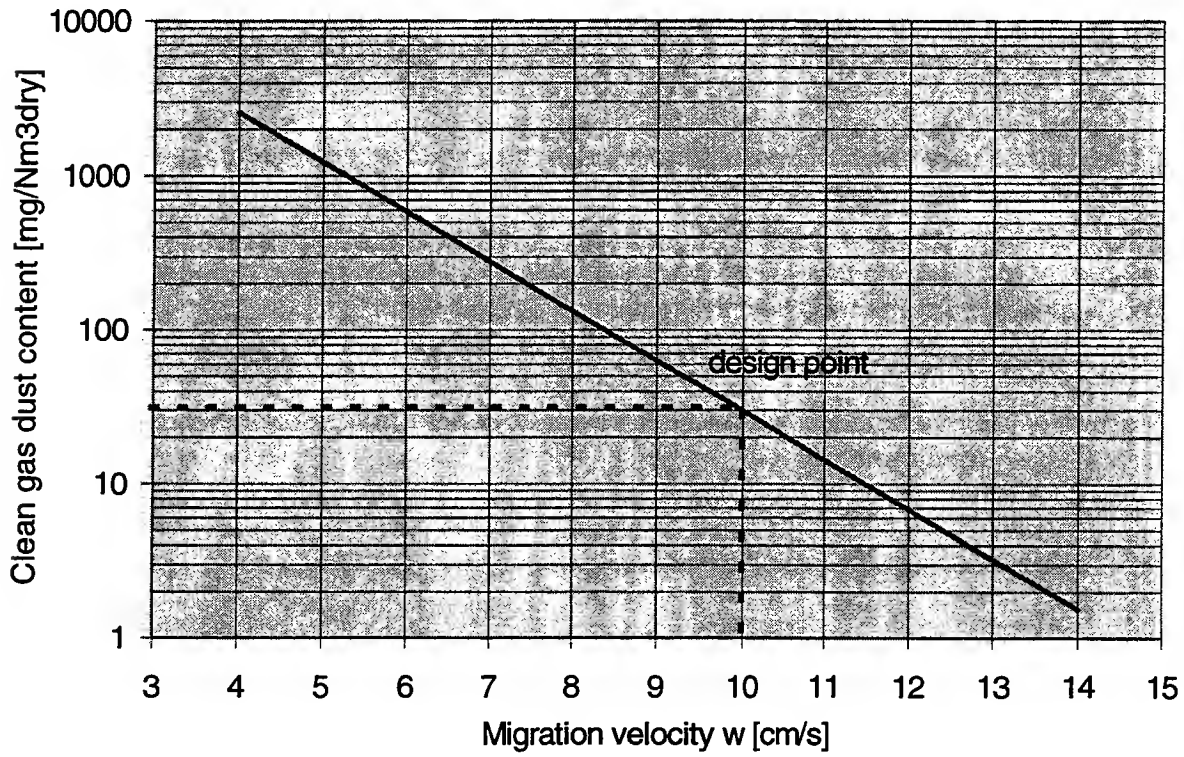
Figure 6 Correlation between the projected collecting area A and the clean gas dust content r



1.3.4 Clean Gas Dust Content (r) in Function of the Migration Velocity (ω)

Among the variables in equation 4 the migration velocity is the one with by far the most practical significance. Not because it has a stronger impact on r than the others (see equation 4) but because it represents the effect of all other variables besides Q, R and A on the EP efficiency.

Figure 7 Clean gas dust content r in function of the migration velocity ω



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The migration velocity can, somewhat simplified, be understood as the average velocity of the dust particles in their migration from the discharge to the collecting electrode in the electrostatic field.

The migration velocity (ω) itself is a function of many other variables like

- Nature of dust
  - Electrical resistivity
  - Size
    Gas condition
  - Temperature
  - Volume
  - Humidity
  - Chemical composition
  - Dust load
  - False Air
- Energization of electrical fields
  - Voltage
  - Current
- EP design
  - Gas distribution
  - Electrode design
  - Electrode cleaning

and these are only the most important ones.
With this information we can rewrite equation 4 as follows:

$$r = R \cdot e^{\left(-\frac{A}{Q} \bullet \omega\,(\Omega, \varphi, T, Q, \tau, c, r, R, I, U, \sigma, \ldots)\right)} \qquad 6)$$

For most of the mentioned variables there exist empirical graphs describing the correlation between the variables and ω. Some of these graphs were published but others are the secrets of the suppliers.

Various attempts to calculate ω theoretically were not successful.

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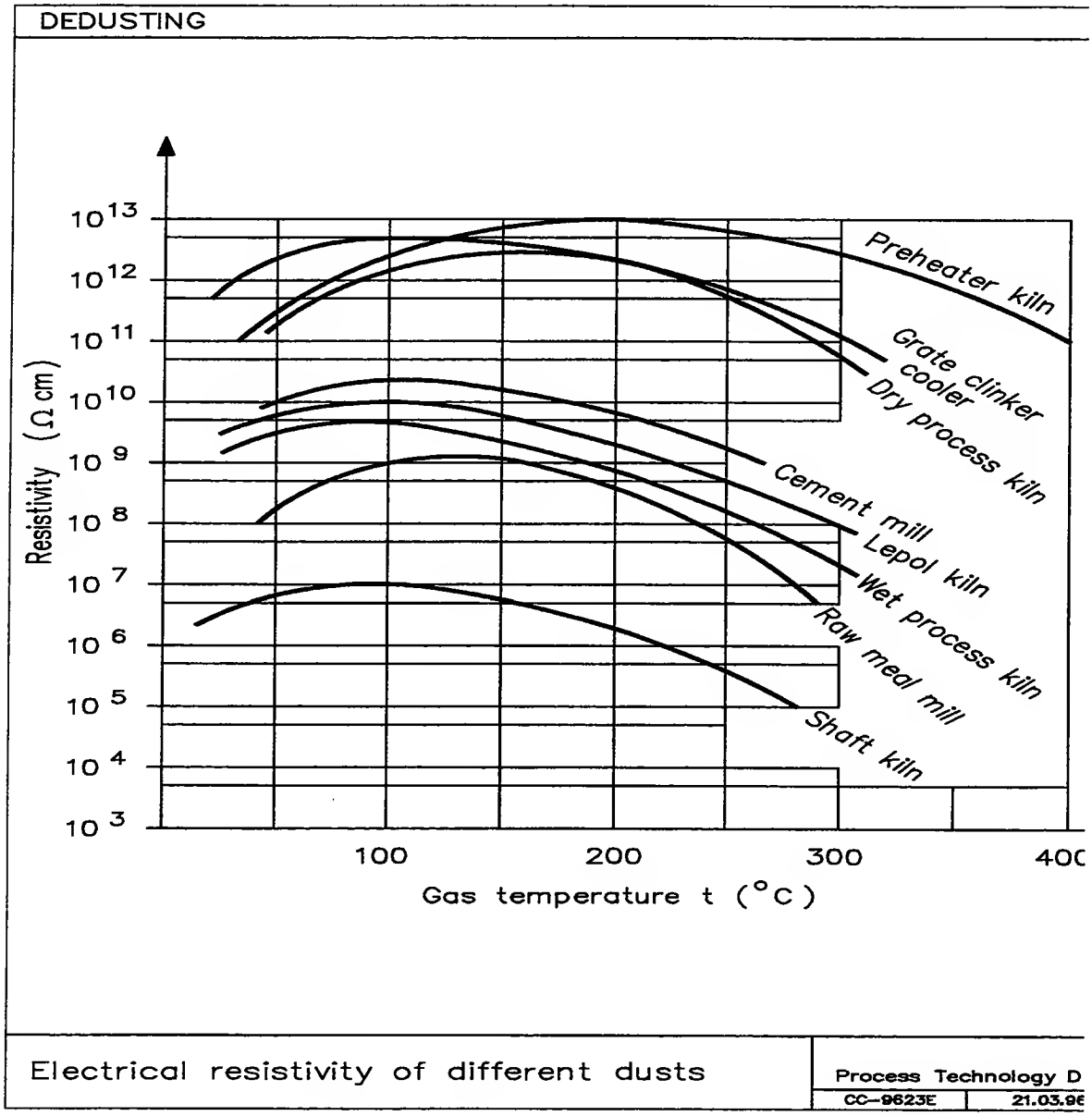
1.3.4.1 Electrical Resistivity of Dust

The electrical resistivity of the dust particles plays a very important part in the precipitation process and depends mainly on the type of the dust, the gas temperature and the gas humidity.

Figure 8 Dust resistivity in function of temperature and dust source



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Three ranges of electrical resistivity can be distinguished:

- less than 10^4 Ω cm
- 10^4 to 10^{11} Ω cm
- more than 10^{11}Ω cm

For particles having a resistivity of less than 10^4 Ω cm the electrical conductivity is so high that although they are charged in the normal manner and move normally under the influence of the electrical field, the attainable dedusting efficiency is poor. The reason thereof is that as soon as they reach the collecting electrodes, the electric charge leaks away so rapidly that the particles are repelled into the gas stream and most likely escape with the outlet gases.

Dust types belonging to the range comprised between 10^4 and 10^{11} Ω cm show a favorable discharge behavior. This means that neither particle repulsion nor back-ionisation occurs, i.e. the particles are nicely deposited on - and sufficiently attached to the collecting electrode.

Cement industry dusts usually belong to these "easily" separating dusts. Dusts stemming from long dry process kilns, suspension preheater kilns and grate clinker coolers, however, may occasionally develop dedusting problems.

Particles having resistivities of more than 10^{11} - 10^{12} Ω cm can form within a very short period of time an electrically insulating layer on the collecting electrodes leading to the so-called back-ionisation (back corona) effect. With "back-ionisation" already captured dust is forced back into the gas flow and a reasonable dedusting efficiency of the precipitator becomes impossible to obtain.

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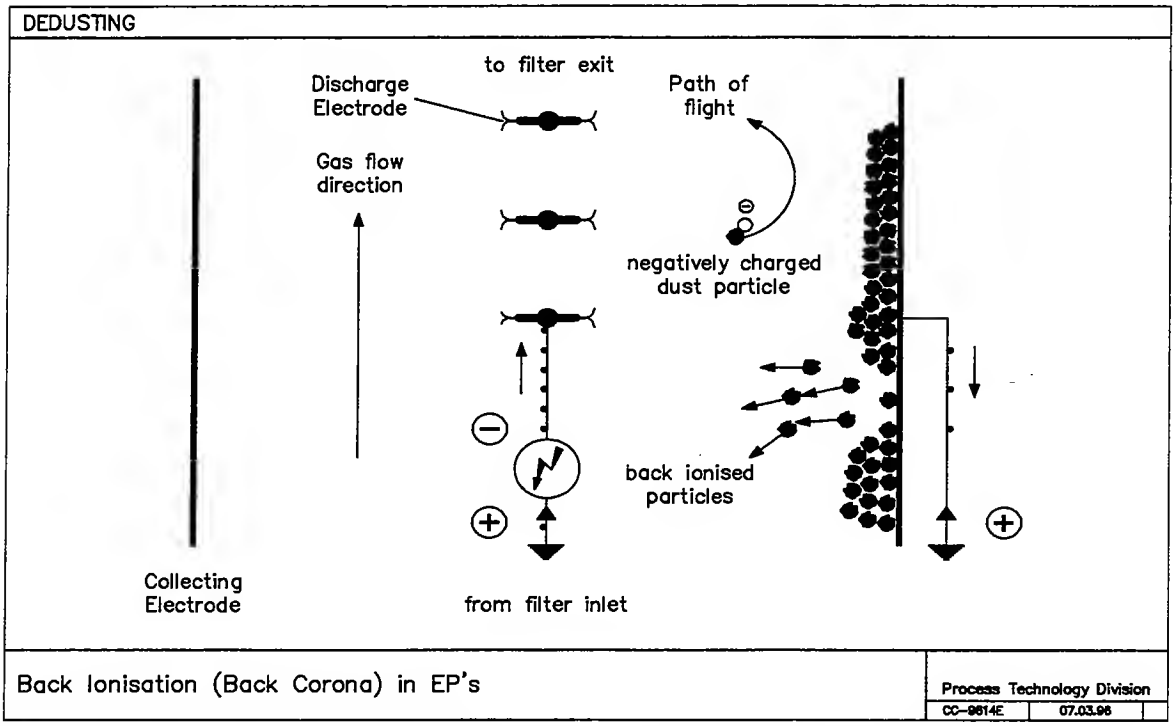
Figure 9 Back ionization of dust particles at high electrical resistivities



Dust resistivity at temperatures below 200°C is primarily determined by the amount of moisture present in the gas. Therefore, a wet kiln will have a much lower resistivity than a standard long, dry or a preheater kiln. In fact that was the reason why a water spray / conditioning tower was added to these kiln systems to tread the exhaust gas.

The variation of the resistivity as a function of the moisture content of the raw gas is due to an extremely thin conditioning layer on the particle surface which modifies the resistivity of the dust.

At higher temperatures (above 350°C) the particles become increasingly conductive and the gas composition ceases to have much effect as a such.

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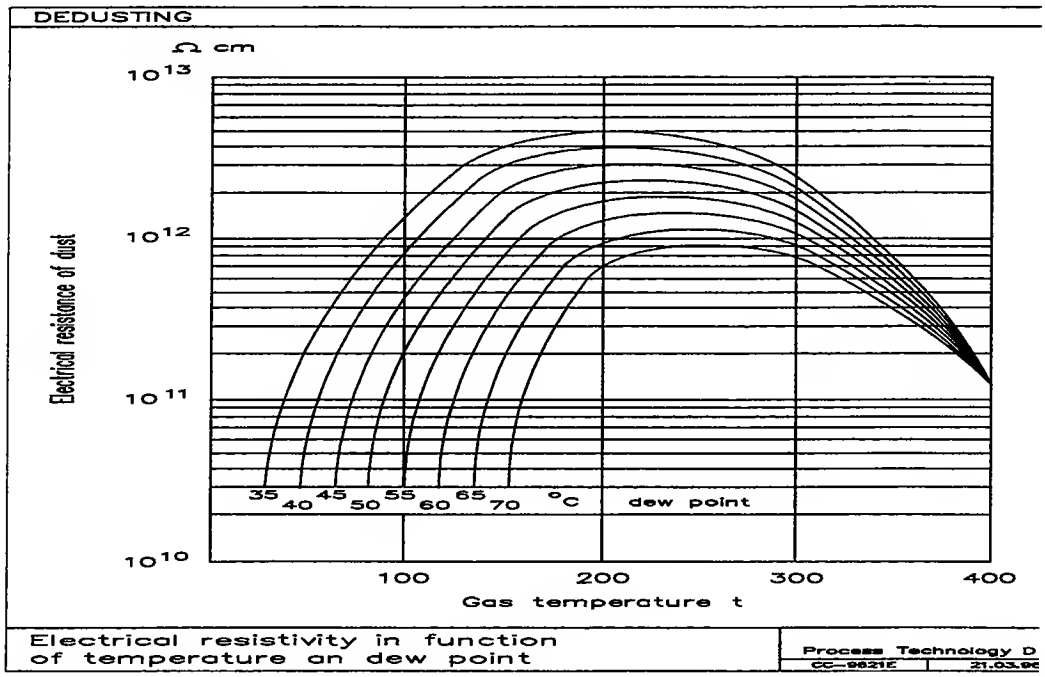
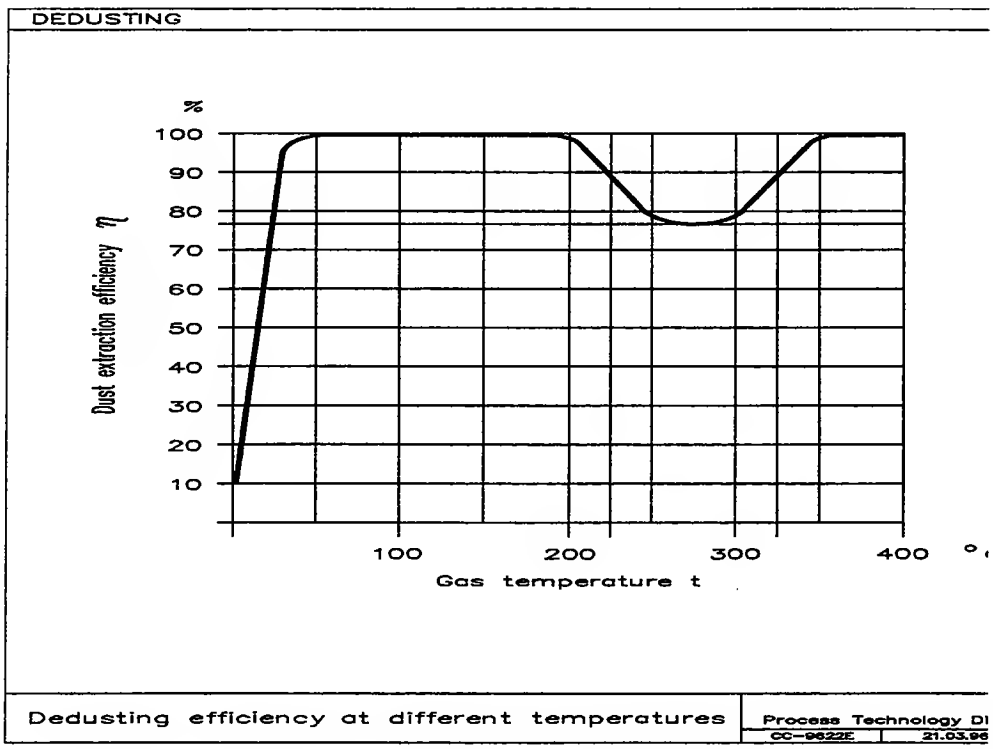
Figure 10 Dust resistivity in function of the temperature and the dew point



At middle-range temperatures of about 200 to 250°C the resistivity curve of some dust reaches a maximum.

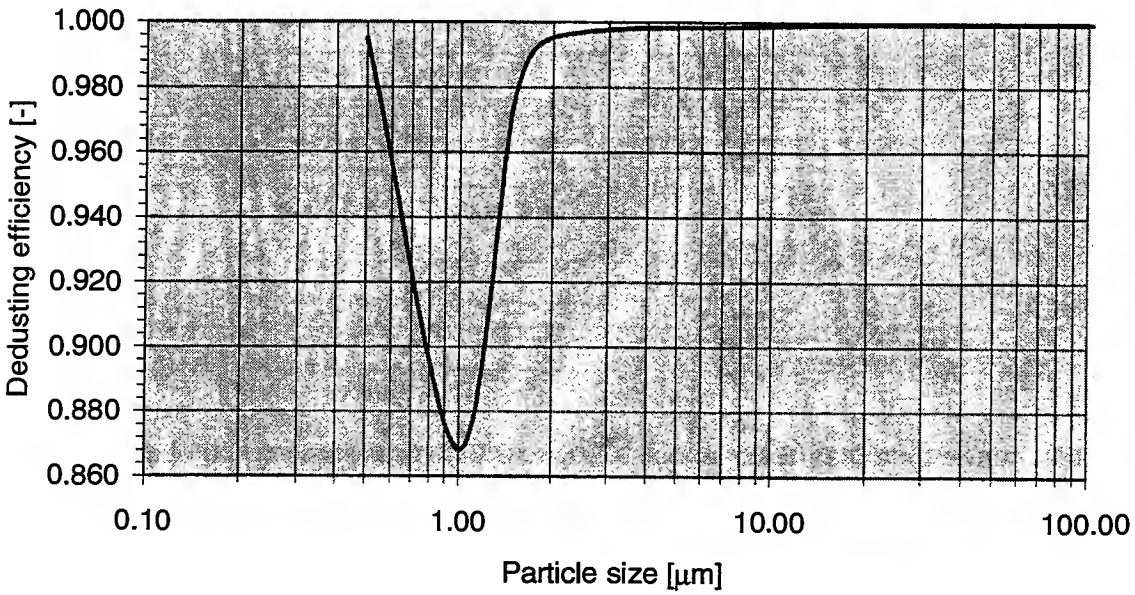
Figure 11 Dust removal efficiency as a function of the EP operating temperature



1.3.4.2 Size of Dust Particulates

According to Stoke's law for particles larger than 1 µm, the migration velocity is directly proportional to the particulates diameter.

Figure 12: Dedusting efficiency in function of the particle size



A dust with a mass mean diameter of 10 microns would require a precipitator only one-third the size of a system collecting dust with a mass mean diameter of two microns. As you can see, ϖ goes down when it is dealing with particulate in the 0.5 micron range and then starts to improve in efficiency when the particulate gets smaller (say 0.05 microns). That has to do with the two principals of particle charging which predominate in a precipitator. Field charging predominance for particulate greater than 1 micron in size and diffusion charging predominates for particulate less than 1 micron in size. That range around 1 micron is kind of a no-man's land where neither field charging nor diffusion charging has much effect. That is why the efficiency drops dramatically and then improves once the particles get even smaller.

What are other consequences for the EP operation based on the correlation between ϖ and particle size:

- EPs are classifying the incoming dust. The coarse particles are found in the first fields and the fine fraction in the last fields.

  This classifying of the dust can be used to extract selectively a dust portion enriched with condensibles like K_2O, SO_3 and heavy metals, thus avoiding generation of larger quantities of "contaminated" dust or enrichment of certain compounds in the process.

- Fine dust particulates and condensibles can be accumulated in the system and reduce the EP efficiency if they are not extracted from the last field.

- The particle diameter of the clean gas dust is generally below 10 µm.

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1.3.4.3 Gas Temperature T

Gas and particulate temperature are usually the same because the particulates are suspended in the gas and the retention time of particulates in the gas is sufficient to reach a temperature equilibration.

The influence of the gas temperature T is mainly:

- Increased dust resistivity Ω at higher temperatures below 250°C (see para 1.3.4.2)
- Decreased dust resistivity Ω at higher temperatures above 250°C (see para 1.3.4.2)
- Increased actual gas flow Q at higher temperatures (see para 1.3.2)

1.3.4.4 Gas Humidity (dew point τ)

The water of the raw gas is originating from:

- Combustion (4 CmHn + (4 m+n) O_2 -> 4 m CO_2 + 2n H_2O)
- Water in ambient air
- Water in raw materials
- Water injection for gas conditioning

The dew point can be calculated as follows:

$$\tau = \frac{336.48}{5.3362 - \sqrt{17.045 + \ln(V_f \bullet P_{tot})}} - 179 \qquad [°C] \quad 7)$$

where:

V_f = Volume fraction of water vapour in the wet gas (m^3 H_2O / $m^3_{wet\ gas}$)
P_{tot} = Total pressure (bar)

As described in para 1.3.4.1 the dew point is influencing the electrical resistivity of the dust particulates at temperatures below 250°C. This is responsible for the increased efficiency of the EP at higher dew points

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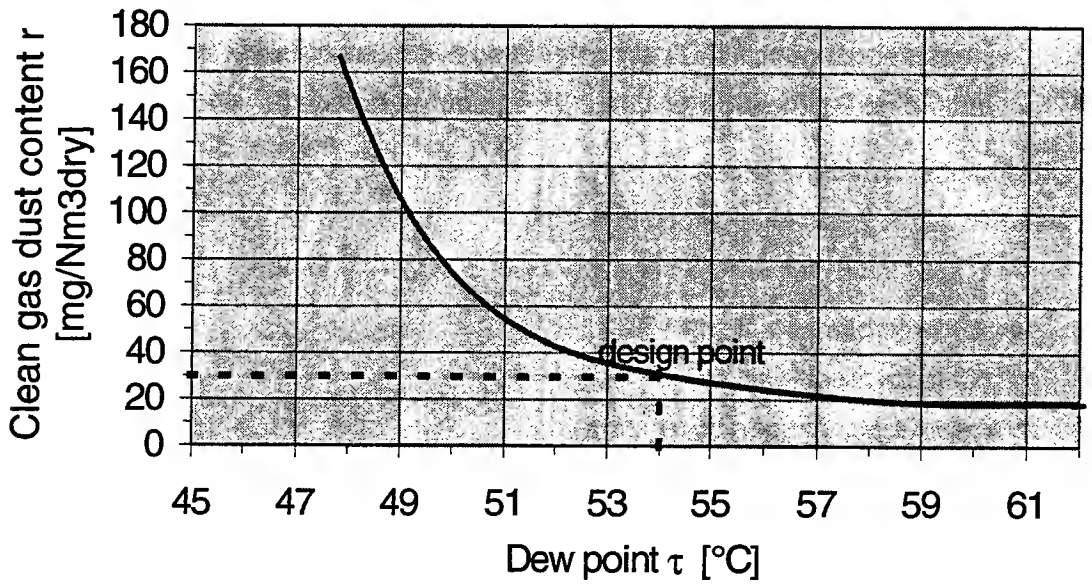
Figure 13 Example for clean gas dust content in function of the dew point τ at temperatures below 250°C



The figure above shows the strong effect of gas dew point τ on EP efficiency η if no back ionization occurs. With back ionization the clean gas dust content r would increase even faster at lower dew points.

A typical example for the influence of the dew point are preheater kilns switching from compound operation (mill on) to direct operation (mill off). When the raw mill is in service, the moisture conditioning (11 % to 12 % at 110°C) of the gas is optimum. When the raw mill goes off line, the spray tower preceding both the raw mill and the EP cannot catch up quickly enough to increase the volume of water to make up for the moisture content lost when the raw mill goes down.

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1.3.4.5 Gas Composition (not including water vapour)

The gas composition of clinker cooler vent air is fairly simple, however, the composition of kiln exhaust gas is a complicated cocktail of many different compounds.

Some compounds like SO_2 can enhance the EP operation by reducing the resistivity of the particulate surface.

Others like organic compounds or condensible alkalis reduce the EP efficiency. It is assumed that organic compounds attached to the particulate surface can increase their resistivity. Condensible alkalis can occur as very fine particulates < 10 µm significantly reducing the average migration velocity.

Condensibles like chlorides can increase the stickiness of the deposited dust on the electrodes which leads to thicker dust layers on the electrodes. This would increase the total electrical resistivity of the dust layer and therefore reduce the EP efficiency.

1.3.4.6 Gas Dust Load S, r and R

An increased raw gas dust load R has a positive effect on the migration velocity but cannot fully compensate the raise in the clean gas dust content according to equation 4.

An increase of the clean gas dust content r is also increasing the migration velocity. According to the explanation under para 1.3.4.2 the lower the clean gas dust emission is the lower is the diameter of the dust particulates and smaller dust particulates have a slower migration velocity ϖ than larger ones.

Therefore, the required collecting area A is increasing exponentially with the reduced clean gas dust content r (see Fig. 6).

1.3.4.7 Energization of the EP

The collection efficiency of a precipitator is directly related to the total power for all fields on the precipitator. In general, the higher operating power levels that each field can achieve, the higher collection efficiency for that field.

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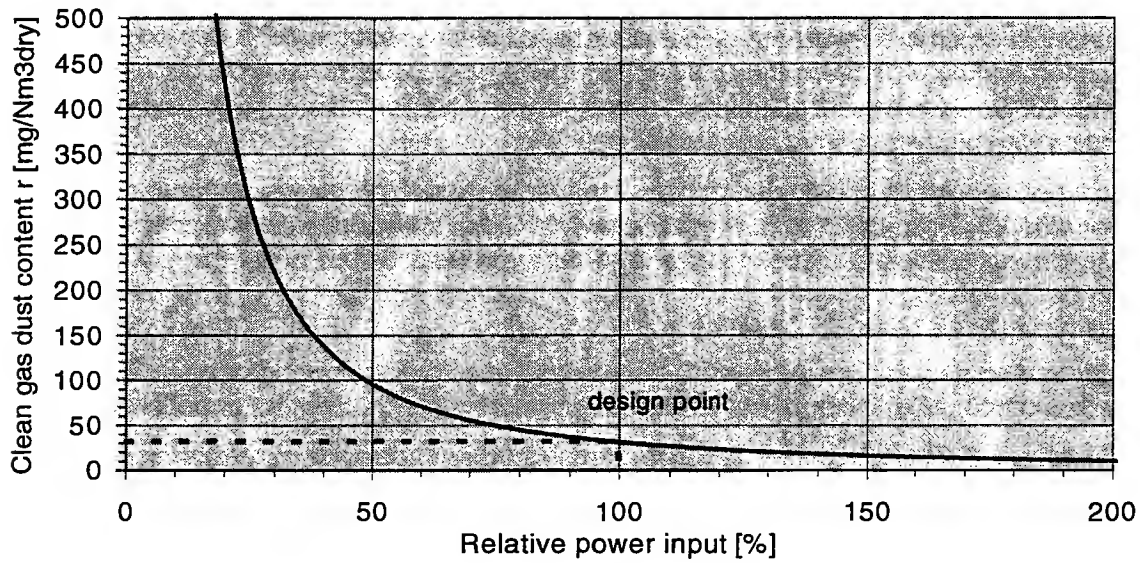
Figure 14 Example for clean gas dust content r in function of the relative power input



Many people believe that a precipitator cannot work (achieve power levels) unless the gas is loaded with dust. This question can be easily examined by energizing any field of an EP in air. By that, it is meant that the kiln is not in operation, and that the temperatures have settled to ambient conditions. Furthermore, the precipitator is not bottled up and dampers are open, allowing for a natural stack draft through the precipitator. It is important to have some air movement in order to obtain a good "air load".

When a precipitator is energized in air, the following results could be obtained:

Table 1: Example of energization of an electrical field of an EP under pure air (without dust)

Precipitator Secondary Voltage (kV)	Precipitator Secondary Current (mA)
0	0
1	0
5	0
10	0
15	0
16.5	1
24	100
28	200
30.6	300
33.5	400
34	500
35.2	600
36.3	700
36.8	750

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Actual results are dependent on T/R set size, type of high voltage electrode, and the electrical clearance between the electrodes.

The mA readings are synonymous with the actual current flowing in the precipitator. Current flowing in a circuit is equivalent to the number of electrons that are moving past the point in that circuit.

For current to flow in a precipitator, that means that electrons need to flow from the discharge electrode to the collecting electrodes in the precipitator. That means that the air in the precipitator must become a conductor. It is easy to think of the various conductors and realize that an insulator is a very poor conductor, a piece of copper wire is an excellent conductor, and an energized precipitator is somewhere in between. The air load demonstrates that current does not start to flow in a precipitator until (in this case) a voltage of 16.5 kV is achieved. That voltage is referred to as the corona onset voltage.

With moderate increases in voltage, a correspondingly increasing current results. If the alignment is correct between the electrodes in the precipitator, then the air load test should achieve either the primary or secondary current rating of the T/R set being energized. In the above example, we ran out of secondary current (705 mA) first.

Therefore, in order to get corona discharge in a precipitator, dust particles are not required. However, the concentration of particulate has a dramatic effect on the power levels in the precipitator. The term "space charge" is used to indicate a precipitator field that is collecting a significant number of fine particles or a heavy concentration of large particles. For our example, we will examine the latter, which is a common occurrence in cement plant precipitator applications.

Space charge - high dust concentrations

As we saw in the section on air load, since there are no particles (dust) in the inter electrode space, there can be no space charge. However, with the influence of a large concentration of large particles, see what affect it has on these two wet process cement kilns. Kiln No. 1 has a cyclone mechanical collector in series with the precipitator, whereas kiln No. 2 does not. The automatic voltage controls for those two precipitators were found to be operating as follows:

Table 2: Energization of two EPs with different dust loads Kiln No. 1: low dust load, kiln No. 2: high dust load

Unit	Amps	Volts	mA	kV	kW	Sparks/Minute
Kiln No. 1-1	123	337	664	50.1	27	0
Kiln No. 1-2	142	247	758	36.2	23	0
Kiln No. 2-1	9	232	39	57.5	1a	20
Kiln No. 2-2	16	324	71	52.0	2	14
Kiln No. 2-3	115	465	940	48.0	38	3
Kiln No. 2-4	120	346	924	35.1	28	0

Because kiln No. 2 does not have the mechanical collector preceding it, the dust loading (concentration) is significantly higher than kiln No. 1. The voltage control readings show the affect of space charge. Space charge is indicated by high voltages, but more importantly, by extremely low current. It is the absence of current flow that can be of significance.

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When asked what is the more important parameter, precipitator voltage (kV) or precipitator current (mA), often times people will say kV. They are partially correct in most cases, but not in this case. Precipitator voltage is responsible for pushing the dust particles toward the plates. Current is responsible for keeping them there. So although kiln No. 2, field 1 has a lot of pushing forced, (57.5), it has no holding force. Most of the dust re-entrains onto the next field.

The other important point to note is that sparking in a precipitator (an electrical breakdown of the gas) is directly related to the precipitator voltage levels. That is why inlet fields have sparking (because of the high kV) whereas outlet fields sometimes do not.

If one looks at the flow of current from the transformer / rectifier to set to the high voltage electrodes through the dust laden gas, to the collecting plate and back to the T/R set (through earth ground) as shown in Fig. 15 the effect of the ion mobility may become apparent.

The air load demonstrated that in air without dust, the main current carriers are the free electrons and the negative ions. These two characters can be compared to running backs on a football team. They are very swift moving and seek the holes, and the mA meter counts a lot of them during an air load.

With the introduction of dust into the precipitator, the ion mobility changes dramatically. The charged particles, which move very slowly, establish a "particulate space charge" in the inter electrode space. Fig. 15 gives an idea of their relative velocity.

The affects of high space charge can be both influential and detrimental. On the positive side, high voltages created by space charge in turn create higher "electric fields". The electric field is the pushing force against the dust particles, accelerating them towards the collecting plates. Higher accelerations toward the collecting plates can result in increased efficiencies.

However, as in our example in table 2, kiln No. 2 was operating with very low current levels. Therefore, the space charge enhanced the particulate collecting field (high voltages), but also contributed towards a suppression of the corona current. Corona current directly affects particle charging. The higher the particle charging ensure that the dust loss due to particle re-entrainment is diminished. If the corona is suppressed, this can promote re-entrainment. That is the case on kiln No. 2.

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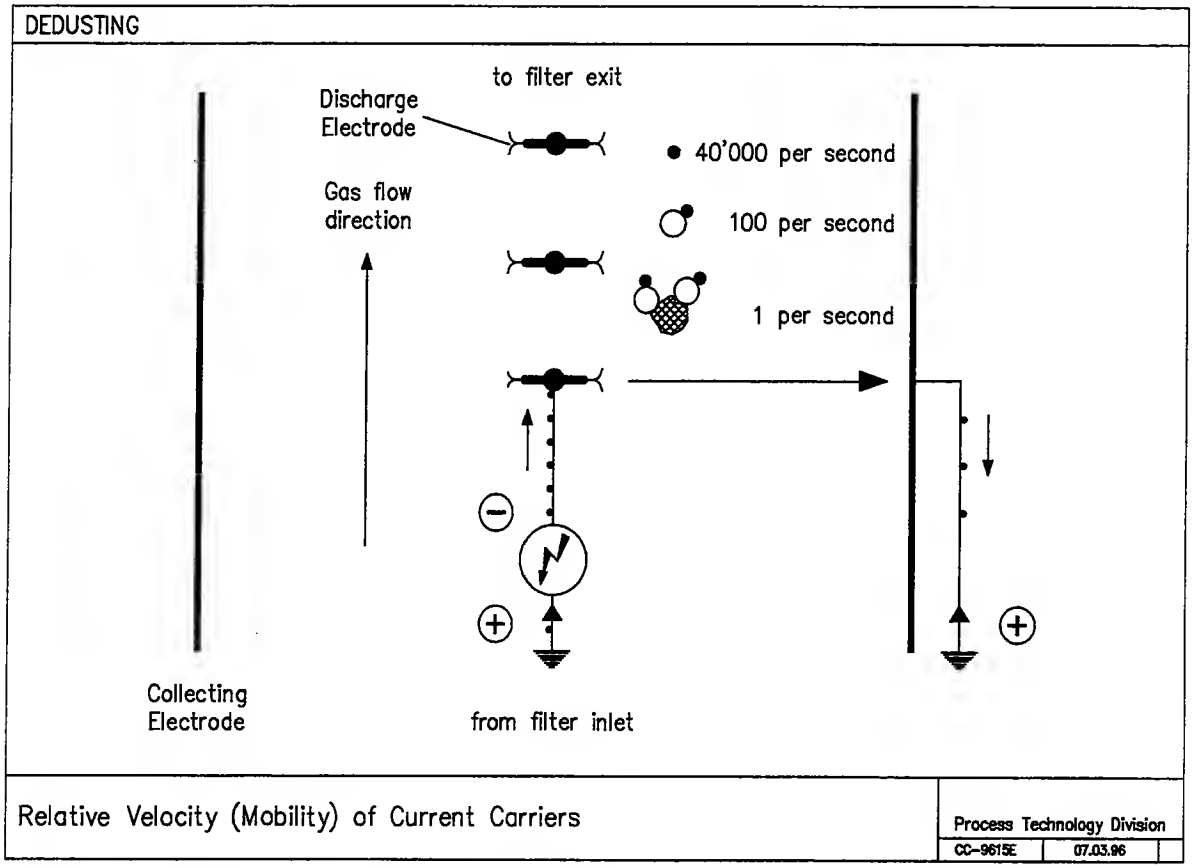
Figure 15 Relative velocity (mobility) of current carriers



The peak value of the precipitator voltage is limited by the dielectric constant of the gas. The arc-over voltage is the only value which determines the maximum possible precipitator voltage. The total power input and therefore the EP efficiency η are strongly influenced by the applied voltage.

$$P_c = I_m \cdot \frac{U_p + U_v}{2} \qquad 8)$$

where

I_m = Mean secondary current
U_p = Secondary peak voltage
U_v = Average secondary voltage

The factors determining the maximum possible precipitator voltage can change quickly. Therefore, the efficiency of the automatic voltage control, that is adjusting the voltage to operate at the maximum value, is directly correlated with the EP efficiency.

The functioning of HT-rectifiers and automatic voltage control is explained in para 1.4.

The electrical operating behaviour is also changing over the length of the field. Gas turbulence and distribution, dust content and particulate size at the EP inlet are very different from the ones at the EP exit. Therefore, to optimize the energization of the EP the electrodes should be subdivided mechanically and electrically in the length direction.

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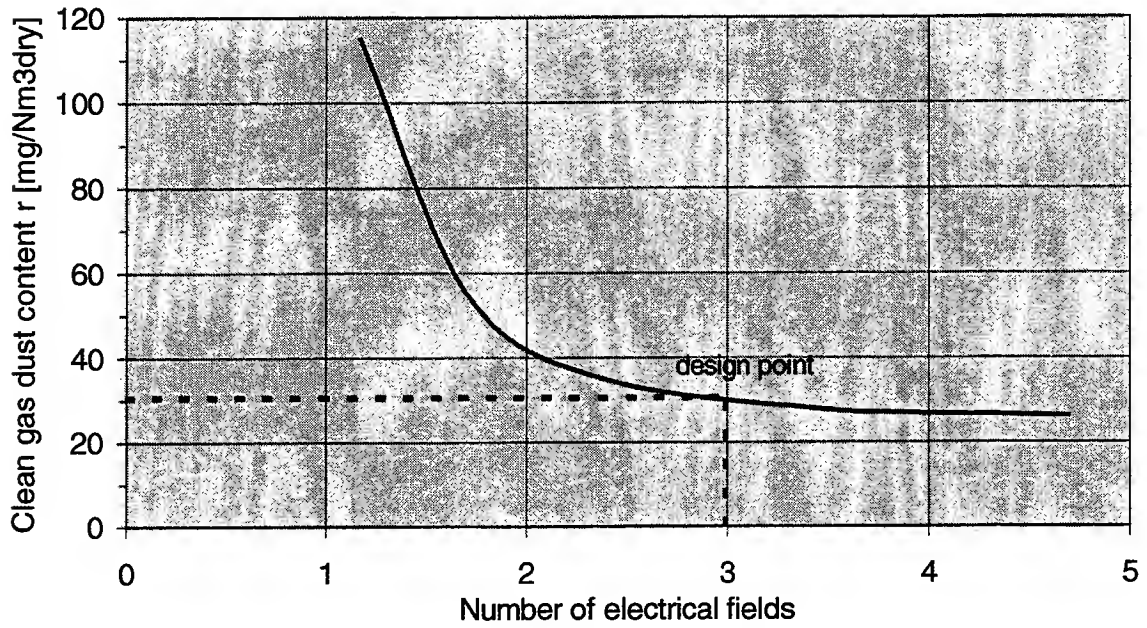
Figure 16 **Clean gas dust content r in function of the number of independent electrical fields at constant collecting area A**



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1.3.4.8 EP Design

The equipment parts with the main influence on the migration velocity are:

- Gas distribution screens
- Electrode
- Electrode cleaning systems

Gas distribution

In general terms the ducting leading to the precipitator and the inlet and outlet funnels should be designed to ensure a proper gas velocity distribution in view of utilizing the whole collecting area and avoiding negative velocity effects. From a practical viewpoint this implies different requirements to the gas distribution in the different parts of the precipitator, and too strictly formulated numerical rules for deviation from uniformity may not be justified.

The inlet gas distribution must be sufficiently uniform to secure a reasonable uniform current distribution. This is especially important for precipitators for processes with high resistivity dust and fine particles. A rule of thumb says that the standard deviation of the gas velocities in the EP should be below 30%.

The velocity profile at the outlet should be specifically selected to reduce the risk of re-entrainment in the bottom region.

Sneakage of dust laden gases around the electrically energized electrode system must be kept at an absolute minimum, in particular at the bottom part of the precipitator. And large eddies in the bottom hoppers caused by the velocity "slip" at the bottom of the electrode system may aggravate the influence from sneakage because particles already picked up by the hopper are swept into the main flow again.

High local velocities may scour away already precipitated dust from the collecting plates. In this case a good gas distribution combined with high average velocity may not be superior to a bad gas distribution combined with low average velocity.

The gas distribution may influence the dust space charge distribution and thereby the current distribution in a separately energized precipitator field. In areas with low velocities or, in extreme situations, areas with recirculating flow, the particle concentration will be much lower than in corresponding areas with higher velocities. Consequently the power input will be limited in the high velocity areas causing a reduction in overall efficiency. In particular with high resistivity dust such uneven current distribution will cause back ionization and frequent sparking, resulting in lower average voltage and current and increased dust re-entrainment. Due to the turbulence the gas distribution in each separate duct will tend to improve through the precipitator, thus smoothing the dust space charge. However, a skew cross distribution at the field inlet will not be smoothed to the same extent, and so the horizontal gas distribution should be fairly uniform in order to maintain a proper current distribution.

Finally, high local gas velocity, combined with high dust content, can result in erosion of the edges of the collecting plates and other internal parts of the EP. Low gas velocity can cause dust build ups.

The cross section of an ideal EP should be designed to achieve an average gas velocity of

- Kilns 0.8 - 1.0 m/s
- Clinker coolers 0.7 - 0.9 m/s

The gas velocity should not drop below 0.5 m/s to maintain a suitable gas distribution.

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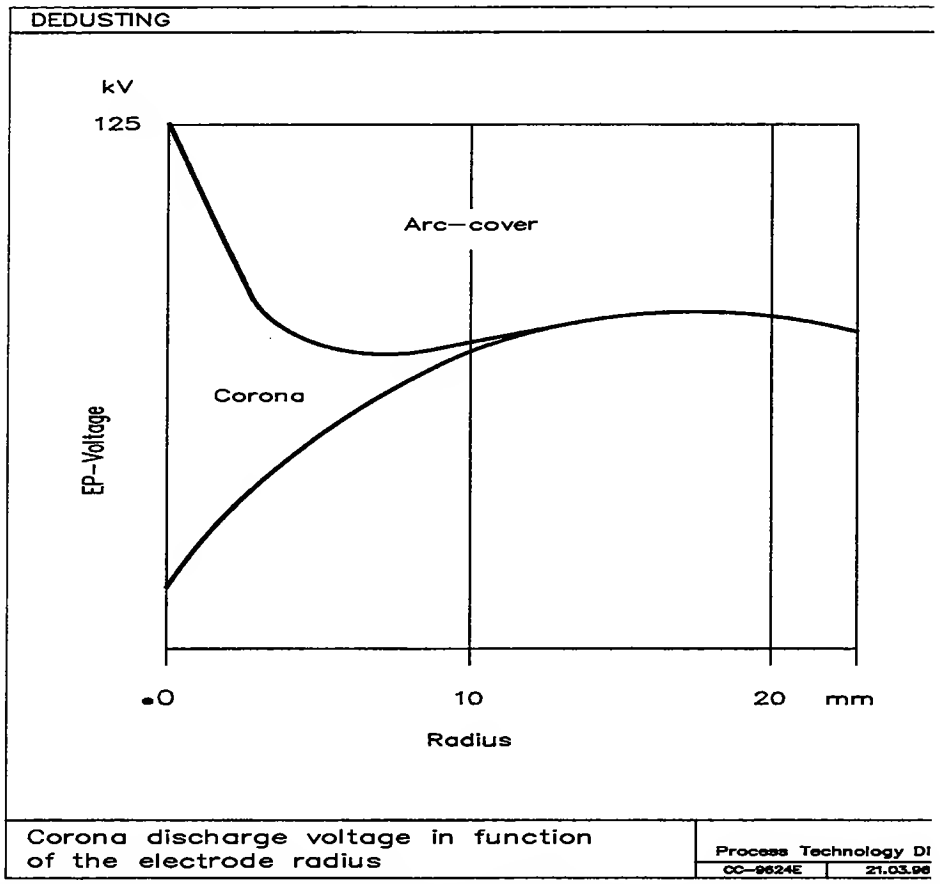
Electrode design

The electrodes have two duties. First emission of electrons (discharge electrode) and second the collection of the dust (collecting plate).

The energization of the fields or in other words the supply with voltage and current is influenced by the discharge electrode design. Various electrode designs to achieve optimum voltage or / and current are employed by the suppliers. An important factor is the corona onset voltage which depends mainly on the radius of the electrode (plan strips) or the radius of spike peaks. The corona onset voltage is increasing with the above mentioned radius.

Figure 17 Corona discharge voltage in function of the discharge electrode radius



In applications, where a high current is required (high dust load, low resistivity), the electrode radius should be small. In situations, where current must be reduced and voltage increased (high resistivity dust -> back corona) electrodes with larger radius (without peaks) can improve the efficiency.

Since corona discharge is also greatly affected by dust settling, the discharge electrodes need rapping, which means that their oscillation behaviour is of utmost importance. Best results have been obtained with rigid frame-mounted electrodes or rigid electrodes.

For maximum collection efficiency, the collecting plates must be rigid to maintain the critical spacing between the different electrodes and withstand bowing during operation. At the same time, they must facilitate the efficient transfer of rapping energy for effective cleaning. Not optimum cleaning can amplitude back corona effects and generally reduce the EP efficiency.

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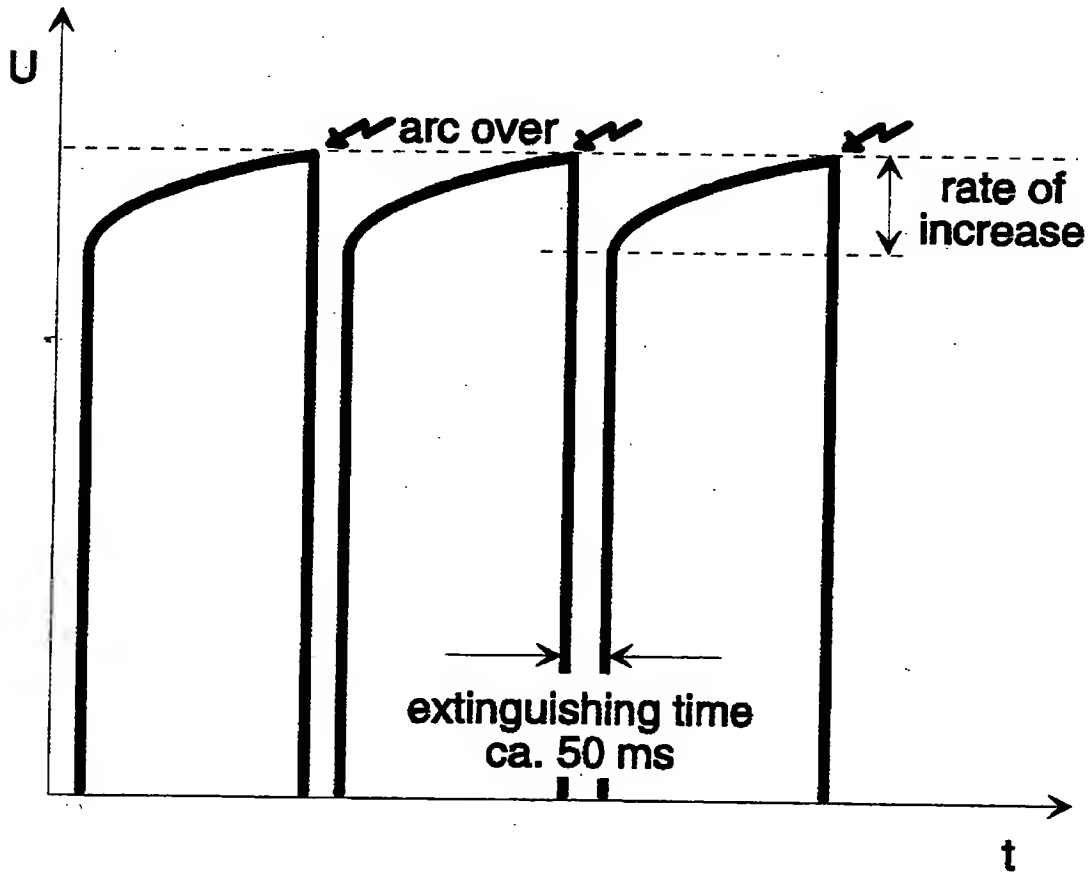
1.4 HT-Rectifier

The High Voltage Rectifiers are responsible for optimum energization of the EP under different operating conditions. Optimum energization means:

- Clean gas dust content r below the target
- Minimum energy consumption

The precipitator energization has a very strong influence on precipitator collection efficiency. As a result of this recognition, the microprocessor-based controller for precipitator high voltage power supplies have in recent years become the general standard. These programmable, fast reacting, digital controllers can implement sophisticated control strategies through their monitoring of secondary current and voltages, including differentiation of reactions according to type of arc or spark in the precipitator, arc quenching, fast voltage recovery after arcing without reignition of the arc, automatic current limitation to the nominal current at overload or short circuit conditions and operation at a precipitator current level just below the onset of "back corona". They continuously control flash-over rate and power input to the precipitator for optimum performance.

Figure 18 Automatic voltage adjustment. Behavior of EP voltage at constant arc-over limit



Most microprocessor-based transformer / rectifier controllers have or can easily be supplemented with an option for semi-pulse energization, as described in the following:

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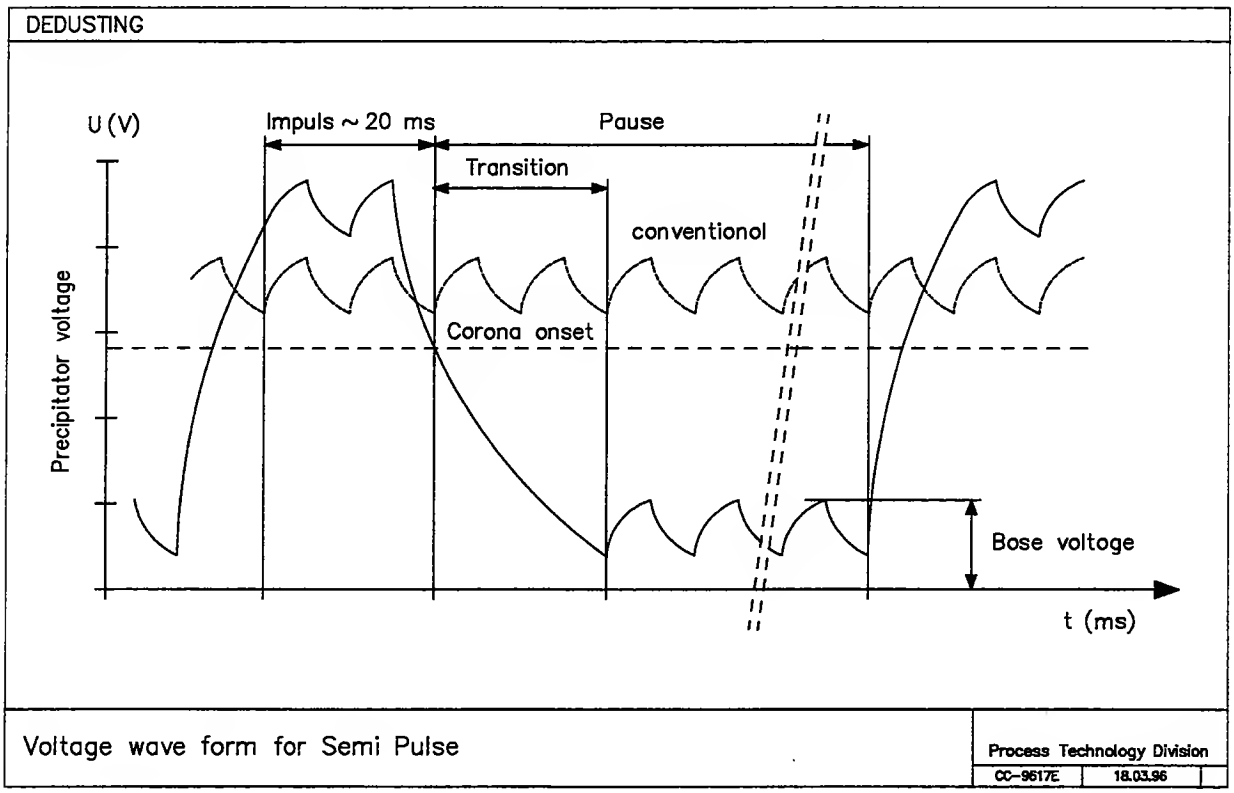
1.4.1 Semi-Pulse Energization

An inexpensive method for reduction of precipitator power consumption, and in some instances also improvement of precipitator performance, is also known under various trade names such as semi-pulse intermittent energization and energy-control.

Semi-pulse energization is implemented at a conventional thyristor controlled full wave transformer / rectifier simply by suppressing for instance two out of three, or four out of five half waves. The ripple of the precipitator voltage hereby becomes more pronounced than with conventional energization, resulting in a voltage wave form that resembles a DC base voltage superimposed with long duration pulses.

The intermittent nature of the corona discharges gives this form of energization certain properties resembling those of the later discussed pulse energization. Semi-pulse has, in some cases, been able to improve the performance of precipitators operating with medium to high resistivity dust, but as a rule not to the same degree as pulse energization. Its main advantage is the resulting power savings. Power saving up to 90% and emission reduction of up to 50% were reported.

Figure 19 Voltage wave form for semi-pulse energization



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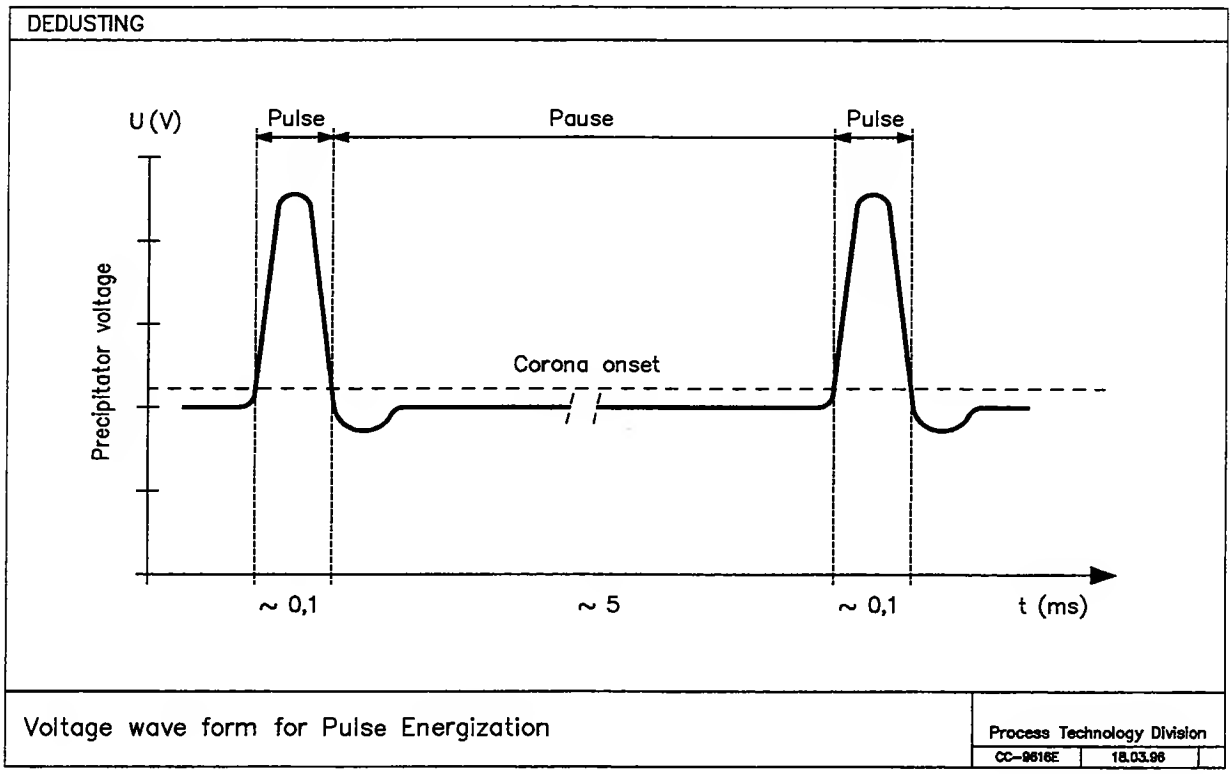
1.4.2 Pulse Energization

Advances in high-power switching technology in recent years have made it possible to develop pulse energization systems with sufficient reliability and capacity to energize large precipitators.

With pulse energization short duration, high voltage pulses are repetitively superimposed on a DC base voltage. Some energy conserving pulse energization systems utilize pulses with a duration in the order of 100 microseconds and pulse repetition frequencies up to 200 pulses per second.

Pulse energization makes it possible to attain more favorable electrical conditions for high resistivity dust than is obtainable with conventional DC energization. Pulse energization, therefore, can successfully be used to improve the performance of an existing precipitator operating with high resistivity dust or to reduce the size of a new precipitator installation for a high resistivity application, as for instance with the so-called "hot" precipitators for kilns. Power saving of up to 90% and emission reduction of up to 60% were reported.

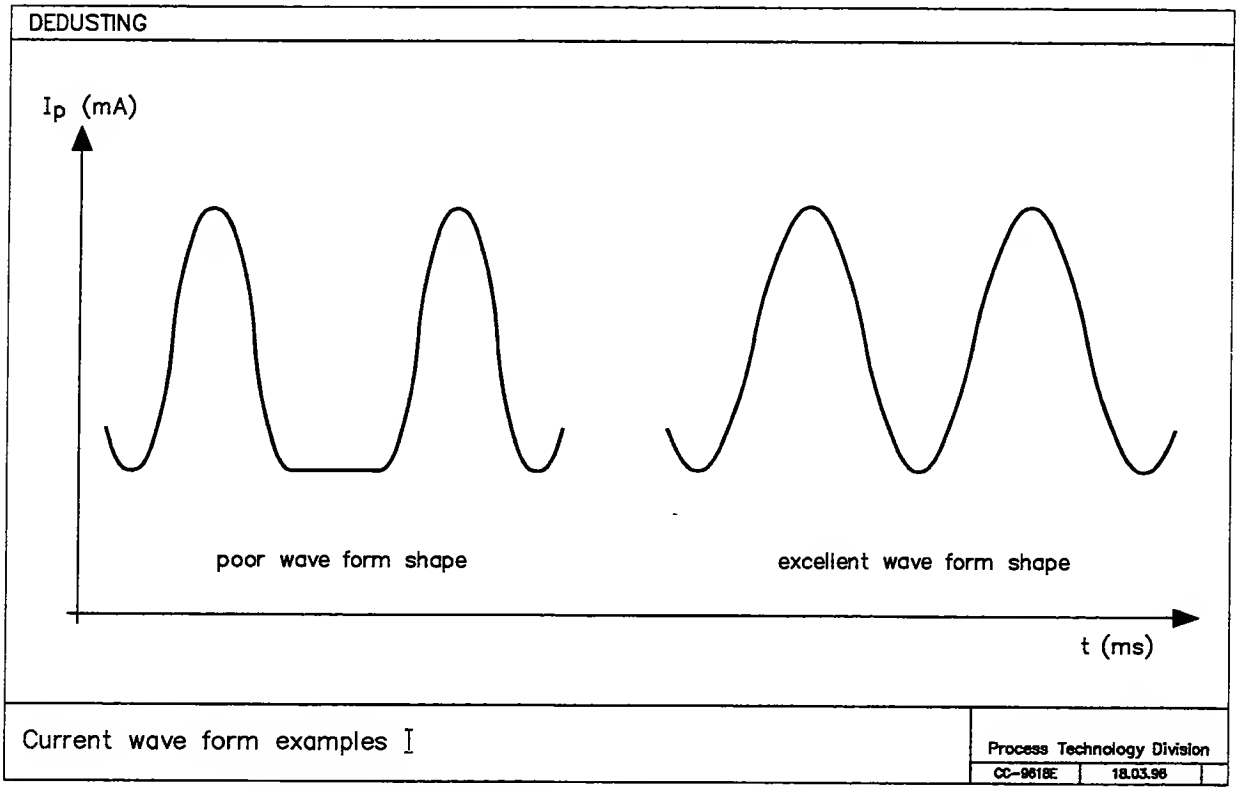
Figure 20 Voltage wave form for pulse energization



1.4.3 Improvement of Voltage and Current Wave Form Shape

Modern precipitator power supplies include silicon controlled rectifiers (SCR's) and current limiting reactors (CLR's). SCR's and CLR's are designed to produce an optimum energization of the EP at one specified process condition. Since it is known that this condition can change very frequently, the current and voltage input have to be adjusted continuously. Operation of CLR's at conditions which are not according to the design specifications can produce a poor current wave form shape (poor form factor) which leads to a reduced power input. This can be corrected with a variable inductance current limiting reactor (VI-CLR).

Figure 21 Improving wave form shape with variable inductance current limiting reactor



Another more expensive way to increase the power input is the utilization of a 3-phase energization. The transformer is operated with a square wave voltage with a frequency of 500 Hz. This produces a very flat direct voltage that can under certain circumstances result in a higher power input. Unfortunately, very little experience is available for this system.

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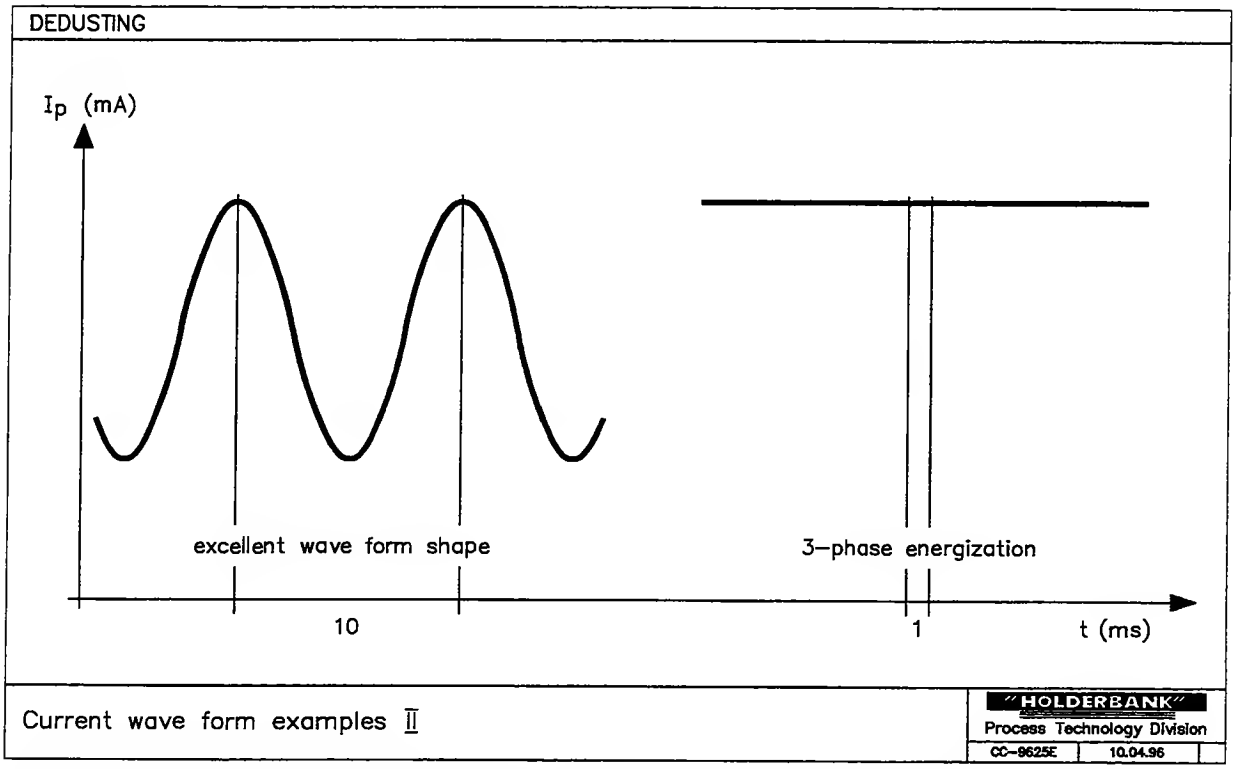
Figure 22 Improving current wafe form shape with 3-phase energization



Intelligent EP control systems do limit the power input if additional power input does not result in significantly reduced dust emission (see Fig. 14).

1.5 Voltage-Current Curves

A voltage-current curve to a precipitator troubleshooter is like a stethoscope to a cardiologist. When a precipitator is running, we cannot see what is happening inside that might affect its performance. However, by a close examination of the relationship between the voltage and current levels in the operating precipitator, one can predict what is affecting performance.

A V-I curve is run by taking the voltage controls to zero then slowly increasing the power levels, recording both the kilovolts and milliamps at convenient intervals (usually 50 mA or 100 mA) until the voltage control sparks over. A curve can then be drawn from the points collected utilizing the "X" axis for the kilovolts and the "Y" axis for the milliamps. Some typical V-I curves for a dry process cement kiln are shown on Fig. 23. Note that the voltage and current corresponding to each field reflects the voltage and current relationships as first shown in Table 2 of our precipitator example.

When there are problems with the operation of the precipitator, Fig. 24, would be more helpful for troubleshooting. For example, the high resistivity dust as indicated by low current levels in the outlet fields may show up as the "moderately high" dust resistivity curve shown on Fig. 24. This short, stubby curve shows corona onset voltage as normal (say around 18 kV), but current level only increases to a very low level as opposed to the way an outlet field should, as shown on Fig. 23.

This contrasts with a misalignment of the electrodes (wire-to-plate spacing) in the precipitator. Misalignment exhibits itself by a very low corona onset voltage (the electrical clearance is decreased), and the spark over.

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These curves can also be utilized to show if there is excessive dust buildup on the high voltage electrodes. Excessive dust buildup exhibits itself almost as if the wire diameter of the high voltage electrode has been increased. Dust buildup on the wire has the same effect of increasing the corona onset voltage from the normal range of 15 - 20 kV on up to 25 to 35 kV. The problem with wire buildup is that you are not able to achieve as high a current as if the wires were clean. Remember, a precipitator needs both high voltage and high current levels.

Figure 23 Normal precipitator voltage - current (V-I) curves

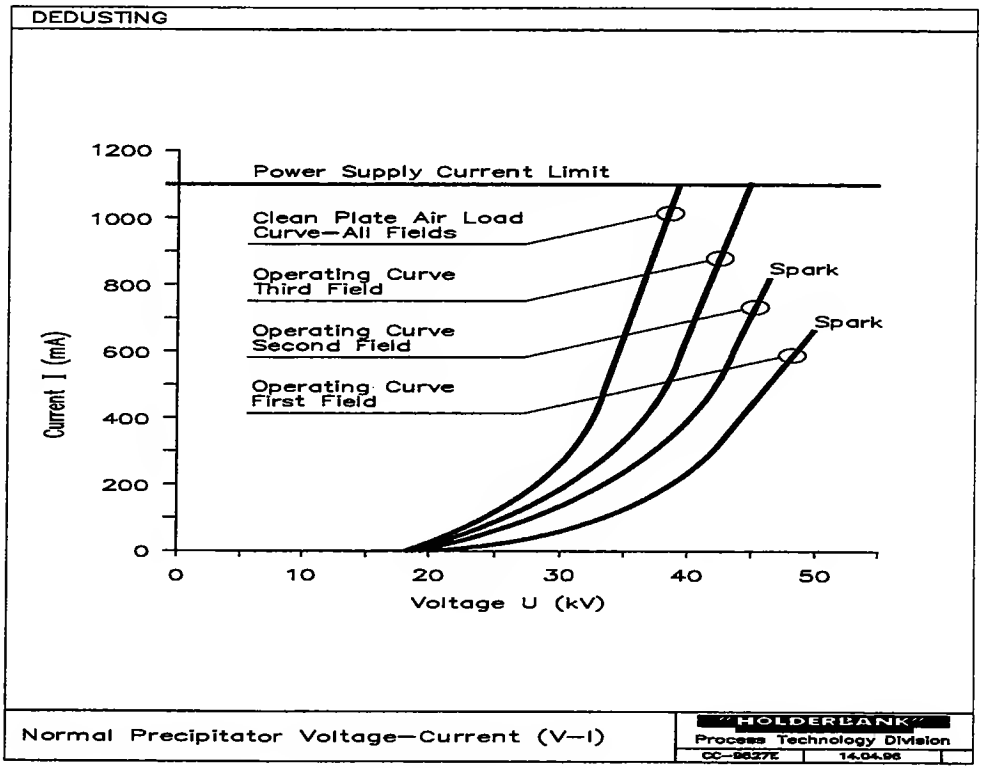

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Figure 24 Abnormal precipitator current-voltage curves

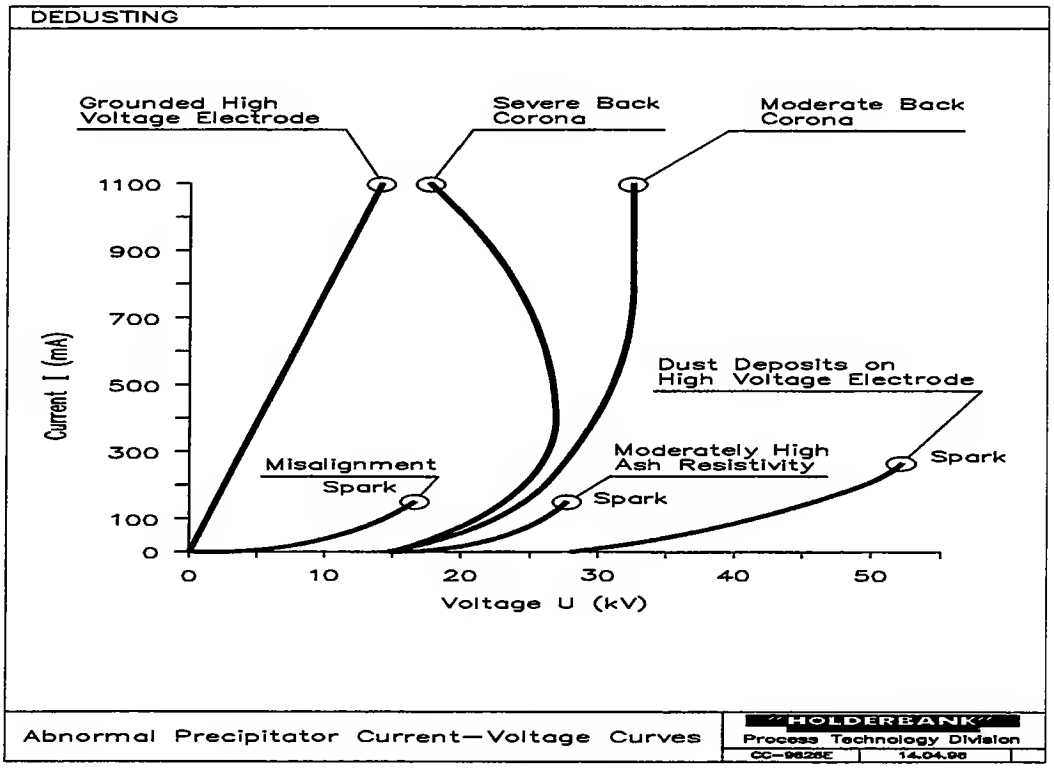

Table 3: Influence of some variables on EP's dedusting efficiency

Variables	Variation	Efficiency	Dust Emission
Raw gas dust content R	↗	↗	↗
Gas flow Q	↗	↘	↗
Collecting area A	↗	↗	↘
Electrical resistivity of the dust Ω	↗	↘/↗	↗/↘
Temperature T1 (<200°C)	↗	↘	↗
Temperature T2 (>300°C	↗	↗	↘
Particle size Ø (> 1 µm)	↗	↗	↘
Humidity (τ)	↗	↗	↘
Organic emission c	↗	↘	↗
Power input P	↗	↗	↘
Standard deviation of gas distribution	↗	↘	↗
Misalignment of electrodes	↗	↘	↗
Speed of the controller for EP energization	↗	↗	↘

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Figure 25 Longitudinal Section of a 2-Field EP (Lurgi)

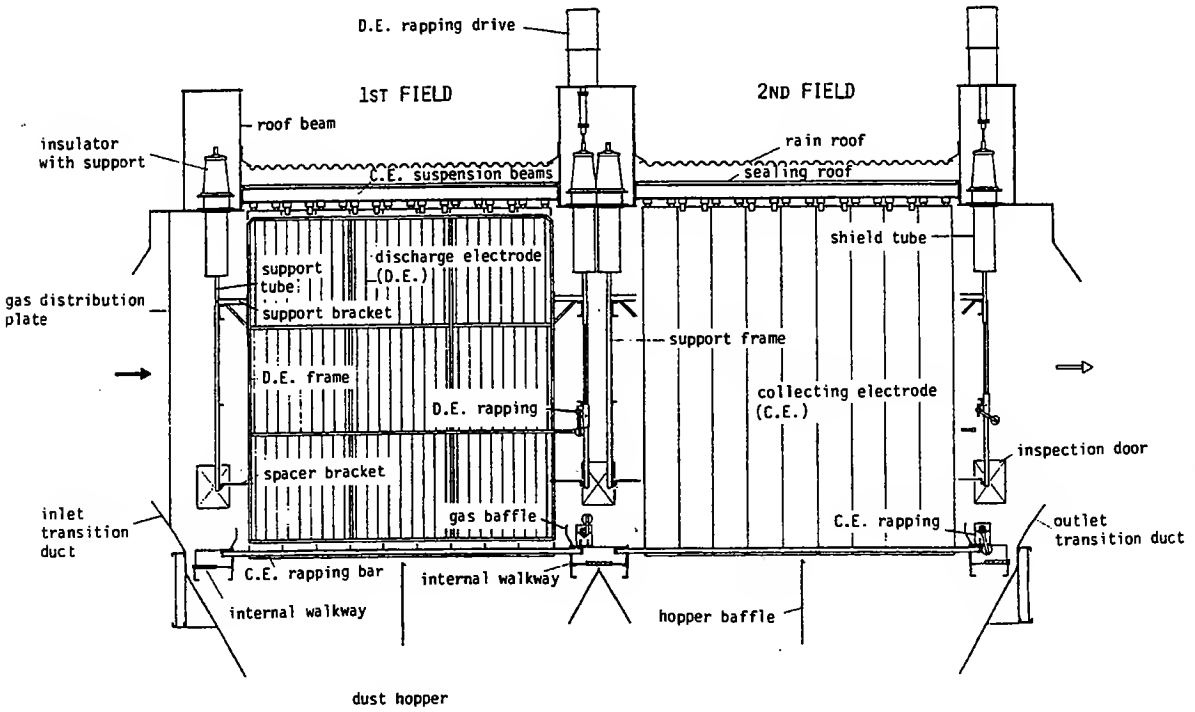

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Figure 26 3D view on a 2-Field EP (ELEX)

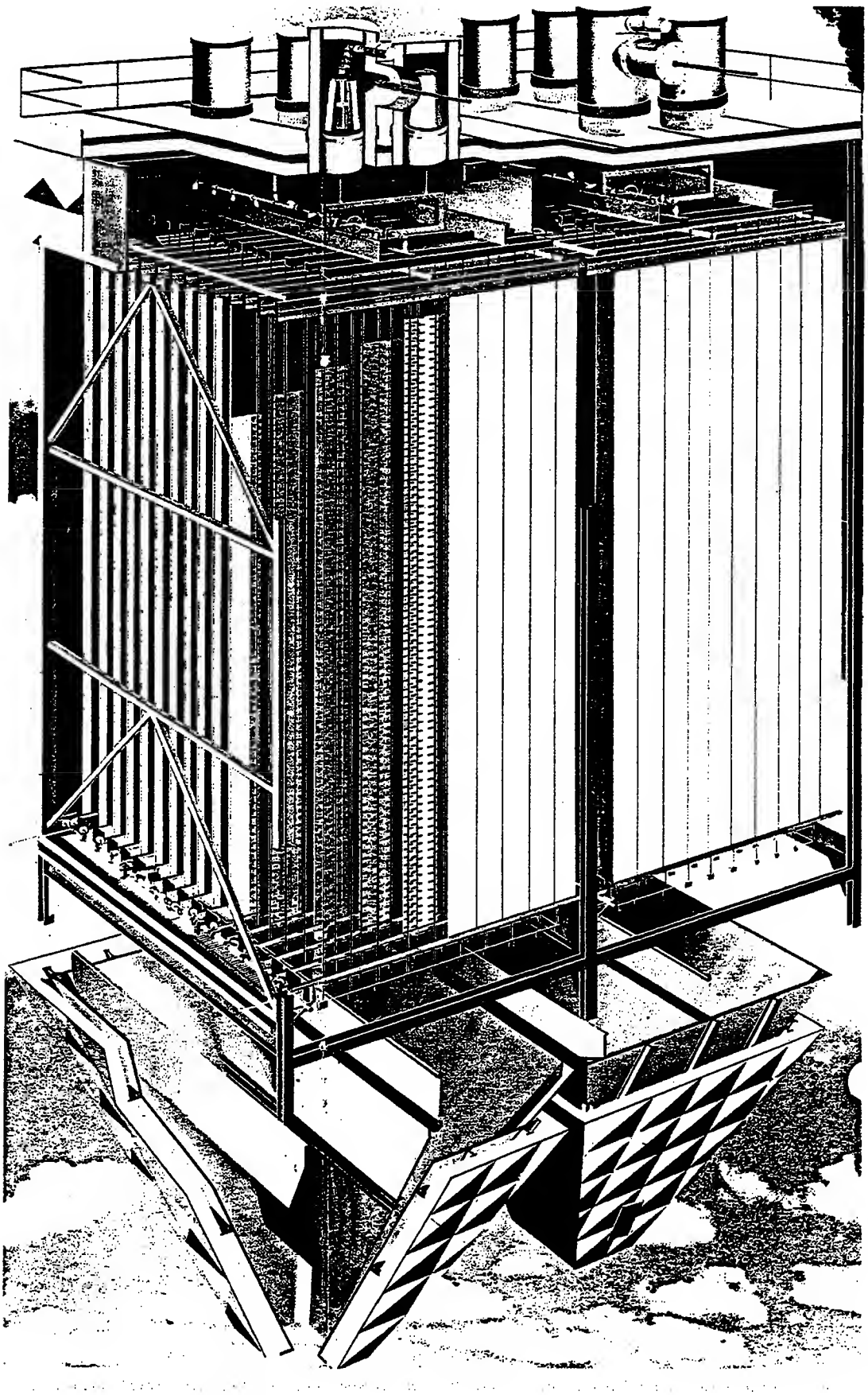

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Figure 27 Insulator Chamber FLS (Type C)

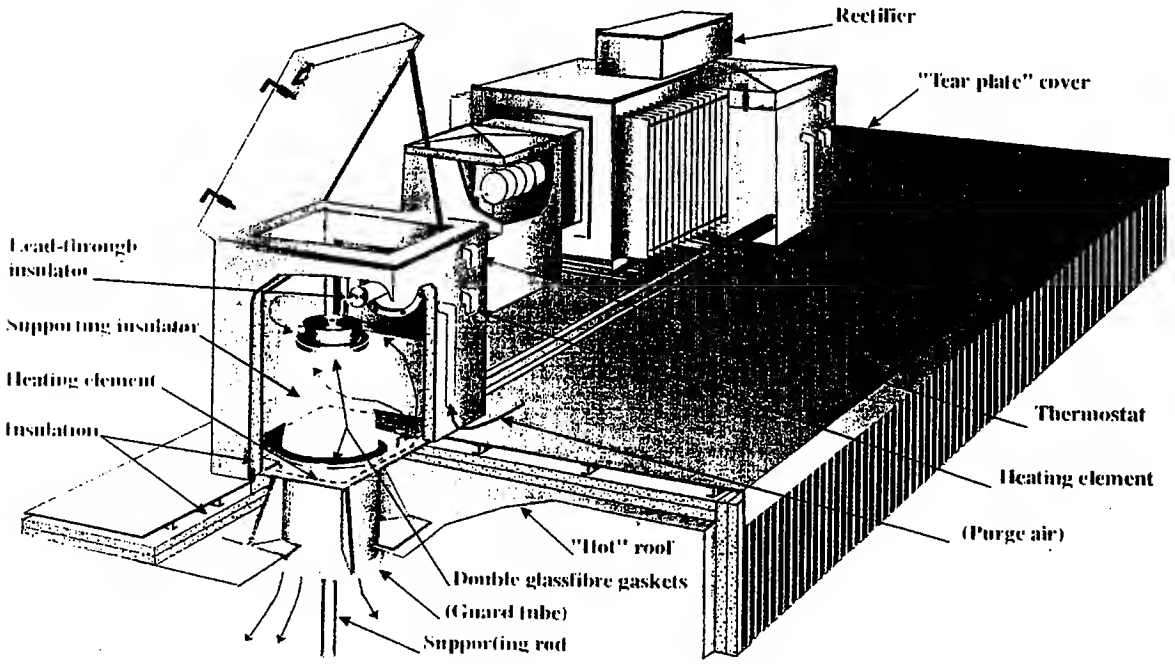

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Figure 28 Electrode System (Lurgi)

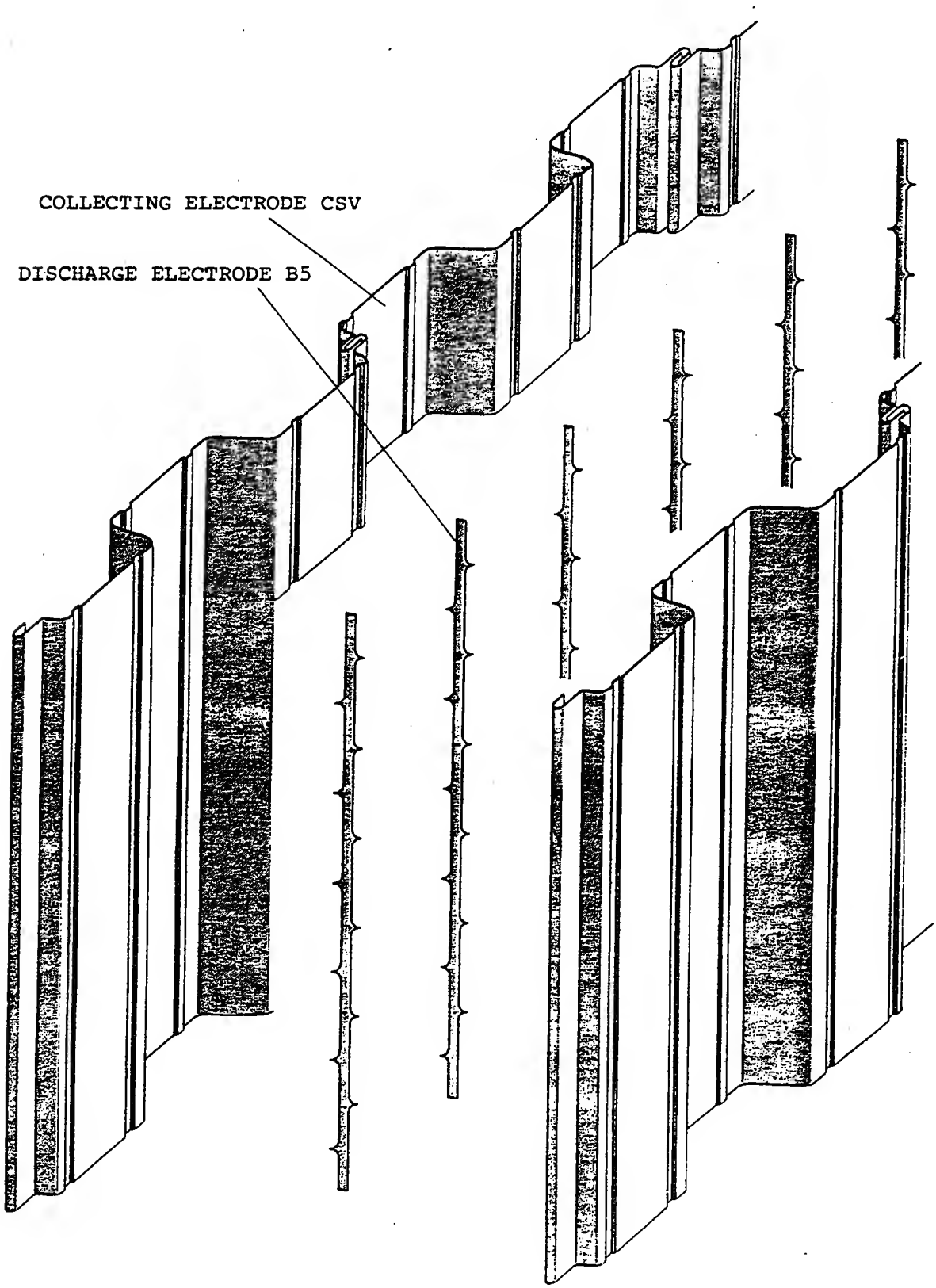

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Figure 29 Collecting Electrode Rapping System (Lurgi)

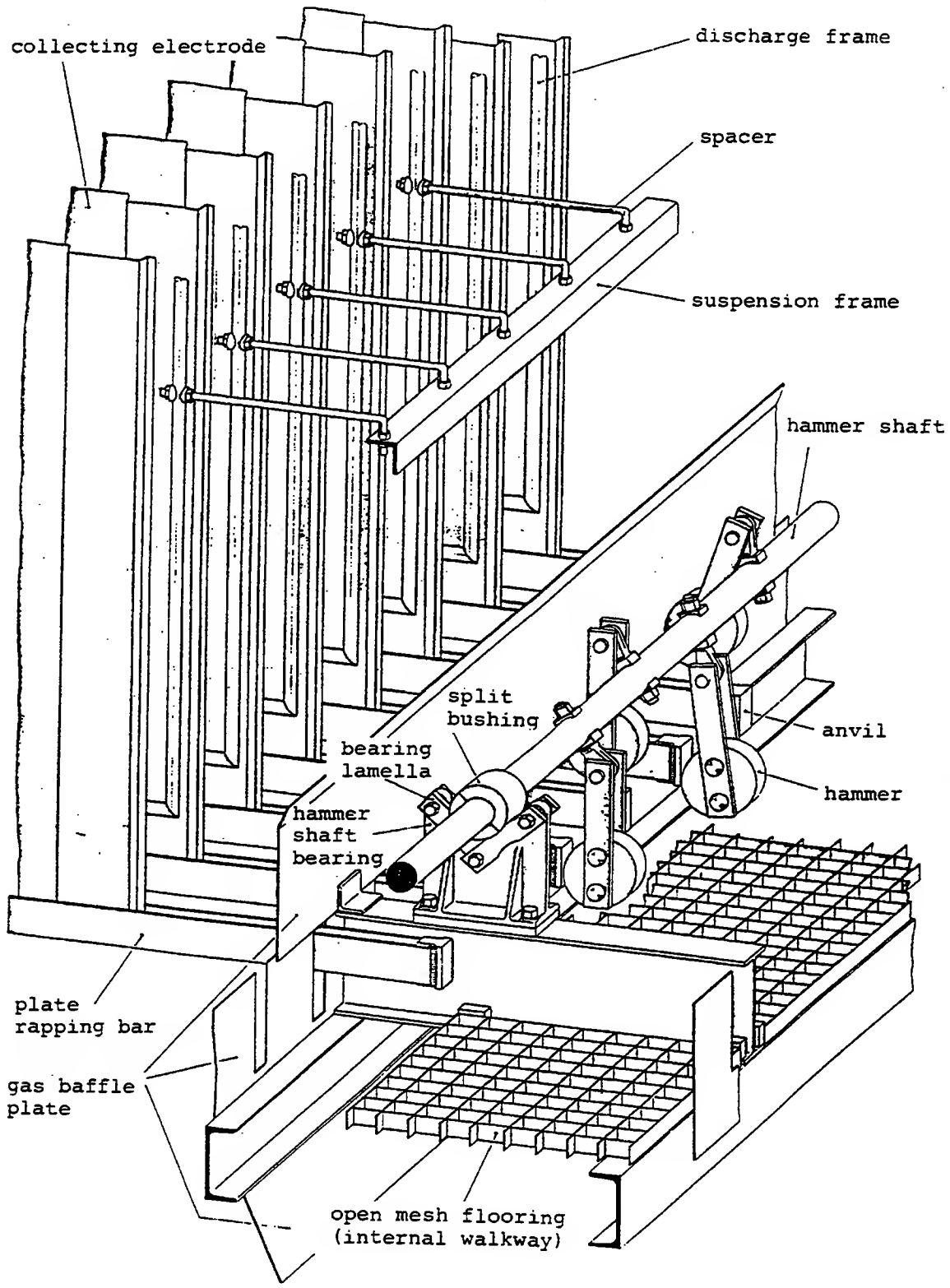

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Figure 30 Suspension and Rapping of Discharge System (Lurgi)

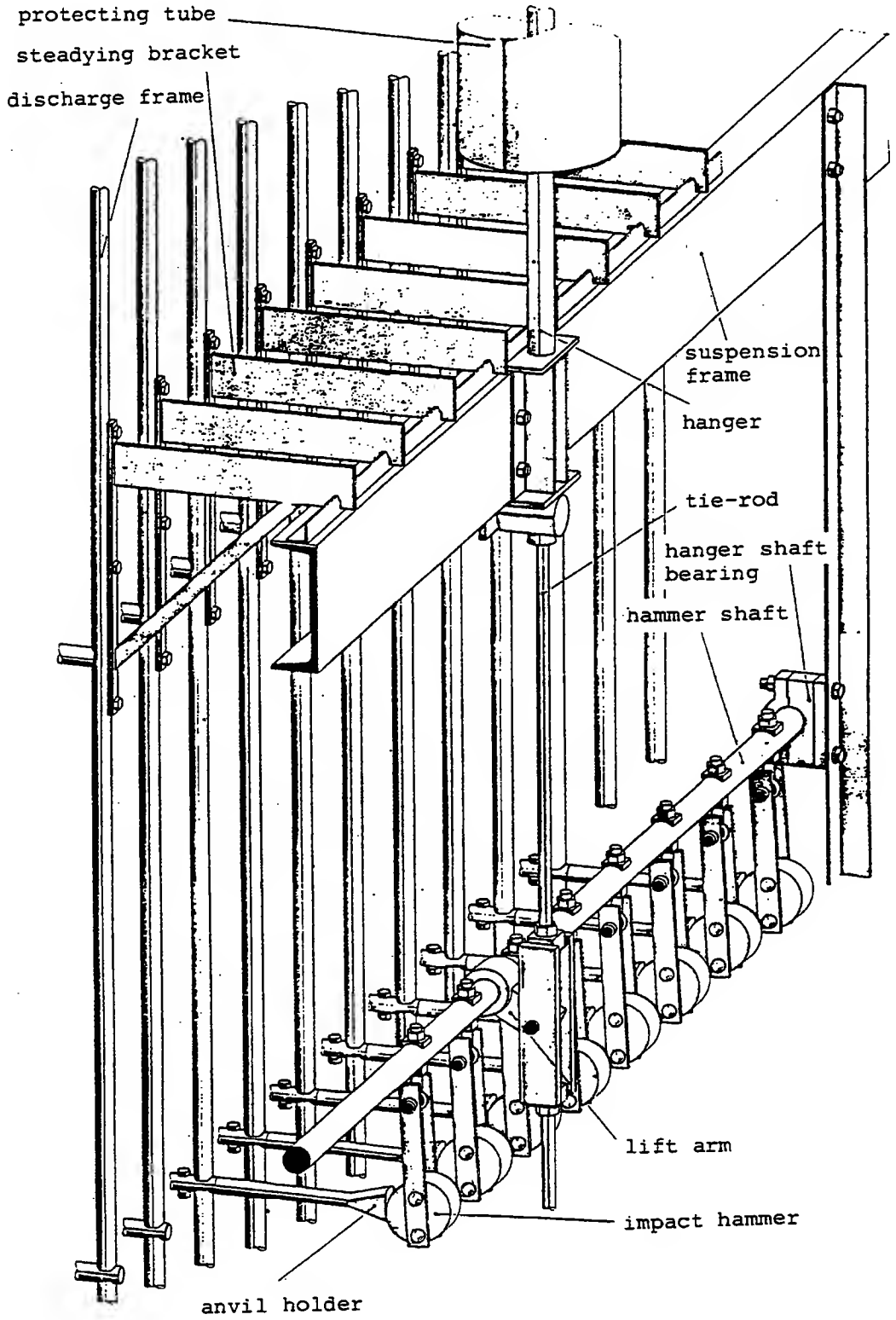


Figure 31 Gas Distribution Screen (FLS)

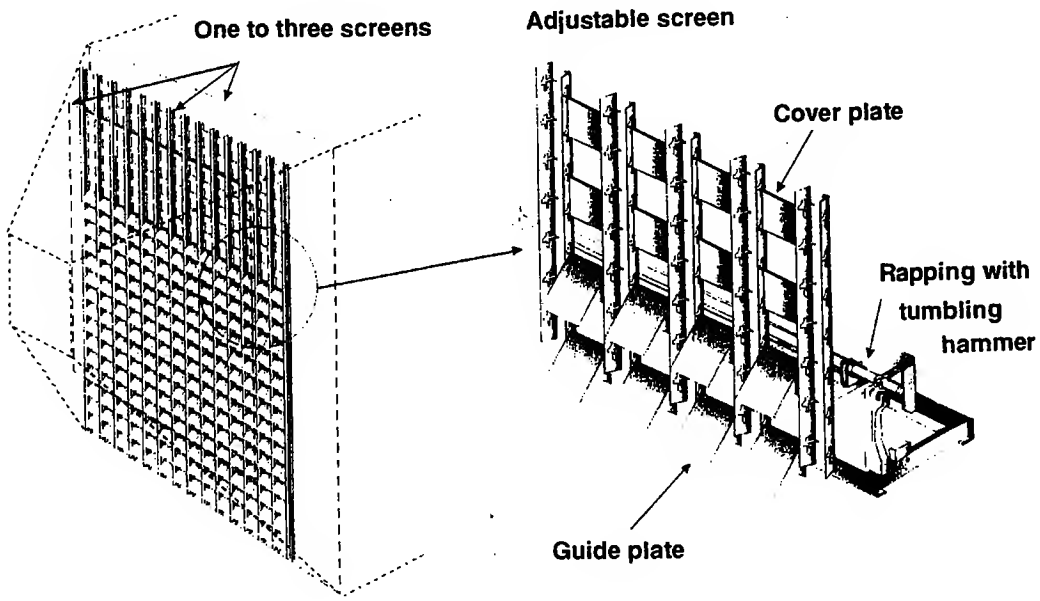
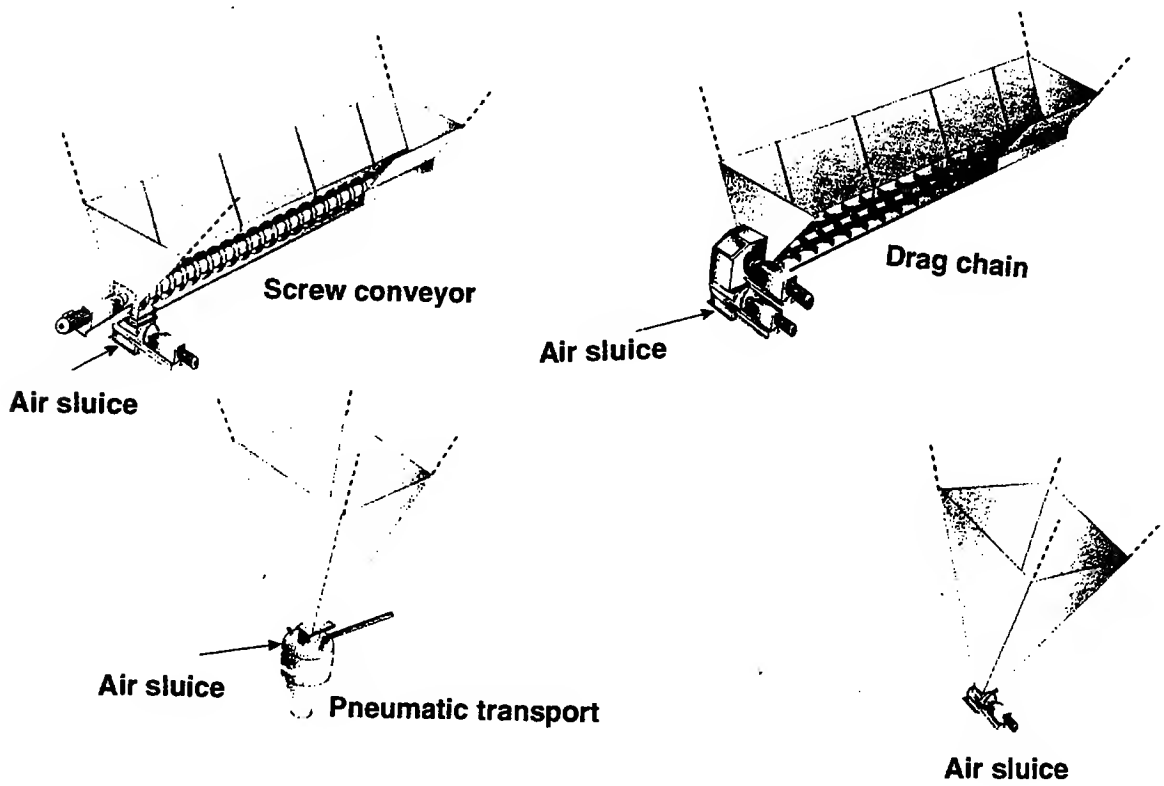


Figure 32 Dust Removal System (FLS)



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Bag Filter (BF)

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The history of bag filters begins in 1886, when engineer Wilhelm Friedrich Ludwig Beth, of Lübeck, was granted the Patent 38396 for a "suction tube filter with automatic cleaning device". Since then, the bag filter technology was continuously improved and is today at a level that makes it superior to any other dedusting system.

1. TECHNOLOGY BASICS OF BF

1.1 Categorizing Bag Filters

To better understand one's bag filter, it is helpful to determine where it fits among the various types of BF. When one tries to group bag filters into a number of categories, it soon becomes obvious that the task is not simple. There appears to be an exception to each of the rules. The creation of certain categories, though they are not rigid, is yet very helpful.

1.1.1 Cleaning Method

One such approach is to group BF designs by cleaning method. There are three major cleaning methods: shakers, reverse gas and pulse jets. In addition to these three dominant cleaning methods, there exist a large number of other cleaning methods which are less often applied. Combinations of the three primary methods have been occasionally employed. For example, reverse air and shake have been used in combination and reverse-air with a pulse assist. Today, in cement plants mainly pure reverse gas or pulse jet BF are applied.

1.1.2 Capacity

Another approach to grouping bag filters is by capacity. Generally, the groupings are small volumes (i.e. below 10'000 m^3/h), medium volumes (i.e. 10'000 to 100'000 m^3/h) and large volumes (> 100'000 to the multimillion m^3/h level).

1.1.3 Filter-Media Type / Temperature Capabilities

The filter-media type and temperature capabilities provide two other ways to categorize and view bag filters. Woven vs. felted media are the media-type categories and high-temperature (> 200°C), medium-temperature (150 to 200°C) and low-temperature applications (< 150°C) are useful temperature groupings. One should be aware of these distinctions and attempt to find where the collector in question fits; thus, when considering operating or troubleshooting recommendations, one only applies recommendations that are suited to the type of collector being used.

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2. FILTER MEDIA

2.1 Selection Criteria

The filter medium is the all-important central feature of any dust collector operating on the filtration principle. With the correct or incorrect choice of the filter material the whole dedusting operation, however well received, will stand or fall in actual practice. Important criteria are:

- filter type, particularly cleaning principle
- gas temperature (average and peaks)
- composition and chemical properties of the gas
- raw gas dust load
- required dust load of the clean gas
- physical and chemical properties of the dust

Furthermore, the filter medium must satisfy the following conditions:

- high air permeability (low pressure losses)
- good mechanical strength
- good thermal stability at operational temperature
- good dimensional stability at operational temperature

2.2 Properties of Various Media

Wool and cotton, the only two raw materials that were available to our grandfathers for making filter media have been nowadays, to a great extent, superseded by synthetic fibers. Even mineral and metal fibers are used for special applications. Table 1 summarizes the main properties of various fibers. The most extensively used ones in the cement industry are polyester, polyacrylnitrile, modificated polyamide (aramide) and glass fibre.

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Table 1: Properties of Various Fibers for Woven Fabrics and Needle Felts

	Fabric, Trademark	Chemical Classification	Tensile strength N/mm2	max. Operating Temperature long time °C	short time °C	Acide Resist.	Alkali Resist.	Abrasion Resist.	Moist Heat Resist.	Price Rating	Density g/m2
Natural Fibers	Cotton	Cellulose	410-670	70-90	120	poor	good	very good	fair	$	150-400
	Wool	Keratin (protein)	120-230	90	120	fair-good	fair	fair-good	fair	$$	400-600
Synthetic Organic Fibers	Dralon T, Acrilan, Dralon, Orlon, Zefran	Polyacryl-} nitrile pure mixed	200-570	120-140 100	150 120	very good	good	good	excellent	$$	400-600
	Polypropylene	Polypropylene (PP)	260-640	100	120	excellent	excellent	excellent	fair	$	550
	Trevira, Dacron, Terylene, Tergal, Vestan, Kodel	Polyester	560-820	130-150	170 (dry)	good	fair-good	very good	poor	$	400-600
	Nylon, Perlon	alipahtic polyamide	370-850	90-110	120	fair	very good	excellent	fair	$	300
	Nomex	aromatic polyamide (aramide)	570-690	200-220	250	good in weak acids	excellent at low temp.	excellent	fair	$$$$	500-600
	Teflon	Polytetra-Fluorethylene (PTFE)	100	260	280	very good	very good	fair	excellent	$$$$$$$	
	Ryton / PPS	Poly-phenylene-Sulfid	1000-1200	180 max. 15%O2	200 max. 15% O2	excellent	excellent	good		$$$$$$	500-800
	P 84	Polyimid	850-900	260	280	excellent	excellent	poor		$$$$$$	550
Synthetic Anorganic Fibers	Glass	Glass	1500-2500	270-230	350	fair-good	fair-good	fair	good	$$$	300-400
	Stone Wool	Mineral	120-260	300-350		fair-good	fair-good				
	Various Steels	Metals	500-750	up to 600		excellent	excellent	excellent			
	Ceramic	Silicium Oxyde		870		excellent	fair	fair	0	$$$$$$$$	> 30

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Special treatment of the fabrics and needle felts can significantly improve the properties of the bags.

Table 2: Special Treatment of the Bag Surface

Non-Fiberglass	**Finish Purpose**	**Available For**
Singe	Recommended for improved cake release	Polyester, Polypropylene, Acrylic, Nomex, Ryton, P 84 (felts)
Glaze	Provides short-term improvements for cake release (may impede airflow)	Polyester, Polypropylene (felts)
Silicone	Aids initial cake development and provides limited water repellany	Polyester (felt and woven)
Flame Retardant	Retards combustibility (not flame-proof)	Polyester, Polypropylene (felt and woven)
Acrylic Coatings (Latex base)	Improves filtration, efficiency and cake release (may impede flow in certain applications)	Polyester and Acrylic felts
PTFE Surface Treatments and Laminates	For capture of fine particulate, improved filtration efficiency, cake release	Nomex, Polyester, Acrylic, Polypropylene (felt) (Laminates available in Polypropylene, Ryton and Polyester only)
PTFE Penetrating Finishes	Improved water and oil repellency; limited cake release	Nomex (felt)
Acid Resistant	Improved acid resistance and water retardance	Nomex (felt)
Fiberglass	**Finish Purpose**	**Applications**
Silicone, Graphite Teflon	Protects glass yarns from abrasion, adds lubricity	For non-acidic conditions, primarily for cement and metal foundry applications
Acid Resistant	Shields glass yarn from acid attach	Coal-fired boilers, carbon black, incinerators, cement, industrial and small municipal boiler applications
Teflon B	Provides enhanced abrasion resistance and limited chemical resistance	Industrial and utility base load boilers under mild pH conditions
Blue Max-CRF/70	Provides improved acid resistance and release properties, superior abrasion resistance, resistant to alkaline attack, improved fiber encapsulation	Coal-fired boilers (high and low sulfur) for peak load utilities, fluidized bed boilers, carbon black, incinerators

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2.3 Hydrolytic Influences

One of the greatest enemies of the textile filter media is hydrolysis. By this is understood the breakdown of the molecular chain of the polymerizate by the action of moisture. Hydrolysis is intensified by the action of heat, acids and alkalis.

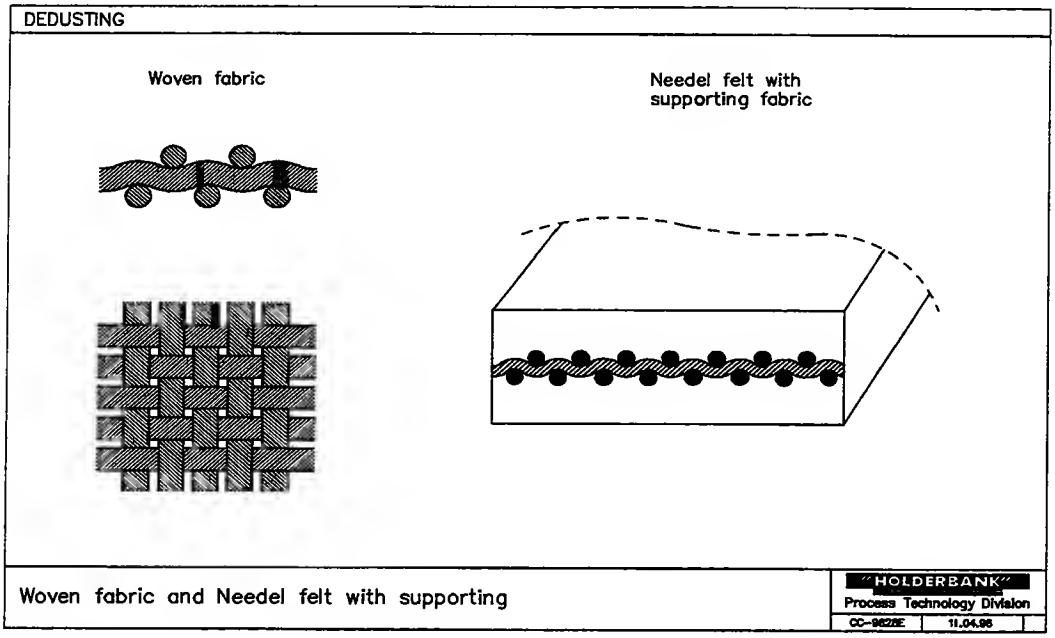
Polyester, for example, should not be used under conditions in which moisture and elevated temperatures occur in combination. Also aromatic polyamides (Nomex) are subject to hydrolytic influences at temperatures above 100°C, especially if acids or alkaline agents additionally act as a catalyser.

Fortunately, in recent years chemical modification processes have emerged which have enabled these polyamides and also polyester to be substantially improved in this respect.

2.4 Woven Fabrics and Needle Felts

Filter media are available either in the form of woven fabric (cloth) or felt fabrics (more particularly needle felt). The structure of these two types of filter media is shown as follows:

Figure 1 Woven fabric and needle felt with supporting structure



Characteristic of woven fabric is its system of warp and weft threads crossing one another. Essentially the fabric pores, i.e. the holes between the warf and the weft threads, are decisive with regard to the filter properties. Effective fabrics for dust collection purposes have a free perforated area of about 40%. Therefore, they must have a permanent dust crust to maintain their good filtering effect.

In contrast with woven fabrics, needle felts are "three-dimensional" filter media. Their active filtering surface is located both on the surface and in the interior of the medium. The dust collection process, beside the sieving effect as in woven fabrics, additionally takes place through inertia and barrier effects. For reinforcement, needle felt can be provided internally with a supporting woven fabric interlayer which is only of secondary importance as regards its dust- collecting effect but which serves primarily to give tear resistance and dimensional stability to the material. The pore volume of needle felts is 60 - 90%. This porous structure allows higher admission velocities with lower pressure drops and higher dust collection efficiencies.

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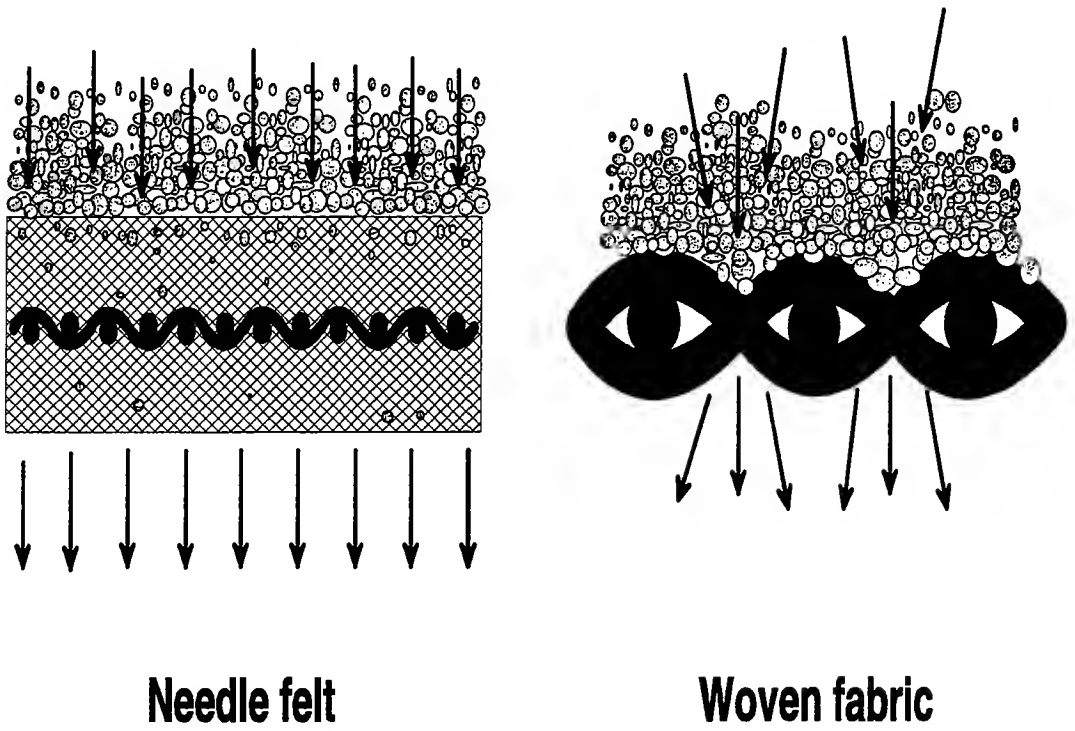
The filtration process for both types of filter media is shown in Fig. 2:

Figure 2 Dust Separation with Needle Felt and Woven Fabric



For the cleaning of the woven fabric bags reverse gas is usually used. The needle felt bag filters generally have a jet pulse system for cleaning.

The high cleaning air pressure of the jet pulse allows the use of denser filter media which in turn achieve higher dust collection efficiencies.

Special finishing or application of membranes on the bag surface become more and more important, especially for the jet pulse filters (see table 2). The purpose of those treatments is to give the bags improved resistivity against chemical and mechanical attack as well as optimum filtration efficiency and cake release (especially for fine dust particulates). Bags with such a treatment may have a very much improved filtering efficiency and therefore do not need any more cake formation on their surface to achieve a good dedusting efficiency. Such bags can be operated with a much thinner dust crust and have therefore a reduced flow resistivity.

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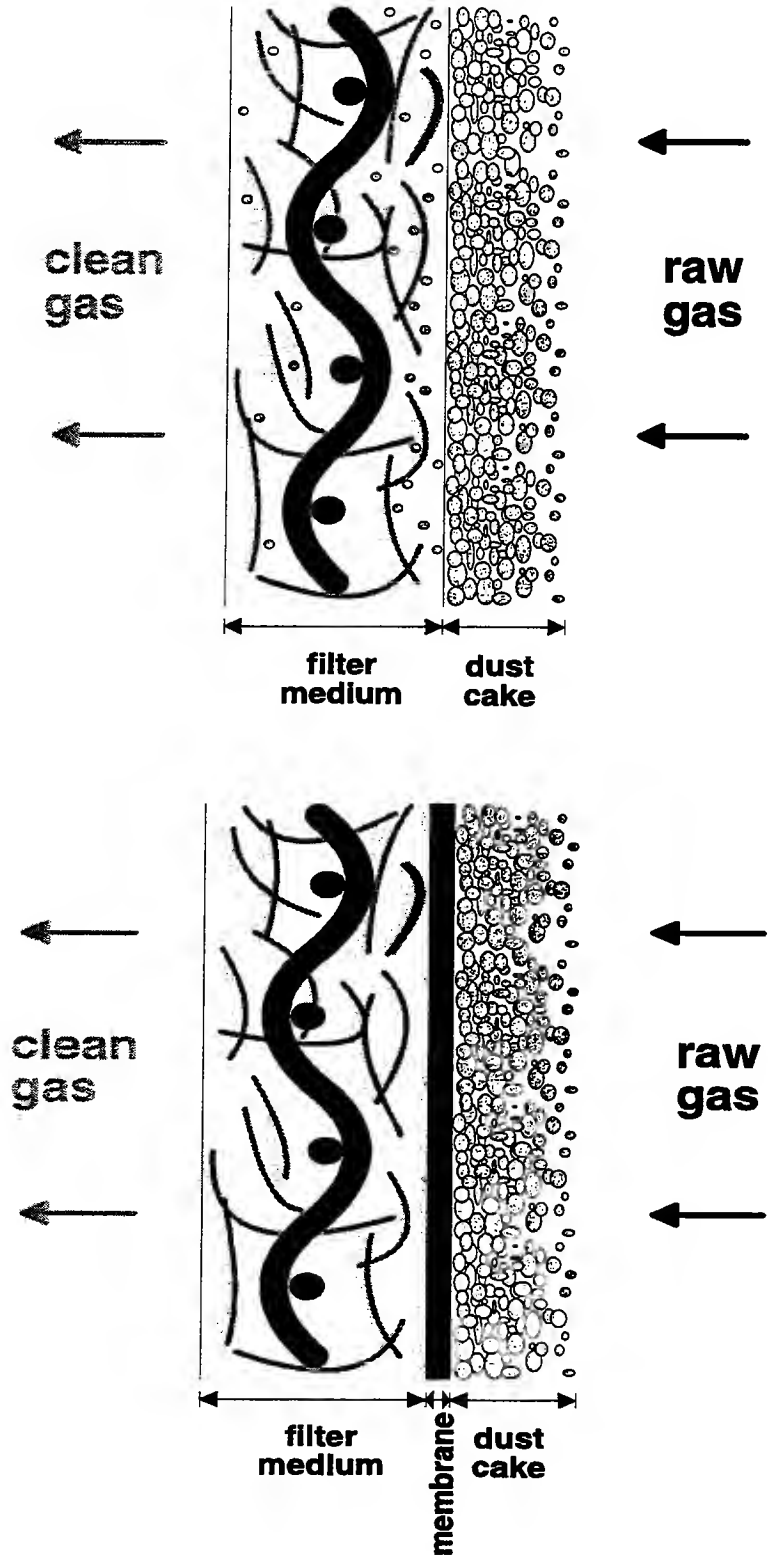
Figure 3 **Needle felts without and with membrane**



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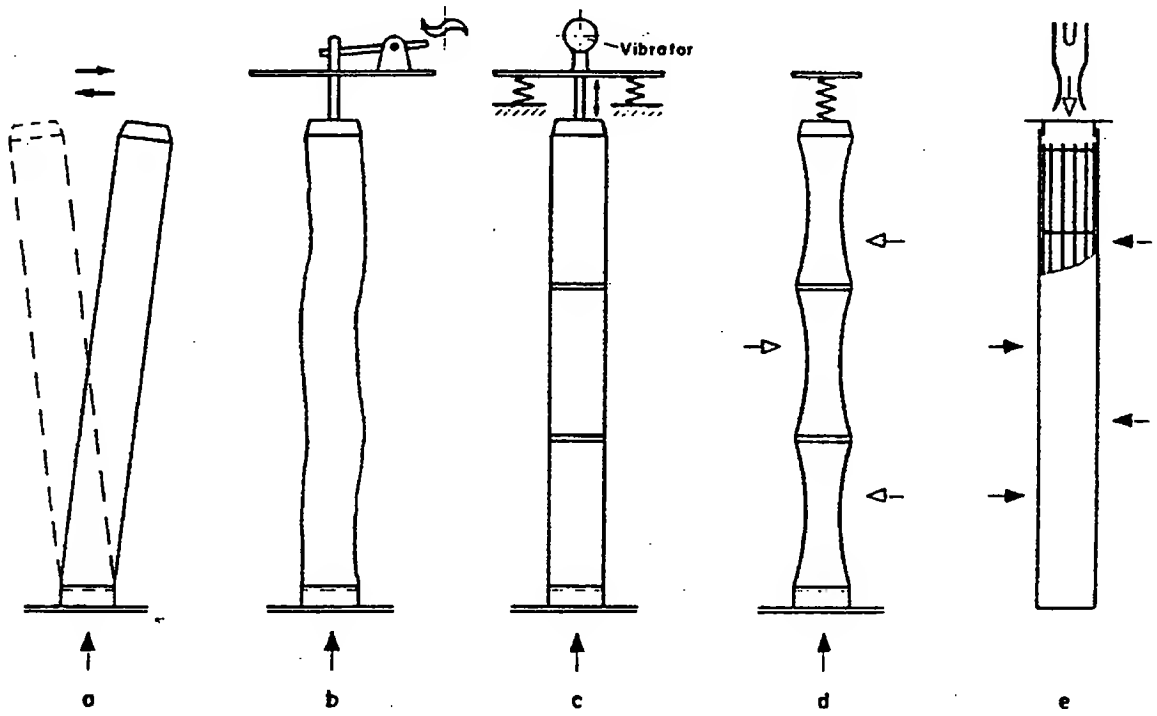
3. CLEANING SYSTEMS

3.1 Overall View

Ten or twenty years ago, fabric filters used to be cleaned by rapping or shaking, sometimes in combination with low-pressure air purging but in the early 1960's a new agent for filter media cleaning was introduced: compressed air.

Fig. 4 gives a summary of the various cleaning principles used for bag filters.

Figure. 4 Cleaning Principles of Bag Filters



a, b: manual or mechanical, by rapping or shaking
c: mechanical, by vibrating
d: pneumatic, by reverse air flow (often combined with shaking or vibrating)
e: pneumatic, by compressed air (pulse jet)

Since most of the mechanically cleaned filters have been superseded by compressed air ones, mainly in the cement industry, the mechanical cleaning devices will not be described further here.

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3.2 Reverse Gas Cleaning

The reverse air cleaned bag filters usually contain woven filter bags. The raw gas enters the bags from the bottom. It flows from the bag center to the outside of the bag. The dust is deposited on the inner surface of the bag. Removal efficiency is improved and maintained by these particulate deposits (residual dust cake). With time excess particulates are deposited on the bags and increase the system resistance to the gas flow (pressure loss).

To allow the filter fan to operate within the design parameters and to reduce the fan power consumption, this dust cake must be partly removed. Bag cleaning methods must be designed properly - not over cleaning or under cleaning.

The bag cleaning process is triggered either by a timer or, better, when the pressure drop over the bag filter reaches some predetermined level. A reverse air bag filter consists of several compartments, usually of more than ten. When the bag cleaning process is started, the outlet valves of one of the compartments are closed (off-line cleaning). Then, an auxiliary fan forces a relatively gentle flow of filtered gas backwards through the compartment and bags to be cleaned. This causes the bags to partially collapse inward, dislodging the dustcake. This falls through the bags, the thimble and the tubesheet into the hopper. Metal anticollapse rings sewn into the bags along their length prevent complete bag collapse.

Fig. 5 is a schematic of compartments in a reverse-gas-cleaned BF showing operation during filtering, cleaning and purging prior to shut sown or maintenance. To support the cleaning of the bags low frequency, pneumatic horns ca be installed and used simultaneously with the normal reverse gas flow.

Table 3: Typical data of reverse gas filter

Reverse gas pressure	30 - 40 mbar
Reverse gas flow	2.0 - 3.5 m^3/m^2h
Power consumption (installed capacity)	~ 0.0075 kW/m^2

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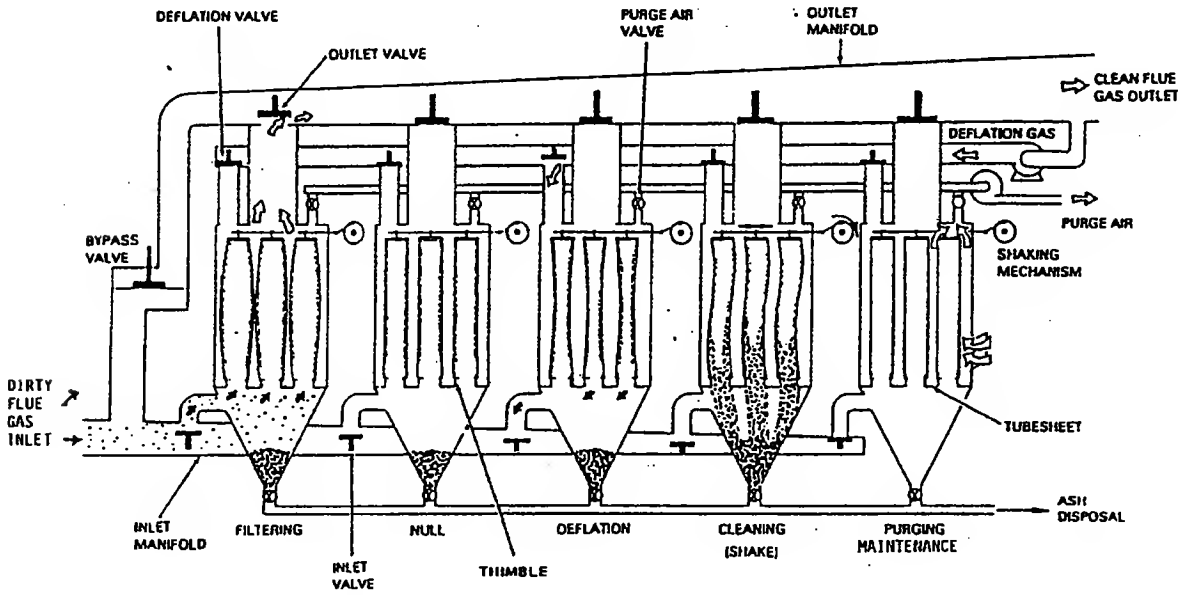
Figure 5: Schematic of a shake / deflate-cleaned baghouse filtering flue gas, isolated prior to bag cleaning (null), with deflation gas entering the compartment prior to bag shaking, bag shaking and purging (or ventilation) prior to maintenance



The main criteria that are defining the size of a reverse gas by filter are:

- maximum actual flow
- maximum permissible air to cloth ratio (A/C)
- number of compartments

Table 4: Recommended A/C ($\frac{m3}{m2h}$) for reverse gas filter (net, net)[1)]

Operating mode	Compound	Direct
Kiln exhaust gas	30	36

1) A) Because one compartment is usually isolated for reverse gas cleaning only the filtering area of n-1 compartments are used to calculate the A/C (net).

B) The $\frac{m3}{h}$ of the reverse gas must be added to the filter inlet flow to calculate the total gas flow passing the bags and the A/C (net).

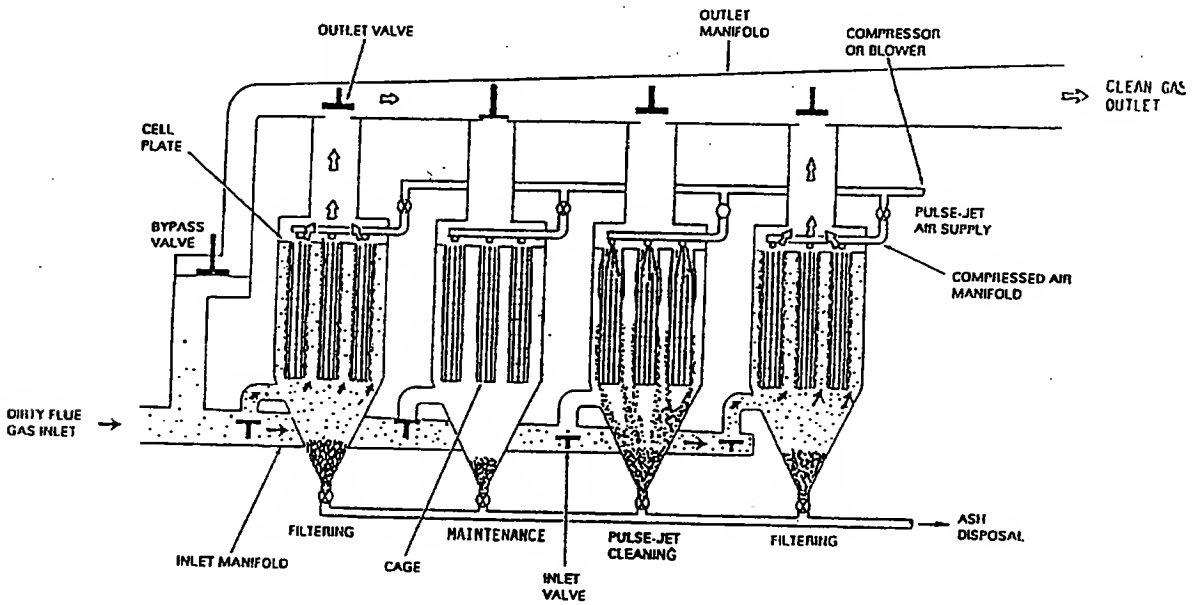
C) A) + B) → A/C (net, net)

3.3 Pulse Jet Cleaning

Pulse jet cleaned bag filters normally employ felted fabrics of various types. The raw gas enters the bags from the outside. The cleaned gas flows through the center of the bag to the clean gas plenum and from there to the stack. The dust is deposited on the outer surface of the bag. To prevent bag collapse during filtering, metal cages are inserted inside each bag. Just like the reverse-gas cleaned bag filter, periodic bag cleaning is required to remove excess residual dust cake. This is accomplished by pulsing compressed air down into each filter bag. Bag cleaning can be accomplished either with the compartment isolated or not isolated (on-line or off-line cleaning, see Fig. 6).

The cleaning made is selected based on the particulates being filtered, process conditions and bag quality.

Figure 6 Schematic of compartments in a pulse-jet cleaned baghouse filtering flue gas, purging (or ventilating) prior to maintenance, cleaning bags and filtering flue gas again



Most "jet" filters use injectors for the periodic purging of the individual filter elements with a nozzle, usually disposed centrally.

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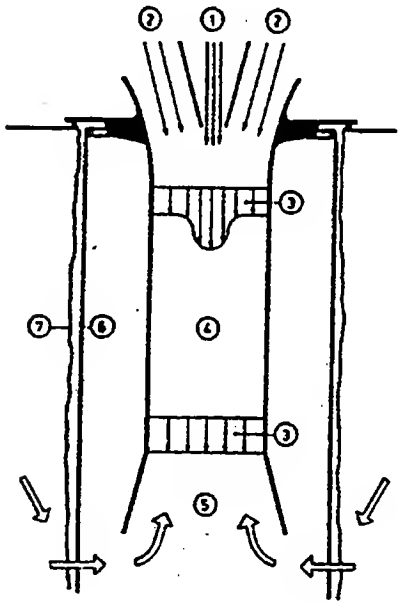
Figure 7 Injector with central nozzle



Each row of bags or each individual bag is equipped with an injector which operates as follows:

When the bag is in service, i.e. engaged in dust collecting, the clean gas flows from the interior of the bag through the injector into the clean gas plenum.

When compressed air is released as a jet from the injector at a velocity which may be above or below the velocity of sound (depending on its type and design), secondary air is entrained from the clean gas chamber of the filter, and a purging air flow comprising the actual jet plus this entrained secondary air is introduced into the filter bag. The ratio of secondary flow to jet flow is called mass flow ratio.

Cleaning the bags involves three stages:

1) The normal filtration gas flow is briefly interrupted by the barrier effect of the purging air flow in the opposite direction.
2) The purging air injected into the bag expands it to its original circular section (Fig. 7) and removes the dust cake which falls down into the dust bunker.
3) The purging air then flows outward through the filter medium in the direction opposite to that of the raw gas flow.

A compressed air pulse of only 0.1 - 0.2 s duration is sufficient to perform all three above mentioned stages. The pulse is applied at intervals of 1 to 10 minutes. Thus, the duration of the cleaning operation amounts to only 0.02 - 0.3% of the overall operating time of the filter. Practically speaking, therefore, the whole filter surface area is always available for filtration, and the net area is virtually equal to the gross area.

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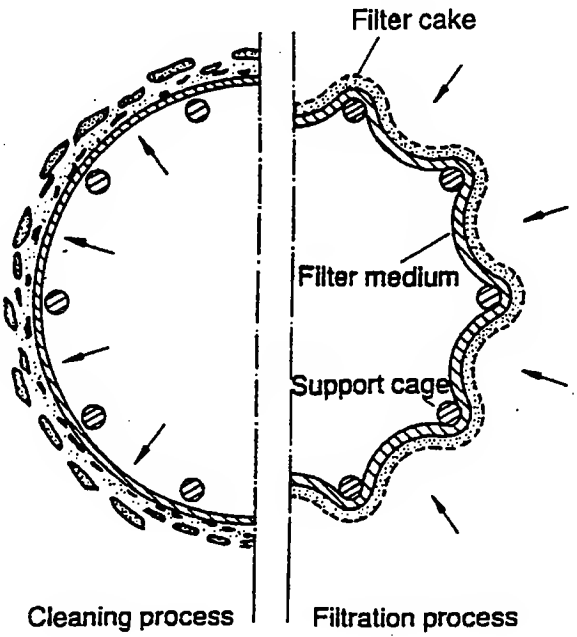
Figure 8 Jet pulse bag charged from the outside



3.3.1 Guide Values for Jet Pulse Filters

Although there exist many different forms of construction of jet pulse filters it is possible to give some guide values.

For common jet pulse filters, with vertical bags and nozzle injectors, the following data can be given:

Table 5: Typical data of jet pulse filters

Compressed air pressure	bar	4 - 7
mass flow ratio	-	3 - 6
Compressed air flow rate	Nm^3/m^3h	0.05 - 0.10
spec. power consumption[1)]	kW/m^2	0.005 - 0.010

1) installed capacity

The main criteria that are defining the size of a jet pulse bag filter are:

- maximum actual gas flow
- maximum permissible air to cloth ratio (A/C)

Table 6: Recommended A/C $(\frac{m3}{m2h})$ for jet pulse filters

Kiln Exhaust Gas	Clinker Cooler Vent Air	Cement Mill	
		with Cyclone	without Cyclone
55 – 65	65 - 80	60 - 70	90.00

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3.3.2 Bag Dimensions

Jet pulse bag filter bags are smaller than bags from reverse gas bag filters because of the different cleaning principles. The diameters of the bags are usually between 130 and 150 mm. The length of the bag should be between 4 and 3 meters, if a conventional gas flow pattern is applied, which means, that the gas enters through the hopper and then flows upwards, along the bags. In newer applications, the gas is distributed partially through the hopper and partially horizontally to the bags. Together with new low pressure cleaning, here larger bags of up to 6 m are applicable.

Generally longer bags are more difficult to clean and have therefore an increased compressed air consumption. It is also very difficult to remove a bag of more than 3 to 4 meters length if it is filled with dust because of a hole in the felt.

3.3.3 Bag Material

As mentioned above, most bags for jet pulse filters are made with needle felt. This material allows a higher air to cloth ratio and resist the stress of the jet cleaning. For some exceptions (mainly bypass bag filter) and other high temperature applications, glass fiber bags are used which resist the high temperature and the aggressive environment.

Experience during the last few years showed that jet pulse bag filter (operated at lower temperatures, < 130°C) are more reliable and economical than others. At temperatures below 130°C low cost bags like polyester or polyacrylnitril bags can be applied and the power consumption of the filter fan is reduced as well.

The still on-going fast development of fabrics significantly improved the performance of bag filters. Due to this development it is possible to install very efficient bag filters at cost that are in the same range or below the cost of EPs.

Special treatment of the bag fabric surface allows for more frequent cleaning without increased dust emission. The smaller dust cake on the bags is reducing the pressure drop and therefore saves fan power.

3.3.4 Maintenance of Jet Pulse BF (for Kiln and Clinker Cooler)

Except for the valves and dust discharge equipment jet pulse BF have no moving parts. The most important aspect for maintenance are the bags. Holes in the bags can significantly increase the dust emission. Defective bags in reverse gas BF are replaced on line by isolating the respective compartment. Modern jet pulse BF do not need compartments like the reverse gas BF but usually consist of about 4 to 6 modules (static stability) with manual outlet and inlet valves for each module.

It is proposed to only visually inspect the modules for defective bags through the inspection window or from the open door (values closed!). If a defective bag is found it does not need to be replaced immediately. It is sufficient to isolate the respective row of bags (~ 10) from the jet cleaning. The incoming dust will close the hole after a short while. The bag can be replaced during the next kiln shutdown.

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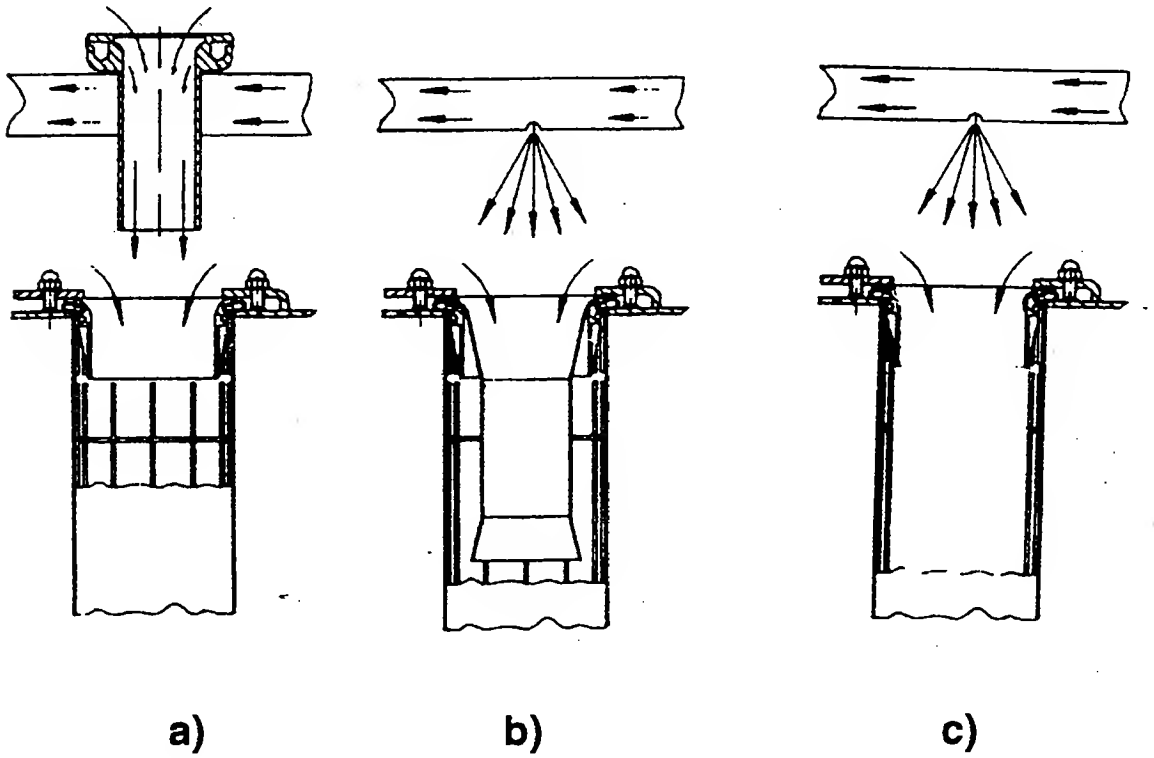
Figure 9



Injector Systems for Jet Cleaning
a) "Coanda" Ring Gap Injector
b) Central Nozzle with Injector
c) Central Nozzle without Injector

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Figure 10 **Jet Pulse Bag Filter designed for Off-line Cleaning with High Clean Gas Plenum (NEU Process)**

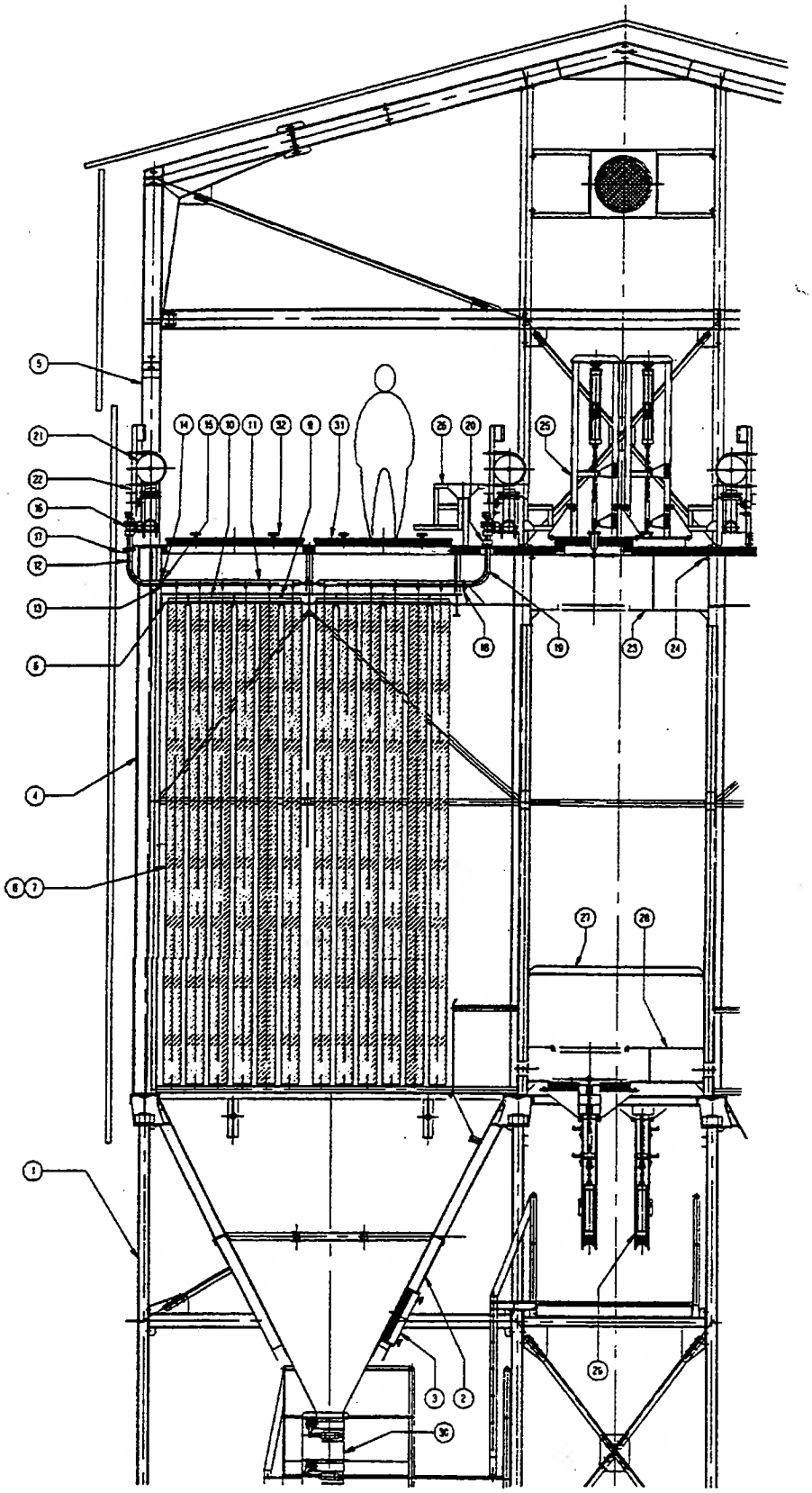

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Figure 11 **Jet Pulse Bag Filter with High Clean Gas Plenum for 5'000 t/d Kiln. Dedusting of Kiln Exhaust Gas (Redecam)**

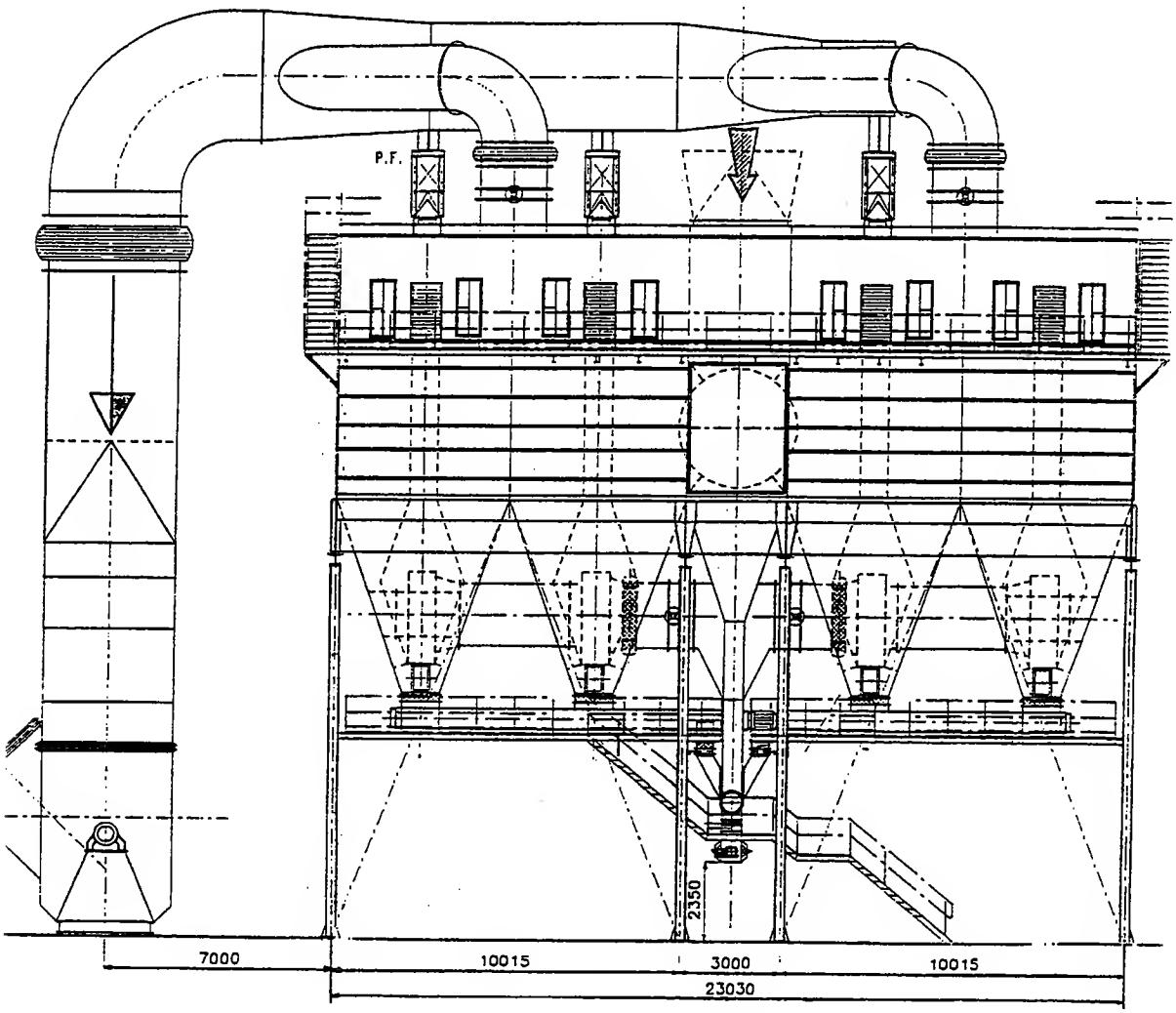

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Figure 12 **Jet Pulse Bag Filter with Low Clean Gas Plenum for 5'000 t/d Kiln. Dedusting of Kiln Exhaust Gas (Redecam)**

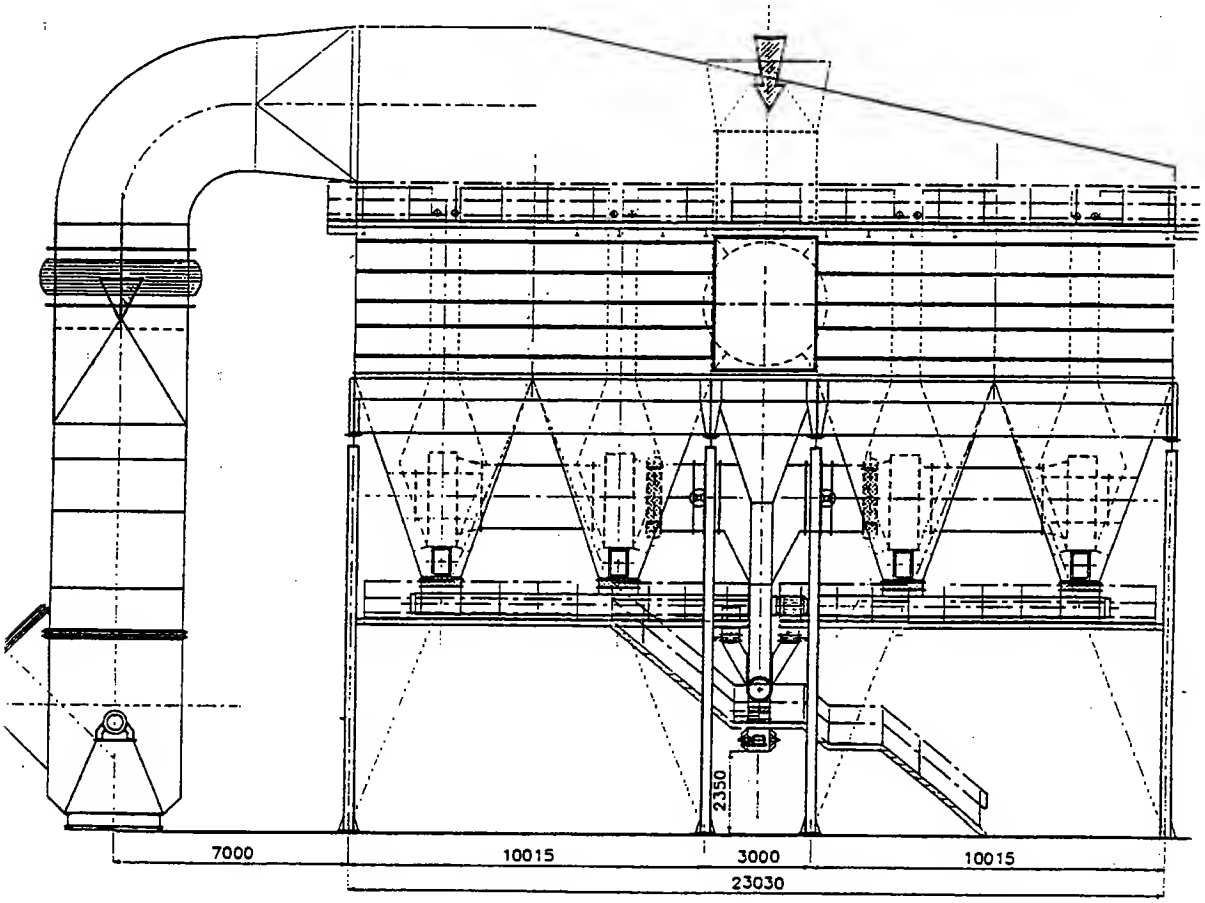

Figure 13 **Air to Air Heat Exchanger and Jet Pulse Bag Filter with High Clean Gas Plenum.**
Dedusting of Clinker Cooler Vent Air (Redecam)

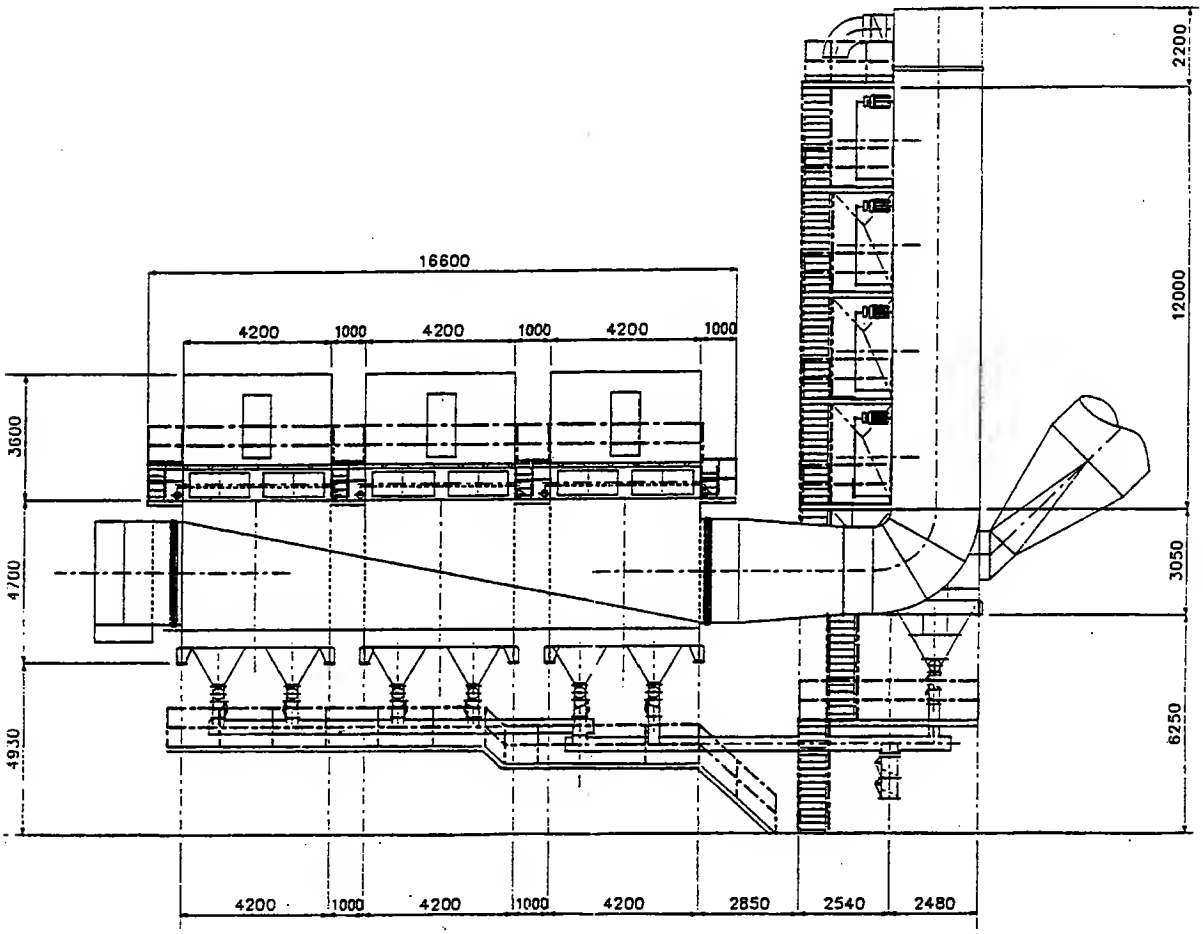

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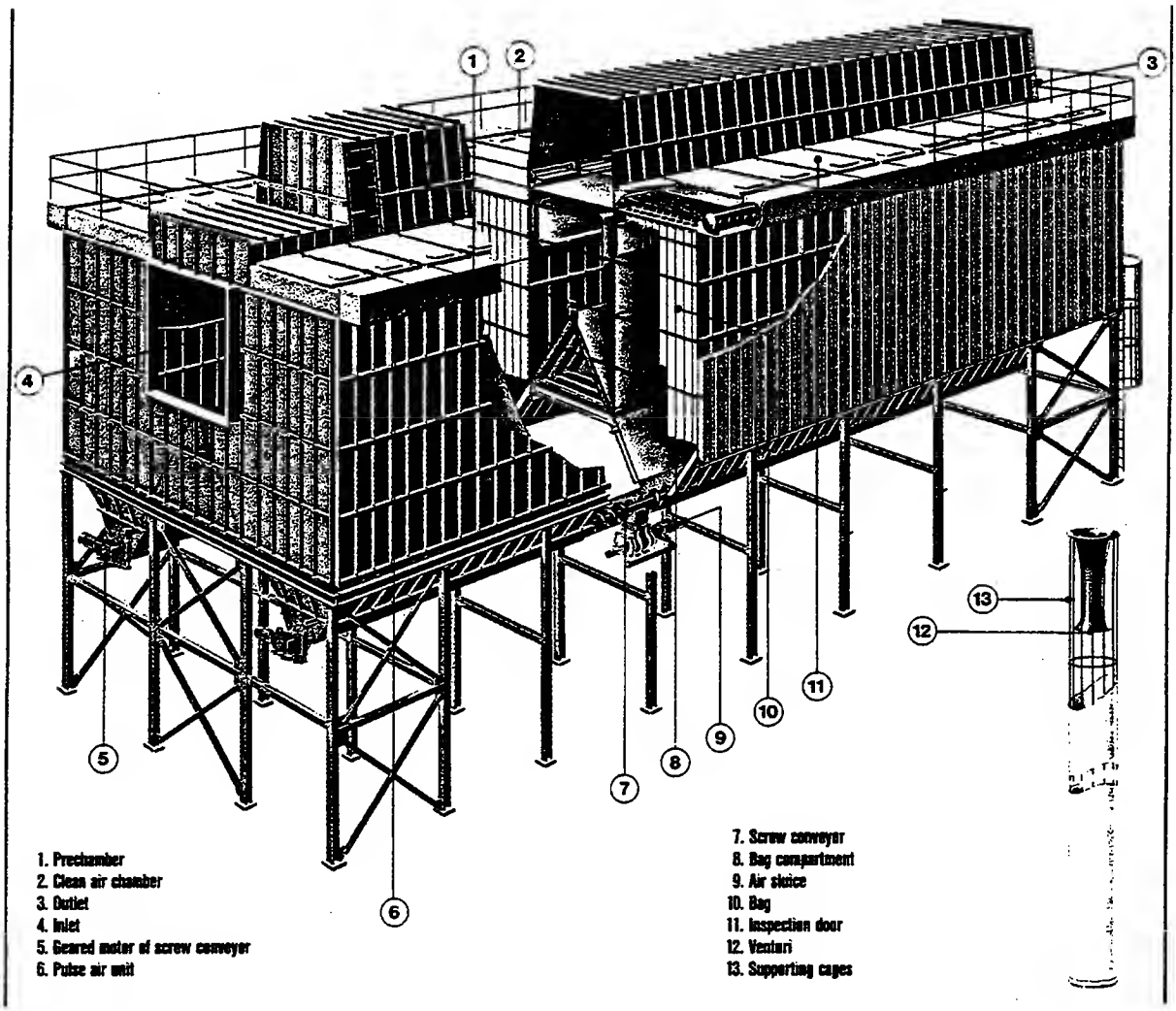
Figure 14 3-D View on Jet Pulse Bag Filter with Low Clean Gas Plenum



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Dedusting Concepts for Cement Kilns

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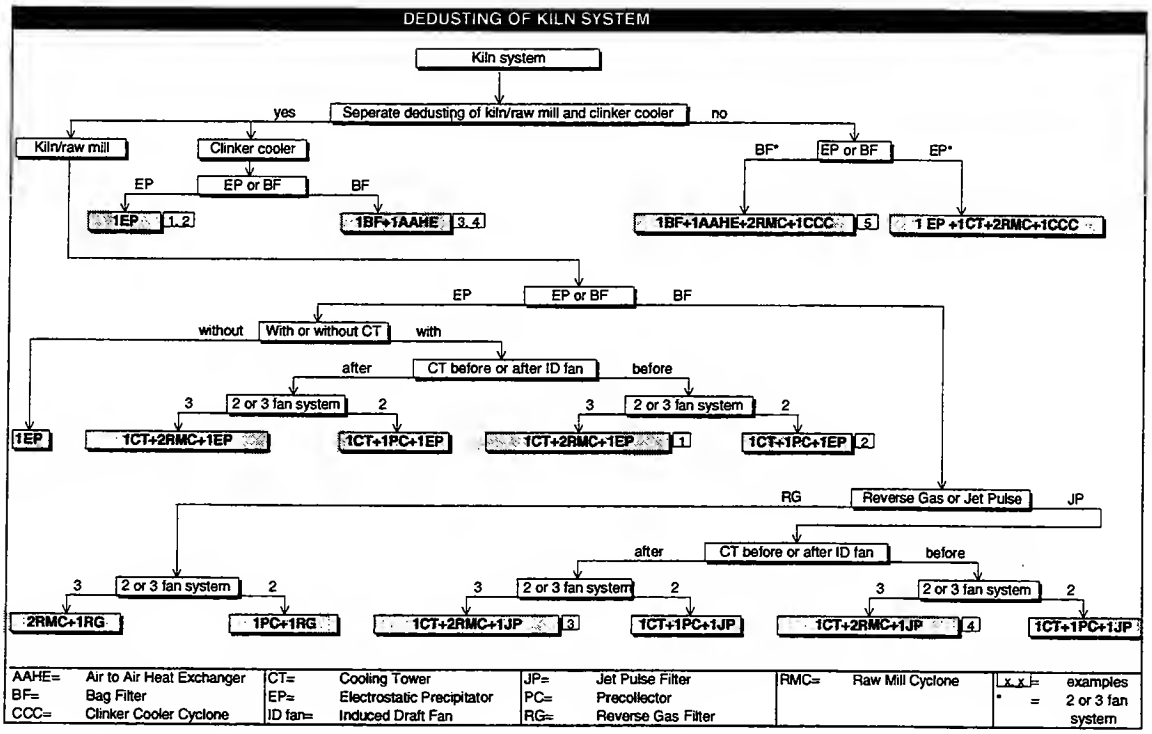
Because of the changing needs and the different filter and cement production technologies, the concepts for cement kiln dedusting are numerous. Each of them has certain advantages and disadvantages.

The dedusting of preheater kilns with waste heat utilization (raw mill) and great cooler is the most difficult case. Therefore, the following examples will be based on such a modern cement kiln system.

Chart 1 shows the different possible dedusting concepts. Within the presented concepts there is still some differentiation possible; e.g. the bag filter concepts can be further differentiated by the fabric quality or the type of pulse jet that is applied.

Figure 1 Different dedusting concepts



It can be seen that the first decision which must be made is, if the kiln and clinker cooler shall be dedusted separately or in one common filter. The standard solution is to use two filters, one for the kiln and one for the cooler, but we will see later that the simultaneous dedusting in one filter has some important advantages.

The next decision is whether bag filters or electrostatic precipitators shall be used.

Because of the more and more stringent dust emission limits and the superior dedusting efficiency of bag filters the present trend is to apply bag filters.

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1. COMPARISON BETWEEN ELECTROSTATIC PRECIPITATORS AND BAG FILTERS

The basic working principle of the two filters are completely different. The EP efficiency depends on many variables like gas volume collecting area, dust characteristics, dew point etc. Once the electrodes are not energized, the dust emission is increasing rapidly. The bag filter efficiency mainly depends on the quality of the bags and the sealing between clean gas plenum and raw gas plenum and is not affected by the process.

The EP used to be a very popular filter in Europe and other places because of the low pressure drop, the little maintenance required and the high efficiency under normal operating conditions. Today, the main problem of the EP is its efficiency under not normal operating conditions. Changing the operating mode from compound operation to direct operation and start-up and shut down of the kiln always causes increased dust emission (reduced EP efficiency) because of unfavorable temperatures and dew point during the transition phase. Disturbances in the combustion process or excessive concentration or organics in the raw material can produce CO peaks in the exhaust gas which force the operators to shut the kiln EP down to prevent CO explosion. During kiln upset conditions the vent air (clinker cooler) temperature, volume and dust load are increased. This usually leads to elevated dust emission from the clinker cooler.

Some decades ago, increased dust emission during such incidences where generally accepted by the neighbors and the authorities. Nowadays, with the stronger environmental awareness of the people and the more stringent emission limits (20 mg / Nm^3, continuous dust emission control) it can be difficult to achieve the required long term efficiency with an EP.

An important advantage of the EP can be the classification of the dust particulates. The coarse particulates are collected in the first hoppers, the fine ones in the last hoppers. Condensibles like alkalies and metals are enriched in the fine fraction of the dust. With an EP less dust has to be extracted from the filter to reduce the concentration of the condensibles in the system.

Bag filters, if well maintained, have a very high efficiency unaffected by the process conditions. Their main disadvantage is the high pressure loss and the additional maintenance cost for the regular replacement of the bags.

The new generation of jet-pulse filters have significantly reduced the pressure drop over the filter and the maintenance cost compared to the reverse gas filters.

Table 1: Comparison of jet pulse filter with reverse gas filter for 4-stage preheater:

	Kiln Bag Filters	
	Jet Pulse	Reverse Gas
Bag quality	Polyacrylnitrile	Fiber glass
Relative cost for one set of bags	~ 1	~ 3 - 4
Pressure drop [mbar]	~ 8 – 12	~ 10 - 20
Bag cleaning 2) [Wh/kg cli]	~ 0.6 - 0.8	~ 1.1 - 1.5
CT water pump [Wh/kg cli]	~ 0.6 - 0.8 1)	---
Filter fan [kWh/kg cli] 2)	~ 6.7 - 7.5	~ 6.5 - 9.0

1) Only during direct operation 2) Installed capacity

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A disadvantage of the bag filters is that they produce a not neglectible amount of waste (used bags). Depending on the local regulations for waste elimination and the quality of bags their elimination can be expensive.

Generally, the investment cost for jet pulse filters and electrostatic precipitators are about the same. For installations above 3'000 t/d and 50 mg/Nm^3 it is slightly in favor of the EP, below 3'000 t/dry in favor of the bag filter. This is only a general rule and because of the very strong competition this may be significantly different for individual projects.

For very low dust emission like 10 mg/Nm^3 EPs are higher in price than BFs because the required collecting area of the EP is growing exponentially with the reduction of the clean gas dust content:

Table 2: Additional equipment cost for reduction of the clean gas dust content from 50 to 10 mg/Nm^3:

	50 mg / Nm^3 → 10 mg / Nm^3
EP	~ 20%
BF	~ 5%

The comparison of the operating cost is very difficult because of the variable cost of the bags and the electrical power. As a general rule it can be said that the operating cost of EPs are still slightly lower compared to BFs. For very low dust emissions (e.g. 10 mg/Nm^3) the difference between EP and BF becomes neglectible because the power consumption of the EP is increasing exponentially with the reduction of the clean gas dust content.

Conclusion of Comparison of EP and BF:

For low clean gas dust contents the investment and operating cost of EP and BF (jet pulse) are about the same. The dedusting efficiency of bag filters are superior to EP because bag filters are less sensitive to process changes.

Among the bag filters there are two major technologies available:

- the reverse gas filters
- the jet pulse filters

Recent experience shows that the dedusting efficiency of jet pulse filters is the same as or better than the efficiency of reverse gas filters. Investment and operating cost of jet pulse filters are significantly lower compared to reverse gas filters. Below you will find six examples for cement kiln dedusting concepts with short comments on their advantages and disadvantages.

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2. CONCEPT 1

Separate dedusting of kiln exhaust gas and clinker cooler vent air.

2.1 Kiln Exhaust Gas

2.1.1 Concept

- Conditioning tower before ID fan to cool the exhaust gas to below 150°C during direct operation
- 3 fan system (separate raw mill fan and cyclones)
- Electrostatic precipitator without pre-separation chamber

2.1.2 Advantages

- Length of duct between preheater and conditioning tower very short because the conditioning tower is placed ahead of the ID fan. Therefore, the investment costs are reduced.
- Improved raw mill control because of separate mill fan
- Possibly reduced operating cost compared to bag filter
- Less false air intake and danger of corrosion at the electrostatic precipitator because of small negative static pressure (due to 3 fan system)
- No precollection required at the filter (due to raw mill cyclones)

2.1.3 Disadvantages

- Strong load changes on the ID fan because of different gas temperatures during direct and compound operation (due to conditioning tower before ID fan)
- More false air intake into conditioning tower and higher corrosion risk because of strong negative static pressure (60 - 40 mbar) compared to conditioning tower positioned after ID fan
- Slightly higher pressure drop between ID fan and filter fan because of the raw mill cyclones
- Over all dedusting efficiency of electrostatic precipitator is lower compared to bag filter

2.2 Clinker Cooler

2.2.1 Concept

- Electrostatic precipitator

2.2.2 Advantage

- Possibly reduced operating and investment cost

2.2.3 Disadvantage

- Efficiency of electrostatic precipitator is lower compared to bag filter

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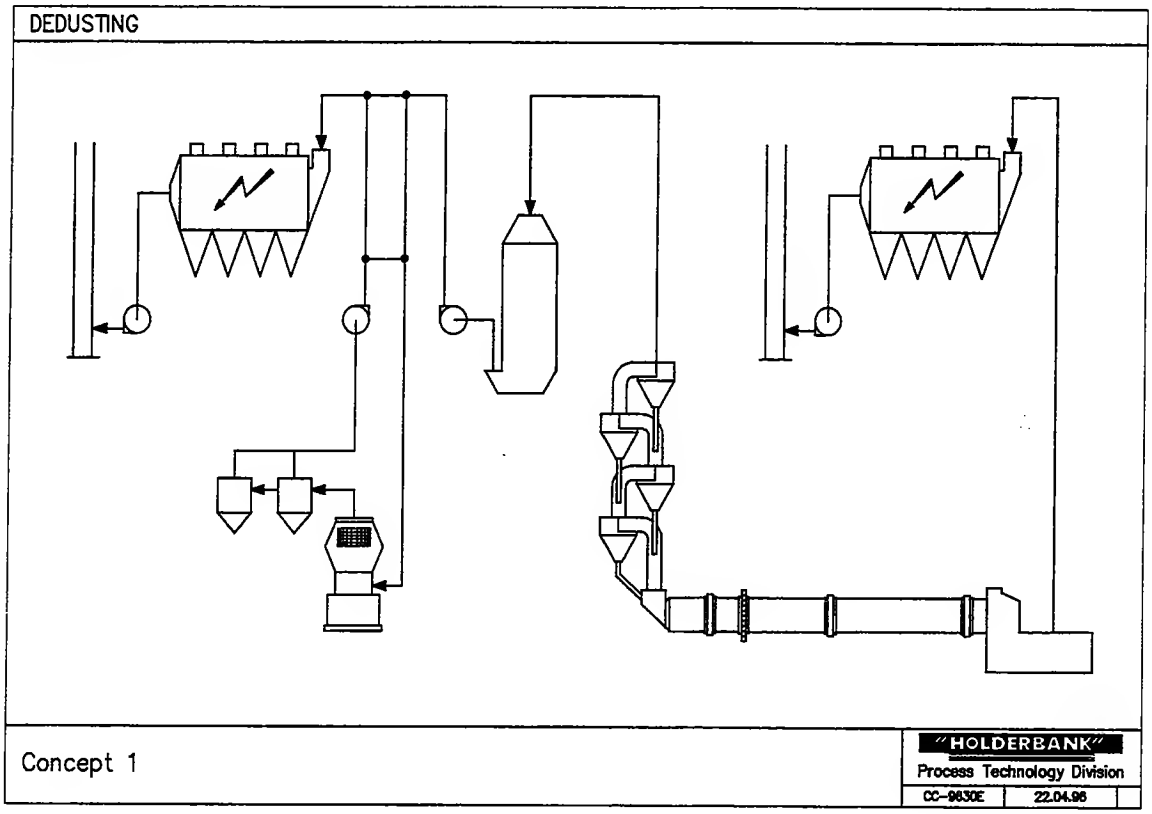
Figure 2 Concept 1



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3. CONCEPT 2

Separate dedusting of kiln exhaust gas and clinker cooler vent air.

3.1 Kiln Exhaust Gas

3.1.1 Concept

- Conditioning tower before ID fan to cool the exhaust gas to below 150°C during direct operation
- 2 fan system (no separate raw mill fan and cyclones)
- Electrostatic precipitator

3.1.2 Advantages

- Length of duct between preheater and conditioning tower very short because the conditioning tower is placed ahead of the ID fan. Therefore, the investment costs are reduced.
- Slightly reduced pressure drop between ID fan and filter fan because of missing raw mill cyclones (compare Concept 1)
- Possibly reduced operating and investment cost compared to bag filters

3.1.3 Disadvantages

- Strong load changes on the ID fan because of different gas temperatures during direct and compound operation.
- Increased false air intake and corrosion at the filter because of very strong negative static pressure
- Electrostatic precipitator need separate precollector chamber
- Efficiency of electrostatic precipitators is lower compared to bag filter
- Gas recirculation to operate raw mill is increasing the required filter size (due to 2 fan system)

3.2 Clinker Cooler

- See concept 1

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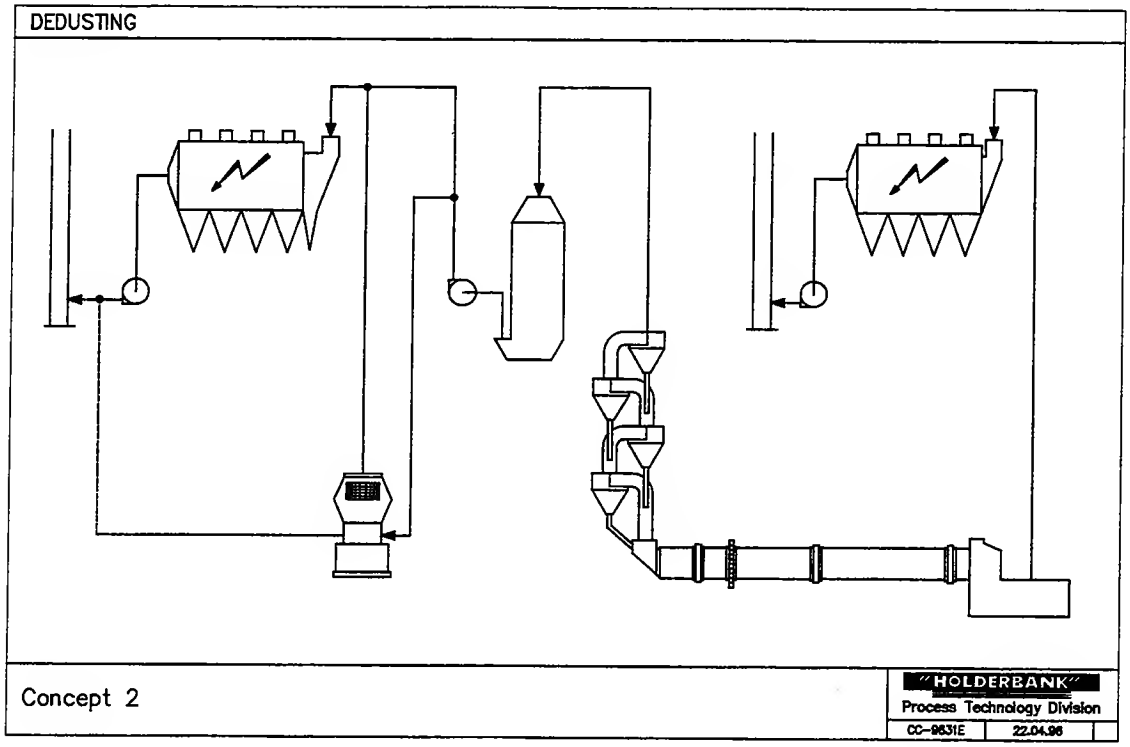
Figure 3 Concept 2



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4. CONCEPT 3

Separate dedusting of kiln exhaust gas and clinker cooler vent air.

4.1 Kiln Exhaust Gas

4.1.1 Concept

- Fresh air intake before and / or after the ID fan to cool the exhaust gas to below 240°C
- 3 fan system (separate raw mill fan and cyclones)
- Reverse gas bag filter

4.1.2 Advantages

- No cooling tower and water injection required
- Higher overall dedusting efficiency than electrostatic precipitator
- Simple filter inlet temperature control
- Reduced corrosion in the filter because of high operating temperature and low dew point
- No precollection required at the filter (due to raw mill cyclones)

4.1.3 Disadvantage

- Higher operating cost than electrostatic precipitator and jet pulse bag filter
- Very large filter
- If the temperature control fails it is possible to burn the bags
- With a bad design it is possible that the pressure drop over the filter is increasing to a point where the capacity of the filter fan is not sufficient anymore to pull the gases

4.2 Clinker Cooler

4.2.1 Concept

- Cooling of the vent air with air to air heat exchanger (designed for up set conditions)
- Jet pulse bag filter; preferably equipped with polyester bags

4.2.2 Advantage

- The air to air heat exchanger serves as a compensator in case of upset conditions and allows therefore a very smooth operation of the filter.
- Higher overall dedusting efficiency compared to EP

4.2.3 Disadvantage

- If air to air heat exchanger and the jet pulse are designed properly, there are no real disadvantages in this concept. With a bad design it is possible to damage the bags during heat excursions or to increase the pressure drop over the filter up to the point where the capacity of the filter fan is not anymore sufficient to pull the gases.

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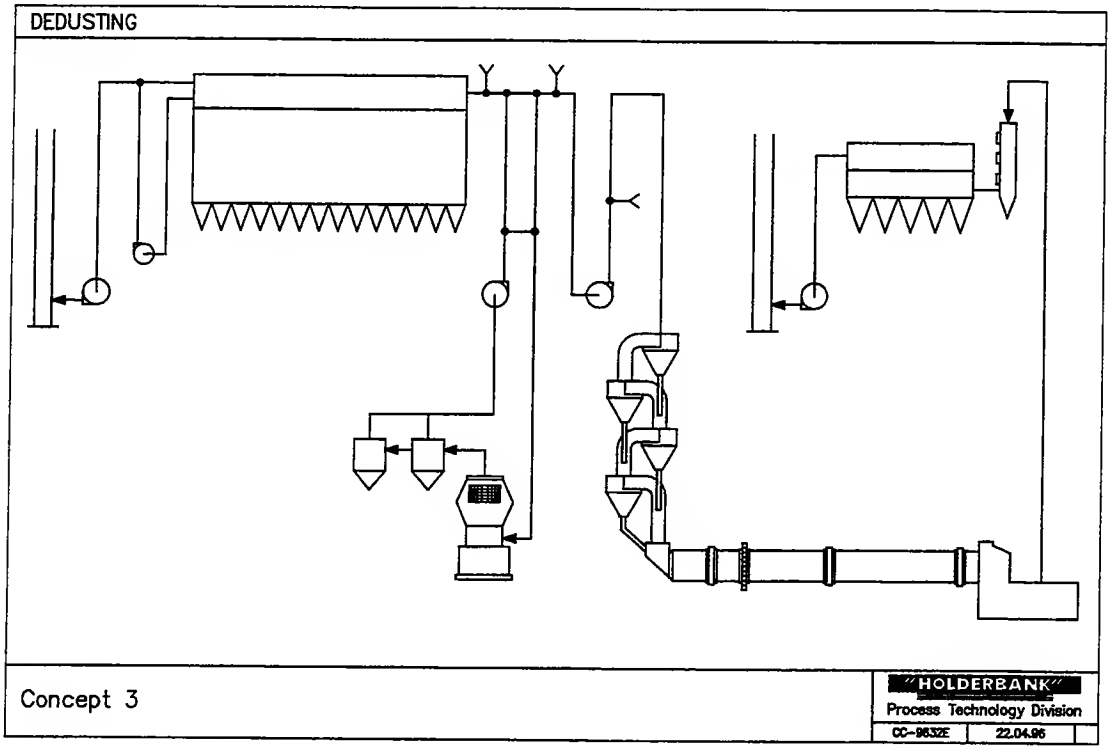
Figure 4 Concept 3



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5. CONCEPT 4

Separate dedusting of the kiln exhaust gas and clinker cooler vent air.

5.1 Kiln Exhaust Gas

5.1.1 Concept

- Cooling tower before ID fan to reduce the temperature to below 150°C during direct operation
- Fresh air intake to reduce the gas temperature to below 120°C
- 3 fan system (separate raw mill fan and cyclones)
- Jet pulse bag filter; preferably with polyacrylnitrile bags.

5.1.2 Advantages

- Length of duct between preheater and conditioning tower very short because the conditioning tower is placed ahead of the ID fan. Therefore, the investment costs are reduced.
- Good mill control because of separate mill fan
- Reduced investment and operating cost compared to reverse gas bag filter
- Less false air intake and reduced corrosion risk at the filter because of low negative static pressure (due to 3 fan system).
- No precollection required at the filter (due to raw mill cyclones)

5.1.3 Disadvantages

- Strong load changes on the ID fan because of different gas temperatures during direct and compound operation
- More false air intake into cooling tower and higher corrosion risk because of strong negative static pressure 860 - 40 mbar) compared to CT after ID fan
- Slightly higher pressure drop between ID fan and filter fan because of the raw mill cyclones
- Very reliable temperature control required to protect the bags
- Higher risk for corrosion at filter because of low operating temperature
- With a bad design it is possible that the pressure drop over the filter is increasing to a point where the capacity of the filter fan is not sufficient anymore to pull the gases.

5.2 Clinker Cooler

See Concept 3.

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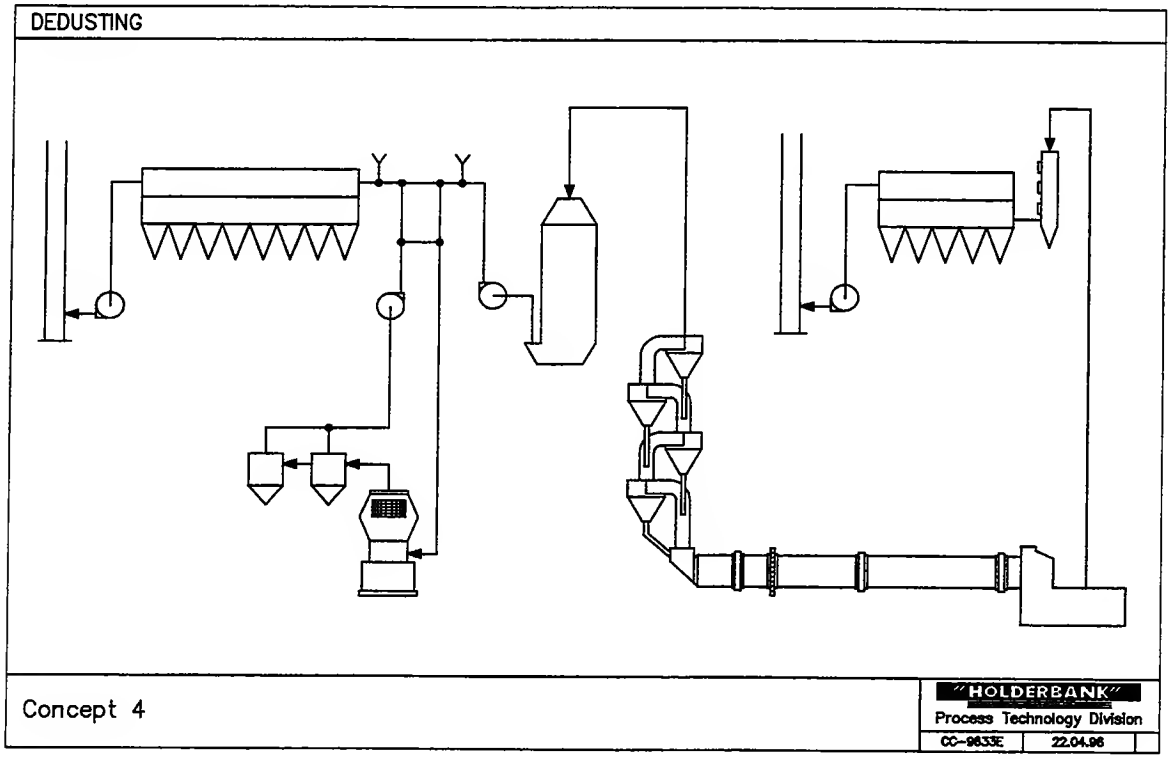
Figure 5 Concept 4



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6. CONCEPT 5

Simultaneous dedusting of the kiln exhaust gas and clinker cooler vent air in one jet pulse filter.

6.1 Kiln Exhaust Gas and Clinker Cooler Vent Air

6.1.1 Concept

- Reduction of the clinker cooler vent air dust concentration in a cyclone
- Direct operation: mixing of the clinker cooler vent air with kiln exhaust gas and reducing the temperature to below 120°C in an air to air heat exchanger
  Compound operation: reducing the clinker cooler vent air temperature and mixing it with the exhaust gas from the raw mill in a air to air heat exchanger
- 3 fan system (separate raw mill fan and cyclone)
- Jet pulse bag filter preferably with polyacrylnitrile bags

6.1.2 Advantages

- Only one filter
- No cooling tower
- No water injection
- Simple temperature control
- Reduced gas volume compared to all other solutions
- No precollection required at the filter (due to raw mill cyclones)
- Good mill control because of separate mill fan
- Reduced investment and operating cost compared to the other bag filter concepts
- Less false air intake and reduced corrosion risk at the filter because of low negative static pressure.
- Only one stack
- Clinker cooler gas can be used to dry raw material in the raw mill
- No visible plume at the stack because of the reduced dew point (no water injection and "dry" clinker cooler vent air)
- No load changes on the ID fan

6.1.3 Disadvantages

- Because of raw mill cyclones slightly higher pressure drop between ID fan and filter fan
- Kiln and clinker cooler not controlled by separate fan
- Portion of the clinker cooler dust is mixed with the kiln dust

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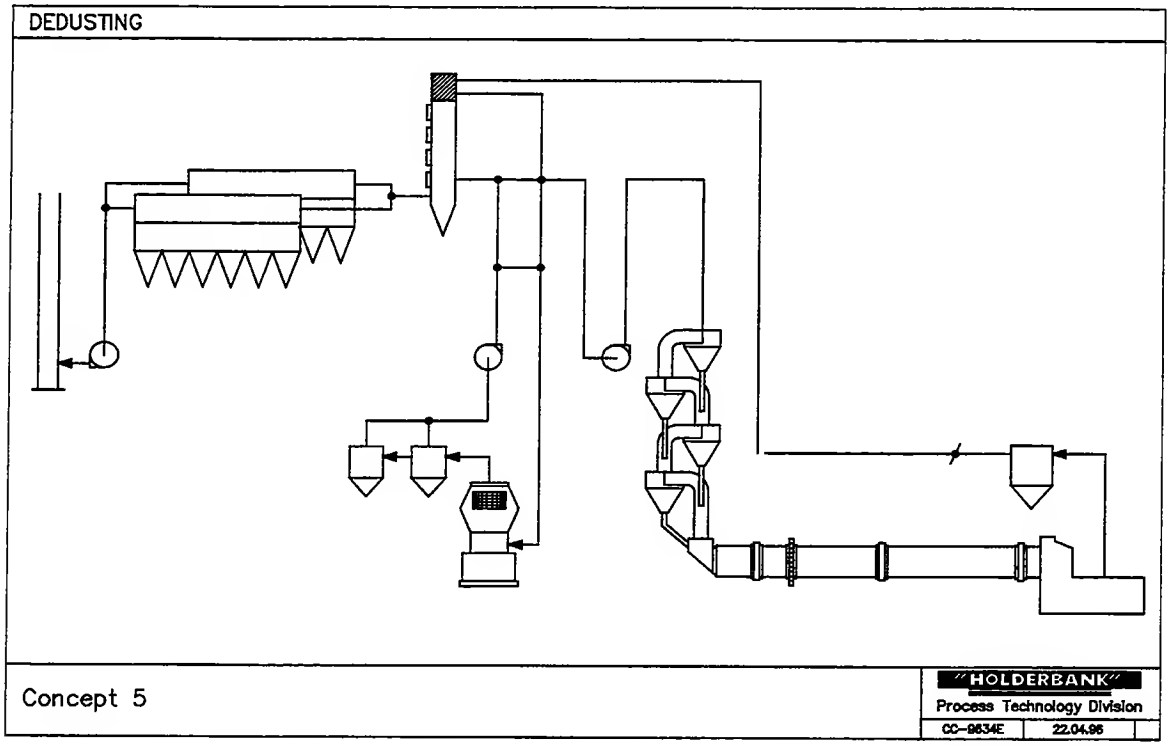
Figure 38 Concept 5



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7. CONCLUSION

All concepts have certain advantages. Therefore, one cannot produce a ranking without respecting the individual situation of the plants. Those that require a very reliable dedusting without short time dust emission peaks should chose a bag filter concept. In case of water shortage the concepts 3 and 5 without cooling tower are most suitable.

In case that bag supply cannot be guaranteed or short time dust emission peaks are accepted EPs may be the best solution.

If alternative fuels are burnt it is possible that CO peaks are produced more frequently than without, especially during the commissioning phase of the waste feed equipment. The plants that are burning alternative fuels or those that are planning to do so are usually under more intensive observation by the neighbors and the authorities. Frequent dust emission peaks caused by EP CO-shutdown or changes of process conditions may be very embarrassing when asking for a permission to burn alternative fuels or when applying for extension of the permit.

Therefore, bag filters, especially the pulse jet type, will for many plants be the preferred solution for the future.

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Chapter 3

Maintenance

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CONDITION MONITORING

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Summary

There have been many changes in the world of maintenance in the past few years. Predictive maintenance is one of the strategies to be used today to guarantee optimal performance at the lowest possible cost. Condition monitoring is a tool needed for predictive maintenance and the main requirement for it, is that it proves to be cost-effective.

On-line condition monitoring techniques are gaining importance in the field of maintenance, sometimes incorporating on-line diagnoses of the results. These include:

- Data from the process side since it can give a hint of a maintenance related problem.
- Expert systems and statistical process control (SPC) techniques for the evaluation of trends used to support decision making.

For a proper selection of a specific CONDITION MONITORING task it is crucial to know the characteristic of the failure it is meant to prevent.

Most failures give a warning that they are about to occur (potential failure). With condition monitoring deviations from the normal condition are detected. Based on these readings decisions can be made as to what corrective maintenance action has to be carried out and when.

Suggestions are made of guidelines for the proper selection and implementation of condition monitoring tasks and the condition monitoring techniques described.

1. THE MAINTENANCE ENVIRONMENT TODAY

The world of maintenance has drastically changed in the past few years. The requirements have been growing and still are. The main aspects that maintenance has to cope with today are:

- Higher Plant Availability and Reliability
- Greater Cost Effectiveness
- Greater Safety
- Better Product Quality
- No damage to the environment
- Longer Equipment Lifetime

The first two points are focused on, because they are easy to quantify by means of money.

Maintenance strategies have to be selected to produce the lowest overall cost possible.

Overall costs include:

- Prevention cost
- Repair cost
- Secondary damage cost (e.g. a bearing sizes due to a broken tube oil line)
- Loss of production (due to the down time of the equipment)

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2. TYPES OF MAINTENANCE TASKS

Apart from servicing it is possible to split maintenance tasks into three types.

Two of them do not need condition monitoring:

- Breakdown Maintenance; where the equipment is left in service until it fails. This can be both, dangerous and expensive.
- Scheduled Overhaul and Exchange; where the equipment, or part of the equipment, is restored or changed irrespective of its state. This is expensive and increases the risk of premature failures.

The third type strongly depends on condition monitoring:

- Predictive Maintenance; where the equipment, or part of the equipment, is to be restored or changed the moment before it fails. One could say "Just in Time".

3. WHAT IS CONDITION MONITORING

For many people "Condition Monitoring" is vibration analysis. In this paper we will look at it in a much broader sense.

Condition monitoring is everything which helps us to establish the state of equipment or the part to be maintained. Based on the findings, it is possible sometimes to estimate the residual lifetime of it.

According to standards [1], condition monitoring can be explained as:

- Measures to establish and evaluate the actual condition.
- It serves to recognize that repair work has become necessary at a time sufficiently early to allow preparation for such work, thereby permitting the work to be performed according to a schedule and avoiding secondary defects.

The term "Inspection" is also often used in this context.

Most failures give some warning of the fact that they are about to occur. This warning is called a potential failure. It can be defined as an identifiable physical (abnormal) condition which indicates that a failure is either about to occur or in the process of occurring.

With different techniques we intend to detect these deviations to the normal condition. These are condition monitoring techniques

Condition monitoring detects deviations to the normal conditions

The actual state of equipment can be established in different ways:

- Continuously or periodically
  On-line measurements or routine checks every certain time period
- Directly or indirectly
  E.g. weight belt / power consumption of the drive
- Qualitatively or quantitatively
  As a measured value or subjective (this noise is louder than normal)
- With or without instruments

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4. ADVANTAGES

Predictive maintenance and therefore condition monitoring is applied to reduce overall cost. That means substituting secondary damage and down time cost, by the expense to avoid them (prevention cost). Therefore:

Condition monitoring has to be cost-effective.

Based on experience condition monitoring very often proves to be cost-effective.

The benefits are:

- Prolonged equipment life time
- Minimized unscheduled downtime
- Fewer unnecessary overhauls
- Less stand-by equipment
- More efficient operation
- Increased safety
- Improved quality performance

5. IMPLEMENTING A CONDITION MONITORING TASK

The need for a condition monitoring task can come from various sources:

- Out of the risk assessment process [2]
- From a cost/benefit analysis
- Dictated by law
- etc.

The steps to establish a task can be seen on the flow sheet in Annex 1.

6. THE P-F-CURVE

For the suitable selection of a specific condition monitoring task it is crucial to know the characteristics of the failure it is meant to prevent.

The condition monitored has to have a correlation to the failure; e.g. there is no sense in monitoring the temperature if there will not be a temperature rise before the equipment fails.

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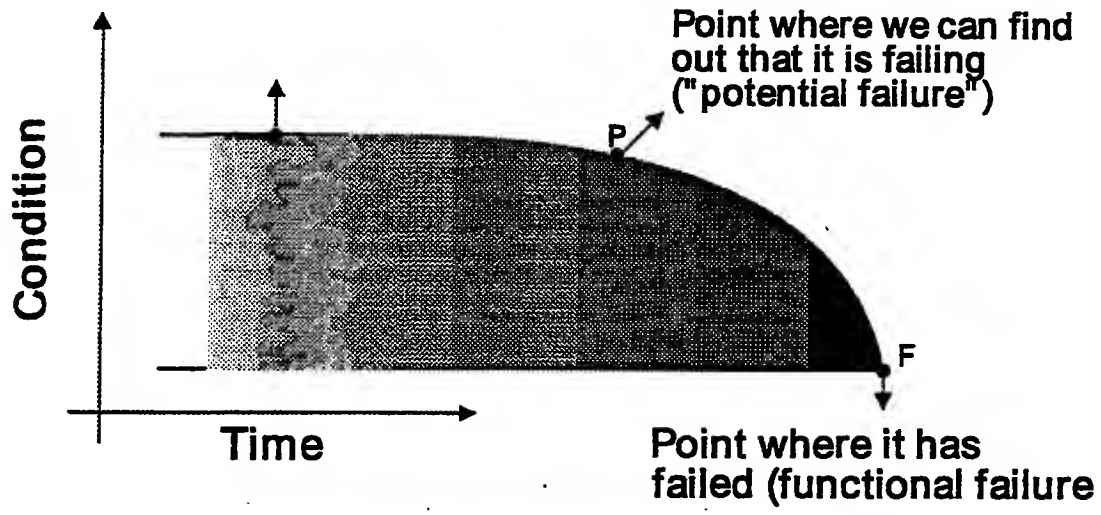
The characteristic's in which failures occur can be visualized in a diagram which plots the condition against time. This forms a curve degrading in time. An example with some explanations can be seen in the following picture.

The P-F-Curve



The most common curves are degrading faster in time or are linear.

The function of the curve is not necessarily age related. It can start at any time. But when a failure starts to occur it will progress according to that curve.

The P-F-Interval is the time taken between the occurrence of a potential failure (detection possible) and its decay into the failure itself.

In reality P-F-Interval's are not necessarily consistent. In fact they can vary over a considerable range of values. For most purposes the shortest P-F-Interval should be taken into account.

A sudden impact from the environment (e.g. Overload, foreign object, etc.) can cause a immediate deterioration of the condition into a functional failure. There is no P-F-Interval associated to these kinds of occurrences.

7. LIMITS

There are some limits for the application of condition monitoring.

The failure occurs without warning or too fast to undertake any action (P-F-Interval close to zero); there is no condition monitoring task to prevent it from occurring.

The deviations are too small to be detected or if it is impossible to establish limits for the condition to be monitored.

The P-F-Interval is so inconsistent, that no meaningful task interval can be established. The ultimate limit is given by the cost for the task in comparison to the money saved.

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8. HOW OFTEN IS A TASK TO BE PERFORMED

As a guidelines the frequency of a condition monitoring task has to be half of the (shortest) P-F-Interval of the failure. Therefore the frequency for the task depends mainly on two things:

- the characteristic in which a failure occurs
- the deviation needed to detect a potential failure

The characteristics of the failure is predetermined mainly by design and equipment operation. This fact does not give maintenance the possibility to act directly on them.

The only way to influence the frequency is to recognize a potential failure earlier in time. This means the deviation from the normal condition will be less and therefore the method of detecting the deviation has to be more sophisticated.

In the case of a detected abnormal condition (potential failure), the frequency can be adjusted if the repair task will not be carried out immediately and the future development of the failure is not known well (lack of experience, inconsistencies of the P-F-Interval). The idea is to have the equipment, or part, remain in service for as long as possible.

9. CONDITION MONITORING METHODS WITHOUT INSTRUMENTS

The basic condition monitoring methods are use of the human senses:

- sight
- sound
- smell
- touch

They exist as long as mankind and should not be forgotten even in the high-tech-times of today. These methods can be improved on by using simple instruments (magnifying lenses, mirrors, etc.). However, the disadvantage of inspections by human senses are that they are relatively unprecise, and therefore the associated P-F-Intervals are usually very short. Most of the smaller deviations tend to be beyond the range of the human senses and need specialized instruments to be detected.

10. CONDITION MONITORING METHODS USING INSTRUMENTS

We have learnt that a longer P-F-Interval means that the task needs to be done less often, and/or that there is more time to take whatever action is needed to avoid the consequences of the failure. In fact this will save money.

This is why so much effort is being spent on trying to define potential failure conditions and develop techniques for detecting them with the longest possible P-F-Interval (as early as possible).

Still one has to bear in mind that it has to be cost-effective.

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10.1 Types of Techniques

In this paper the techniques using instruments are divided into two main groups:

- On-Condition Techniques; where the equipment remains in service
- Inspection Techniques; where the equipment has to be shut down or even dismantled

The above mentioned division is not the only one, neither are the following ones.

Some of the techniques can be used for more than one purpose. In Annex 2 a table can be found with various methods and their possible applications.

10.2 On-Condition Techniques

On-condition techniques have the advantage that the equipment can remain in service. In some cases they even have to be in service; e.g. dynamic measurements. Therefore there is no production loss cost associated with this type of measurement and they do not need special co-ordination with the production. For this reason on-condition techniques have become popular over the past few years. However, they are normally more expensive and the results are sometimes difficult to interpret. Often it is necessary to take baseline readings and decisions have to be based on trends rather than on single measurements.

Very often this type of task is carried out on a regular basis by "inspectors" following a given route in the plant.

A possible way of dividing them into smaller groups is by type of measurements:

- Dynamic; e.g. Vibration Analysis, Acoustic Emission, Torques
- Temperature; e.g. Thermography, Fibre Loop
- Particle and Chemical Analysis; e.g. Spectrometric Oil Analysis, Ferrography
- Electrical; e.g. Meggers, Resistance

10.3 Inspection Techniques

Many inspection techniques are well known and have been used in the cement industry for a long time.

They include a wide range of solutions from simply determining the length with a tape measure to x-ray testing where expensive equipment is needed and good skills are necessary to handle them.

A possible way of dividing them into smaller groups is by the failure they detect:

- Surface Degradation (Wear, Corrosion, Cracks, a.s.o.); e.g. Magnetic Particle Test, Dye Penetration Test, Endoscopy
- Internal Deficiencies; e.g. Ultrasonic Measurements, X-Ray Testing
- Properties; e.g. Hardness Test
- Dimensions; e.g. Meters, Calibration, Shell test
- Alignment; e.g. Laser Distance Measurements, Lead Wire
- Leaks; e.g. Pressure Testing, Ultrasonic Leak Detection

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10.4 List of Techniques

In Annex 3 different condition monitoring techniques are described in detail. For every technique the following parameters are given:

- Condition moitored
- Applications
- Technical Base
- P-F-Interval
- Advantages/Disadvantages
- Skills necessary
- Standards applied
- Approx. Cost
- Supplier/Products

11. OUTLOOK

Taking into consideration all maintenance methods applied today a major movement towards on-condition monitoring techniques can be seen. This movement is expected to continue and to accelerate in the future.

Together with the on-line monitoring the on-line diagnoses of the results will gain in attraction, because the expert systems have, and still are becoming more powerful and easier to handle.

To involve process data into the condition monitoring is another step on the way to "Excellence in Maintenance". A higher power consumption of the equipment or a quantity/quality decrease of the product can be a hint of a maintenance related problem.

With the availability of long term data, statistical process control (SPC) techniques for the evaluation of trends can be used to help determine the actual condition of an equipment. Apart from others, two valuable tools to be applied for condition monitoring are:

- Moving X-bar Charts
- EWMA Charts (Exponentially Weighted Moving Average)

Both of them cope with the small amount of measured values available over the time period.

For the evaluation of on-line data (process or maintenance) a much wider field of SPC-techniques are worthwhile for consideration.

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12. CONCLUSION

Condition monitoring is a tool needed for predictive maintenance. It is one of the tools for state of the art maintenance to cope with the requirements that it is confronted with; basically to guarantee optimal performance at the lowest possible cost. Therefore, every condition monitoring task has to prove to be cost-effective.

For the proper selection of a specific task it is crucial to know the characteristic's of the failure it is meant to prevent.

Most failures give a warning of the fact that they are about to occur (potential failure). With condition monitoring, deviations to the normal condition are detected. Based on these readings, decisions can be made regarding what maintenance action has to be carried out and when.

As a guideline the frequency of a task has to be half of the P-F-Interval of the failure, i.e. half of the time to elapse between the possible detection of a potential failure and its decay into the failure itself. The only way to lower the frequency is to recognize a potential failure earlier in time. This means the deviation from the normal condition will be less and the method of detecting it has to be more sophisticated.

On-line techniques are gaining importance in the field of maintenance, sometimes incorporating on-line diagnose of the results.

Data from the process side has to be involved as well, since it can give a hint of maintenance related problem.

Expert systems and statistical process control (SPC) techniques for the evaluation of trends can be used to support decision making.

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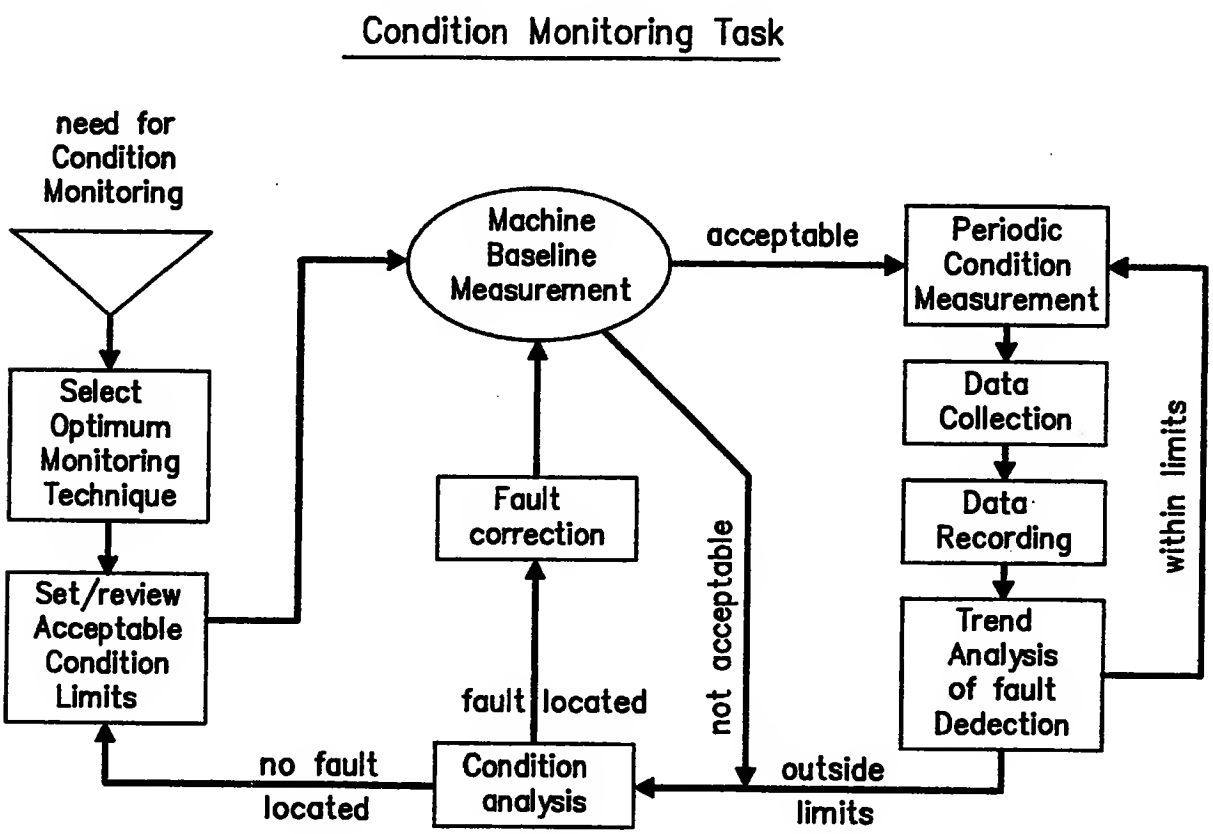
14. ANNEXES

14.1 Condition Monitoring Task



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14.2 Condition Monitoring Techniques and their Applications

14.2.1 Annex 2.1

On-Condition Techniques

MAIN APPLICATIONS / TECHNIQUE	Cracks	Wear	Fractures	Corrosion	Erosion	Leaks	Composition	Properties	Stress	Deformation	Dimensions	Deposits	Alignment	Loose/Thightness
Acoustic emission	X		X			X		X		X			X	X
Graded Filtration		X		X										
Ferrography		X		X										
Magnetic plugs		X												
Shelltest										X				
Lead Wire Test										X			X	
Thermometers,-Couples	X	X							X					
Spectrometric oil analysis		X		X										
Strain gauge									X	X				
Stroboscopy	X	X	X										X	
Test Coupons				X	X									
Thermography	X			X		X						X		X
Temperature Indicating Paint						X						X		
Vibration monitoring	X	X	X					X					X	X
Visual Inspection	X	X	X	X	X	X	X			X		X		X
Sight	X	X	X	X	X	X				X		X		X
Sound		X							X				X	X
Smell						X			X					
Touch		X						X						X
SPM (Shock Pulse Monitoring	X	X	X					X					X	X

14.2.2 Annex 2.2

Inspection Techniques

MAIN APPLICATIONS / TECHNIQUE	Cracks	Wear	Fractures	Corrosion	Erosion	Leaks	Composition	Properties	Stress	Deformation	Dimensions	Deposits	Alignment	Loose/Tightness
Dye penetrant examination	X		X			X								
Eddy current testing	X	X	X	X	X	X					X	X		
Endoscopy	X	X	X	X	X	X						X		
ER-probe				X										
Hardness testing								X						
Dimension Measurement		X		X	X						X	X	X	
Laser distance measurements		X									X		X	
Theodolit											X		X	
Leak testing						X								
Meggers/Voltage Generators								X						
Magnetic particle examination	X													
Engineer's Blue, Oilproof laquer										X			X	
Replica (Electron fractography)	X		X					X		X				
Pressure testing	X		X			X				X				
Radiography (X-Ray)	X	X	X	X	X	X					X	X	X	
Linear Polarisation Resistance				X										
Liquid Chromatography		X		X										
Ultrasonics	X	X	X	X	X	X		X	X	X	X			
Potential Monitoring				X										

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14.3 Annexes 3

14.3.1 Acoustic Emission

Conditions monitored

Plastic deformation and crack formations caused by fatigue, stress and wear.

Applications

Metal materials used in structures, pressure vessels, pipelines and mining excavations.

Technical Base

Stress waves are emitted by the materials which are subjected to loads, due to the crystallographic changes. These stress waves are received by a transducer and amplified at an impulse analyser and fed to a X-Y plotter or an oscilloscope. The curve is evaluated visually.

P-F Interval

Several weeks, depending on the application

Experience and knowledge necessary

Equipment functioning and interpretation of the results: An experienced and trained technician

Advantages

Remote detection of flaws: Covers entire structures: Measuring system set up very quickly: High sensitivity: Only limited access to test objects required: Detects active flaws: Only relatively low loads are required. Can sometimes be used to forecast failure load.

Disadvantages

The structure has to be loaded: A-E activity is highly dependent on materials: Irrelevant electrical and mechanical noise can interfere with measurements: Gives limited information on the type of flaw: Interpretation of results may be difficult.

Standards

ASTM in preparation

Estimated Costs

Ultraprobe USD 6'000.—

Stethoscopes USD 1'000.—

Supplier/Products

SPM/ELS-12

Keel Engineering (CH) / Ultraprobe

Westhill (South Africa)

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14.3.2 Graded Filtration

Conditions monitored

Particles in lubricating oils (such as iron, copper, lead, chromium, aluminium, silicon, etc.) caused by wear, fatigue and corrosion.

Applications

Enclosed lubricating and hydraulic oil systems, such as gearboxes, engine sumps, hydraulic systems, etc.

Technical Base

An oil sample is diluted and passed through a series of membranes (filters) with decreasing particle passing size. The collected particles are counted under a microscope according to the element and size. Its statistical distribution is shown in a graphical form. The analysis of the characteristics of the distribution of the particles shows whether the wear is normal or not.

P-F Interval

Usually from several weeks to months.

Experience and knowledge necessary

Sample: a laboratory assistant; analysis of the characteristics for the distribution of the particles: An experienced laboratory technician or an engineer.

Advantages

Can determine whether wear is normal or not. Relatively cheap. Can be used to compare one element with another.

Disadvantages

It is not an on-line technique: A high degree of experience is necessary to interpret the results of the sample. Identification of particle elements is difficult

Standards

ISO 4406

Estimated Costs

USD 2'000.—

Supplier/Products

Schak 01 (CH)

Cortec Corporation (USA)

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14.3.3 Ferrography

Conditions monitored

Wear, corrosion and fatigue

Applications

Enclosed lubricating and hydraulic oil systems such as gearboxes, engine sumps, hydraulics, etc.

Technical Base

Wear particles are separated magnetically from the lubricating oils onto an inclined glass plate by means of a instrument known as a ferrograph. The particles are distributed along the length of the slide according to their size. The slide is treated so that the particles adhere to the surface when the oil is removed. The total density of the particles and the ratio of large to small particles indicates the type of extent of wear and the analysis is made by means of a technique which is known as bichromatic microscopic examination. An electron microscope can also be used to determine the particles shapes and provide an indication of the cause of the failure.

P-F Interval

Usually several months

Experience and knowledge necessary

To extract the sample and operate the ferrograph: A semi-specialized operative suitable trained. To analyse the ferrogram: An experienced technician.

Advantages

More sensitive than the emission spectrometry at the at early stage of engine wear: measures particles shapes and sizes.

Disadvantages

It is not an on-line technique: measures only the ferrormagnetical particles: Requires an electron microscope for a more profound analysis.

Standards

Estimated Costs

Supplier/Products

BP, Mobil

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14.3.4 Magnetic Plugs

Conditions monitored

Wear and fatigue

Applications

Equipment with closed lubrication systems, such as reductors, collectors of motor oil, compressors, etc.

Technical Base

In the lubrication system a magnetic plug is mounted so that it is exposed to the circulating lubricant. The small metal particles in suspension in the oil and the unfastened metal scales due to fatigue, are captured by the magnetic force. The probe is taken out by regular intervals and the adhering particles are examined under a microscope. An increase in the quantity and size of the particles indicates a potential failure. The particles have different characteristics (form, colour and structure) according to the type and location of the failure.

P-F Interval

From days to weeks

Experience and knowledge necessary

To pick up a sample: A semi-specialized operator trained accordingly. To analyse the particles: An experienced and trained technician.

Advantages

It is a cheap method to monitor the contamination of liquids. Only a ordinary microscope is required to analyse the particles. Some plugs may be taken out during operation.

Disadvantages

Short P-F interval: Experience is necessary to interpret the results.

Standards

Estimated Costs

Supplier/Products

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14.3.5 SHELLTEST

Conditions monitored

The deformation of the shell of a rotary kiln, rigidity of shell and tyre and the play between them.

Applications

Shell of rotary kilns

Technical Base

The linear movements of a pin which is in contact with the surface of the shell is registered on a paper during a complete revolution of the kiln. There exist mathematical relations to the deformations of the shell. This procedure is carried out three times every 120° of the circumference of the kiln at various planes. Additionally the relative movement between the tyre and the shell is measured placing the pin on the shell and a plate with a paper on the tyre.

P-F Interval

Usually several months.

Experience and knowledge necessary

An experienced operator

Advantages

It is an on-line technique; long p-f intervals

Disadvantages

The evaluation of the results needs experience. Kiln operation does have a significant influence on the mesurement results.

Standards

Estimated Costs

Shelltest equipment USD 10'000

Supplier/Products

HMC "Holderbank"

Phillips

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14.3.6 Lead Wire Test

Conditions monitored

Alignment, contact area

Applications

Alignment of tyre / roller kiln

Technical Base

A lead wire is passed between the tyre and a roller. The moulding of the wire is drawn on a sheet. This procedure is carried out three times per tyre every 120° of the its circumference for both rollers. The form of the wire is evaluated for: alignment, straightness of the surface, contact area and stress points.

P-F Interval

Usually several months

Experience and knowledge necessary

No experienced or special knowledge is required

Advantages

Cheap and easy, it is an on-line technique.

Disadvantages

There is no quantitative result; additional measurements are required (oil film on the shoulders) for the evaluation of the alignment of the rollers. Kiln operation does have a significant influence on the measurement results.

Standards

Estimated Costs

< USD 100

Supplier/Products

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14.3.7 Thermometer, Thermocouple

Conditions monitored

Elevated temperatures due to wear, overloading, etc. at the surface or inside of an equipment

Applications

Refractory, reductors, motors, etc.

Technical Base

The electrical resistance of a wire changes with the temperature. The tension loss over the resistance is measured. In case of a PT100 the resistance at 20°C is 100 Ohm.

P-F Interval

From several hours to several days depending on the application.

Experience and knowledge necessary

Permanent installation: A specialist. Measurements with portable equipment: A suitable trained semi-specialized operator

Advantages

Cheap and easy to apply the portable equipment: often used as on-line measurement through the process control system

Disadvantages

In some cases the P-F interval is short - various hours

Standards

Estimated Costs

PT100: > USD 200

Digital thermometers: > USD 1'500

Supplier/Products

Rikenta (CH); Ahlborn (D); Endress + Hauser

Keithley, Bailey, Foxboro, Kent

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14.3.8 Spectrometric Oil Analysis

Conditions monitored

The wear of the following elements can be detected: iron, copper, lead, chrome, aluminium, molybdenum, tin, silver, zinc, nickel, silicon, sodium, boron

Applications

Circulating oil systems

Technical Base

The contaminants in a sample of oil are measured by emission or atomic absorption spectrometry. The emission spectrometry excites the metallic impurities in the sample with a direct high voltage (15'000 V), causing the impurities to emit characteristic radiation's which can be analysed.

The atomic absorption spectrometry works on the principle that every atom absorbs light of its own specific wave length. The oil sample is diluted and vaporised in an acetylene flame, and the presence of each element is determined using a light source of the appropriate wave length. In this way, the wear particles are identified, quantified and qualified so that the source of deterioration can be located. The graphs of the wear rates for each metal show deteriorating or improving conditions.

P-F Interval

Usually from several weeks to months.

Experience and knowledge necessary

To take out a sample: A semi-specialized, suitable trained operator. To operate the spectrometer: a trained laboratory technician. To analyse the results of the sample: an experienced chemical analyst.

Advantages

This test, of atom absorption, is comparatively cheap (sometimes it can be included in the service of the oil supplier): More reproducible at lower concentrations: Emission spectrometry is much faster than the atomic absorption spectrometry.

Disadvantages

Normally the analysis have to be made by specialised contractors (or oil suppliers) because the equipment and experience is lacking (long response time).

Standards

Estimated Costs

Supplier/Products

BP, Mobil

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14.3.9 Strain Gauges

Conditions monitored

Forces and deformations on parts

Applications

Gear reductor, structures, shafts, drives, etc.

Technical Base

The resistance of the wire changes linearly to the prolongation caused by the deformations. This resistance is measured by a bridge circuit.

P-F Interval

Several weeks to several months

Experience and knowledge necessary

An experienced technician is required to decide the position and affix the strain gauges as well as evaluate the results.

Advantages

Long P-F interval

Disadvantages

The measuring equipment are expensive, an external expert is required.

Standards

Estimated Costs

Supplier/Products

MEC; Brüel & Kjær

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14.3.10 Stroposcopy

Conditions monitored

Fissures, wear, direction, rotation speed and alignment in rotating and oscillating parts.

Applications

Rotary equipment such as shafts, ventilators, etc. for fissures or wear; check coupling for distance between the plates, loose screws, etc., balancing of rotary equipment (together with vibration measurement equipment)

Technical Base

The eyes follow the movement of one part in relation to the frequency in which it is illuminated. The frequency of the stroboscope is adjusted until the part appears to be stationary. Then the part can be inspected.

P-F Interval

Depends on the application; from several days to weeks

Experience and knowledge necessary

No experience or special knowledge is required.

Advantages

Easy to use and cheap, it is an on-line technique.

Disadvantages

Gives the impression that the part does not move, therefore security aspect should be kept in mind.

Standards

Estimated Costs

USD 500

Supplier/Products

Picostrob; Tourostrob

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14.3.11 Test Coupon

Conditions monitored

General and localised erosion and corrosion such as metal loss and pitting

Applications

Petrol refineries, process plants, gas transmission plants, underground structures, monitoring of cathode protection, abrasive slurry transport, water distribution systems, atmospherique corrosion.

Technical Base

Usually coupons are produced from mild, low carbon steel with a low coal content or of a grade material which duplicates the wall of a vessel or pipe. The coupons are carefully prepared, weighted and measured before they will be exposed. After the coupons have been submerged in the process flow for a period of time (from several weeks to several months) they are removed and checked for weight loss and pitting. From these measurements, the relative metal loss of the tube walls can be calculated and pitting can be estimated.

P-F Interval

Several months

Experience and knowledge necessary

A suitable trained specialist

Advantages

Very satisfactory when corrosion is constant: Useful in dangerous areas where the electrical dispositives are prohibited: Fairly cheap: Indicates the type of corrosion: Widely used.

Disadvantages

Results take time: The response to dangerous corrosive conditions is low: The use of coupons requires several personnel: The determination of the rate of corrosion usually takes several weeks: Information about temporal or abnormal conditions are not supplied.

Standards

Estimated Costs

Supplier/Products

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14.3.12 Thermography

Conditions monitored

Temperature variances caused by wear, corrosion, fatigue, leaks, poor electrical connections, etc.

Applications

Power transmission lines, transformers, refractories, electrical switchgear, building insulation, hydraulics, bearings, gas mains (dirt accumulations), etc.

Technical Base

Thermography extends the human vision to infrared waves. It is based on the principal that all objects above absolute zero (-273°C) emit infra-red radiation. An infrared camera that produces a live thermal picture detects this energy. Temperature differences of the surface are seen as light and dark areas and false colours. The thermal drawing can be recorded by a video camera attached to the display screen or directly onto a floppy-disk.

P-F Interval

A few days to several months depending on the application

Experience and knowledge necessary

Operation of equipment: A trained specialist. Interpretation of results: An experienced technician.

Advantages

Stationary or moving objects can be examined at any distance without touching or influencing the temperature of the object: Photographs and videotapes provide a permanent record: Examinations are carried out at safe distances from dangerous gases and high temperatures: Equipment portable and quick to use.

Disadvantages

Inaccessible components have to be uncovered: Costs: Needs specialist to interpret results: wide range of applications are needed to justify cost of the equipment.

Standards

Estimated Costs

Portable systems USD 1'500 - 6'000

Complete systems (including software) USD 50'-100'000

Supplier/Products

AGEMA: Thermovision

AHLBORN: Raynger

HENZ: Infratherm

Land (USA); Williamson (USA); Kane May

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14.3.13 Temperature Indicating Paint

Conditions monitored

Equipment temperature at surface

Applications

Refractories, thermal treatment

Technical Base

A chemical substance is applied to the machine surface. If the temperature exceeds the design temperature of the substance, the state changes from solid to liquid. Although the temperature is lowered afterwards, the traces of the liquid are still visible.

P-F Interval

From days to several weeks depending on the application

Experience and knowledge necessary

No experience of special knowledge is required.

Advantages

Cheap and easy to use; provides a maximal temperature indication

Disadvantages

Indicates only a temperature limit; several paints are needed to cover temperature range; Once the temperature has been exceeded a new paint is required.

Standards

Estimated Costs

USD 30 per tube or pencil

Supplier/Products

Medicina (FL): Tempil

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14.3.14 Vibration Monitoring

Conditions monitored

Changes in the vibration frequencies caused by wear, fatigue, corrosion, imbalances, disalingnment, loosening, etc.

Applications

Rotating and oscillating machines in general such as reductors, ventilators, motors, etc.

Technical Base

Vibrations are produced by the movement of the machine or a part of it. The most important characteristics that can be measured are: displacement, speed, acceleration, frequency, phase, using an accelerator or a speed sensor, a vibrometer, filters, oscilloscopes, etc. Several methods exist to evaluate the vibrations (e.g. wide band, broad band, octave band, etc.) each of them has its advantage for a special problem

P-F Interval

From days to weeks depending on the application

Experience and knowledge necessary

To operate the measuring equipment: an appropriate trained technician; To interpret the results: An experienced technician, sometimes of superior level

Advantages

Cheap and compact: Test during service; can be portable or permanently installed. The interpretation can be based on established acceptance criteria for the condition such as VDI 2056

Disadvantages

An in depth analysis requires an expert and a relatively long analysing time. Different evaluation techniques are needed to cover a case in detail

Standards

VDI 2056 ISO 3945

Estimated Costs

Simple Instrument USD 1'000

Date collector, including software approx. USD 30'000

On-line system for several measuring points > USD 60'000

Supplier/Products

IRD; Bruel & Kjaer; Karl Deutsch; Schenk, MAAG, ABB; SPM

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14.3.15 Visual Inspection

Conditions monitored

The range of conditions is enormous: Function, condition, surface, integrity, dimensions, shape, material, etc. of a piece of equipment in any industry not only for condition monitoring but also for damage analysis

Applications

For all equipment

Technical Base

Human sense is the technical base, sometimes supported with simple auxiliary instruments such as mirrors, lamps, etc.

P-F Interval

From several days to several months depending on the application

Experience and knowledge necessary

An attentive operator

Advantages

Most important method; cheap and easy, can be realised by operators of every level and education

Disadvantages

Human senses are limited: results are subjective and sometimes they can not be quantified.

Standards

DIN58220; DIN 8524; DIN 8563

BS 5289; BS 4080; ISO 3058; ASME sect V Art. 9

Estimated Costs

Nothing for inspections that are carried out during the daily work.

Supplier/Products

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14.3.16 Sight

Conditions monitored

The range is enormous: Function, condition, surface, integrity, dimensions, shape, material, etc. of an equipment or of a part of any industry not only for condition monitoring but also for damage analysis.

Applications

For all equipment

Technical Base

Use of eyes

P-F Interval

From several days to several months depending on the application

Experience and knowledge necessary

An attentive operator

Advantages

Very important method; cheap and easy, can be realised by on an every day base.

Disadvantages

Human senses are limited; results are subjective and sometimes they can not be quantified.

Standards

DIN 58220; DIN 8524; DIN 8563

BS 5289; BS 4080; ISO 3058; ASME sect V Art. 9

Estimated Costs

Nothing for inspections made by the operators during the daily work

Supplier/Products

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14.3.17 Sound

Conditions monitored

In majority: loosening, wear and forces

Applications

For all equipment or moving parts

Technical Base

Use of ears

P-F Interval

From several hours to several months depending on the application

Experience and knowledge necessary

An attentive operator

Advantages

Very important method: cheap and easy, can be realised by operators

Disadvantages

Human senses are limited, results are subjective and sometimes they can not be quantified.

Standards

Estimated Costs

Nothing for inspections made by the operators during the daily work

Supplier/Products

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14.3.18 Smell

Conditions monitored

In majority leaks and forces (overloads)

Applications

For all equipment or moving parts, electrical, tubes, tanks, etc.

Technical Base

Use of nose

P-F Interval

From minutes to several hours depending on the application

Experience and knowledge necessary

An attentive operator

Advantages

Cheap and easy, can be realised by operators

Disadvantages

Human senses are limited; results are subjective and sometimes they can not be quantified, P-F interval is very short.

Standards

Estimated Costs

Nothing for inspections made by operators during the daily work

Supplier/Products

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14.3.19 Touch

Conditions monitored

In majority loosening, wear and property

Applications

For all equipment

Technical Base

Use of hands

P-F Interval

From several hours to several months depending on the application

Experience and knowledge necessary

An attentive operator

Advantages

Cheap and easy, can be realised by operators

Disadvantages

Human senses are limited; results are subjective and sometimes they can not be quantified; can be dangerous (hot surfaces, splinters, etc.)

Standards

Estimated Costs

Nothing for inspections made by operators during the daily work

Supplier/Products

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14.3.20 SPM (Shock Pulse Method)

Conditions monitored

Shock waves due to fissures, wear, disalignment, insufficient lubrication, etc.

Applications

Rolling element bearings, pneumatic impact tools, valves of internal combustion engines

Technical Base

A accelerator detects the shock waves transmitted by the machine. The signals passes through a band pass filter which selects only frequencies exceeding 10 kHz. This high frequency input is converted into square pulses. The peak values of these pulses are read off as a measure of bearing damage.

P-F Interval

Depends on the application, but usually several weeks to months

Experience and knowledge necessary

An experienced and suitably trained technician

Advantages

Long P-F intervals: Equipment portable: Simple to use, on-line technique

Disadvantages

Not suitable for slow-moving machinery with high levels of product impact noise unless adaptive noise cancelling" is also used. Application is limited to a shock impulse measurement and transitory signals.

Standards

Estimated Costs

USD 3'000

Supplier/Products

SPM Instrument AG

AE Advanced Engineering, Rolle (CH)

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14.3.21 Dye Penetrant Examination

Conditions monitored

Surface discontinuities, fissures, etc. caused by fatigue, wear, surface shrinkage, grinding, heat treatment, laminations, corrosion, corrosion stresses

Applications

Ferrous and non ferrous materials such as welds, machined surfaces, shafts, boilers, plastic structures, compressor receivers, etc.

Technical Base

The penetrant liquid is applied to the test surface and sufficient time is permitted for it to penetrate the surface discontinuity. The excess surface penetrant is removed. A developer is applied which draws the penetrant from the discontinuity to the test surface where it is interpreted and evaluated. The liquid penetrants are categorised according to the dye type (visible dye, fluorescent or penetrates of double sensity) and the required procedure to eliminate them from the test surface (washable with water, post emulsified or solvent removed)

P-F Interval

From several days to several months depending on the application

Experience and knowledge necessary

To apply the penetrate: a semi-specialized suitable trained operator. Interpretation: A suitable experienced technician

Advantages

The sets of visible penetrate dye are very cheap but the fluorescent sets are a lot more sensitive. Detects surface discontinuities also on non ferrous materials.

Disadvantages

Fluorescent penetrates require an darkened area for inspection; Not an on-line technique, monitors only surface-breaking defects; It cannot verify materials with a porous surface; use is limited at temperatures from 0° to 50°C.

Standards

DIN 54152; BS 4416

ASTM-E 165; MIL-I-25135

Estimated Costs

USD 40 per set

Supplier/Products

ARDROX; MAGNAFLUX; CASTROL

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14.3.22 Eddy Current Testing

Conditions monitored

Surface and sub-surface discontinuities caused by wear, fatigue and stress, detection of dimensional changes produced trough wear, strain and corrosion, determination of material hardness.

Applications

Ferrous materials used for boiler tubes, heat exchanger tubes, hydraulic tubing, hoist ropes, railway lines, etc.

Technical Base

A test coil carrying alternating current between 100 kHz and 4 MHz induces eddy current in the part being inspected. Eddy current detours around cracks, becoming compressed, delayed and weakened. The electrical reaction on the test coil is amplified and recorded on a cathode ray tube or direct reading meter.

P-F Interval

Several weeks, depending on the application

Experience and knowledge necessary

An experienced and suitable trained technician.

Advantages

Applicable to a wide range of conducting materials. Can work without surface preparation. High defect detection sensitivity: Strip chart recorder provides a permanent record

Disadvantages

Poor response from non-ferrous materials. Usually an external specialist is required.

Standards

ASNT; DIN

Estimated Costs

from USD 3'000

Supplier/Products

Förster (Germany)

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14.3.23 Endoscopy

Conditions monitored

Surface cracks and their orientation, oxide films, weld defects, corrosion, wear, leaks

Applications

The internal visual inspection of narrow tubes, bores and chambers of engines, pumps, turbines, compressors, boilers, etc. in several industries

Technical Base

These instruments are known as endoscopes or borescopes. The light is channelled with mirrors or fibre cables to allow inspection of otherwise inaccessible points of view. If the light is insufficient, an external light can be sent through some of the fibre cables. For this an equipment producing cold light is used. It lightens the area so that photographs can be taken or video equipment used.

P-F Interval

Several weeks depending on the application

Experience and knowledge necessary

An experienced and suitable trained operator

Advantages

A detailed inspection of the surface in inaccessible areas can be obtained without having to dismantle the pieces; photographs can be taken to provide permanent records. They can be magnified; portable equipment.

Disadvantages

Only surface defects can be detected; Not an on-line technique; when equipment with cold light source, video camera, etc. is used, the method becomes costly

Standards

Estimated Costs

Inflexible: USD 2'000; with fibre cables USD 5'000-10'000

Complete systems: USD 30'000-50'000

Supplier/Products

Volpi; Classen + Co; Olympus

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14.3.24 Electrical Resistance (Corrometer)

Conditions monitored

Integrated metal loss (i.e.: total corrosion)

Applications

Petroleum refineries, process plants, gas transmission plants, underground structures, cathodic protection monitoring, abrasive slurry transport, water distribution systems, atmospheric corrosion

Technical Base

The system is composed of a probe and an instrument to read it. The probe consists of a wire, strip or tube of the same metal in the plant being monitored. The electric resistance of the probe which is measured by a bridge circuit, increases as the probe cross-section decreases with corrosion. This increase in resistance enables total metal loss to be read out which is easily converted to corrosion rate

P-F Interval

Depends on the application and on the corrosion rate. Usually several months.

Experience and knowledge necessary

A trained specialist

Advantages

When plotted against a time scale, yields both corrosion rate and total metal loss; Can be used in any environment. Portable equipment available. On-line monitoring possible: In-plant equipment provides permanent records: Interpretation normally easy.

Disadvantages

Gives no indication of whether the corrosion rate at a particular time is high or low; portable equipment provides no permanent record.

Standards

ASTM D 1776-79

Estimated Costs

Supplier/Products

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14.3.25 Hardness Test

Conditions monitored

Propriety of a material (hardness, crystallisation)

Applications

Shafts, gears, wear plates (i.e. clinker cooler), laminations, castings, welds

Technical Base

A test body is accelerated and collides with the test surface. The resistance of the material against penetration of the body into the surface is an indication of the hardness. Two types of evaluations are used depending on the method applied: A typical dimension (i.e.diameter) of the trace of the test body on the surface is measured or the energy difference of the test body is measured before or after the collision. For both methods the value is converted in hardness using tables.

P-F Interval

Depends on the application, but usually from several weeks to months

Experience and knowledge necessary

A suitable trained operator

Advantages

Rapid and simple measurements

Disadvantages

Application in a plant limited: inaccuracy of approx. 10%, Not an on-line technique. The measurement point has to be easily accessible.

Standards

Estimated Costs

Equotip USD 4'000; Poldi USD 600

Supplier/Products

Poldy Hammer

Equotip

Shore Hardness Tester

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14.3.26 Dimension Measurement

Conditions monitored

Dimensions of parts and dimension changes due to deposits or wear

Applications

For all equipment or parts such as shafts, bearings, tubes, etc.

Technical Base

There exist a wide variety of methods

A dimension is measured with a calibrator, a micrometer, etc. The results are in units of length and are compared with previous or basic values.

A dimension is compared with a calibre. The results are in a digital form , it "DOES" or "DOES NOT" have the required value including tolerances.

The contour of a part is formed with a wire and compared with the previous or basic contours.

P-F Interval

Depends on the application, but usually from several weeks to months

Experience and knowledge necessary

No experience or special knowledge is required.

Advantages

Cheap and simple to use and evaluate

Disadvantages

Difficult to apply to big dimensions; only a decision "YES" or "NO" in the case of calibre's, not an on-line technique

Standards

ISO

Estimated Costs

Very depending on the technique selected, from USD 100 for calibrators, micrometers, etc.

Supplier/Products

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14.3.27 Laser

Conditions monitored

In the majority dimensions or distances between two points are measured or it is used for alignment purposes

Applications

Kilns, transport ways, couplings, etc.

Technical Base

It is the modern form of the theodolite. A laser source is used to create an uniform and visible light of an exact wave length. This light is sent to a point on the surface. With a second instrument this point is also adjusted to this point, producing a triangulation measurements and evaluating them trigonometrically. With another Methode the beam is reflected and the difference at the source is evaluated.

P-F Interval

Depends on the application, but usually from several weeks to months

Experience and knowledge necessary

A suitable trained technician

Advantages

Rapid, can be measured without contact, measurement possible with the equipment in service.

Disadvantages

Relatively expensive, practice is required.

Standards

Estimated Costs

Coupling alignment systems USD 10'000

Supplier/Products

Optalign FLS (kiln alignment service)

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14.3.28 Theodolite

Conditions monitored

In the majority dimensions or distances between two points are measured or it is used for alignment purposes

Applications

Kilns, transport ways, coupling, etc.

Technical Base

Triangulation measurements and evaluating them trigonometrically

P-F Interval

Depends on the application, but usually from several weeks to months

Experience and knowledge necessary

A suitable trained technician

Advantages

Rapid and relatively cheap, can be measured without contact, sometimes possible with the equipment in service

Disadvantages

Practice is required, limited use in a cement plant

Standards

Estimated Costs

Supplier/Products

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14.3.29 Leak Testing

Conditions monitored

Leaks in tube systems and tanks, etc.

Applications

Distribution systems, tanks and vessel for oil, petroleum, lubricants, chemicals, liquid alternative fuels, etc.

Technical Base

The range of methods is so great that only the types are mentioned:

- by lost quantity
- by pressure differences: see also Pressure Test
- by tracing substances

P-F Interval

Depends on the application, but usually from several weeks to months

Experience and knowledge necessary

A semi-specialized suitable trained operator. In some countries a certification is required to be allowed to perform the tests

Advantages/Disadvantages

The range of techniques is so great that the advantage of one technique is the disadvantage of the other. Some are simple to use and cheap.

Standards

BS 3636; ASTM-E 432-71

Estimated Costs

Supplier/Products

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14.3.30 Voltage Generator

Conditions monitored

Resistance of electrical isolation

Applications

Electrical Circuits

Technical Base

The measurement is based on the ratiometer principle using two moving coils connected mutually at right angles (90°) within a permanent magnetic field. The reference coil is connected in series to a constant resistance, the other (deflecting coil) in series with the isolation resistance to be measured. The amount of deflection is a function of the resistance of the isolation. Test voltages of 250 to 10'000 V are used.

P-F Interval

From months to years

Experience and knowledge necessary

Operator or technician, depends on the voltage

Advantages

Simple and known technique

Disadvantages

No on-line technique

Standards

Estimated Costs

Supplier/Products

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14.3.31 Magnetic Particle Test

Conditions monitored

Surface and near-surface cracks and discontinuities caused by fatigue, wear, laminations, inclusions, surface shrinkage, grinding, heat treatment, hydrogen embrittlement, laps, seams, corrosion fatigue and corrosion stress.

Applications

Ferromagnetic metals such as compressor receivers, welds, machined surfaces, shafts, steel structures, boilers, etc.

Technical Base

A test piece is magnetised and then sprayed with a solution containing very fine iron particles over the area to be inspected. If a crack exists, the iron particles will be attracted to the magnetic flux leaking from the area caused by the discontinuity and give an indication. These leakage fields act as local magnets. The indication is then interpreted and evaluated. Fluorescent magnetic particle sprays provide greater sensitivity and inspection should be carried out under ultra-violet light in a darkened room.

P-F Interval

From days to months depending on the application

Experience and knowledge necessary

Application: a suitably trained semi-specialized operator, Interpretation: an experienced technician.

Advantages

Reliable and sensitive: relatively cheap and simple; independent of temperatures

Disadvantages

Detects only surface and near surface cracks: Time consuming: Contaminates clean surfaces: Not an on-line monitoring technique; only for ferromagnetic materials.

Standards

DIN 54130 and following; BS 4397

MIL 1949; ASTM-E and ASME-SE various

Estimated Costs

USD 3'000

Supplier/Products

Tiede; Magnaflux

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14.3.32 Oilproof Lacquer

Conditions monitored

Alignment, contact area

Applications Girth gear/pinion, gears

Technical Base

A coloured liquid is applied to the contact surface of one of the two parts. When the equipment is moved the coloured area comes into contact with the opposite part and leaves a "fingerprint" on it. Those are to be examined for colour distribution.

P-F Interval

Depends on the application, but usually from several weeks to months

Experience and knowledge necessary

No experience or special knowledge is required

Advantages

Cheap and simple to use

Disadvantages

There is no quantitative result; clean surfaces are required.

Standards

Estimated Costs

USD 50 per 0.2 litres

Supplier/Products

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14.3.33 Electron Fractography (Replica)

Conditions monitored

The growth of fatigue cracks

Applications

Metallic components in motor vehicles, industrial equipment, etc.

Technical Base

Every fracture has its own "fingerprint", in that the history of the fracture process is imprinted on the fracture surface. By studying a replica of the actual fracture surface with an electron microscope, it is possible to establish the causes and circumstances of failures.

P-F Interval

Depending on the application

Experience and knowledge necessary

Replica of the fracture surface: suitably trained technician. Analysis and reading: experienced engineer.

Advantages

Failures can be analysed with a high degree of certainty: No damage caused the actual to fracture surface when replica is made.

Disadvantages

Electron microscope is expensive: High degree of specialisation required to read the results: Not an on-line monitoring technique: Inaccessible components must be dismantled.

Standards

DIN 54150

ISO 3057

Estimated Costs

Supplier/Products

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14.3.34 Pressure Test

Conditions monitored

Leaks, fissures, fractures and deformations in tanks, pressure vessels, etc.

Applications

Tanks for, gas, pressured air, etc.

Technical Base

For security reasons the tests should be carried out using water or oil if possible. The systems have to be adequately ventilated. Higher pressures than required during operation are introduced to the system to see if it can withstand them. The percentage of overpressure required before rupture, depends on the safety regulations applicable.

P-F Interval

"YES" or "NO" decision

Experience and knowledge necessary

A semi-specialized suitably trained operator. In some countries certification is required before testing can be carried out.

Advantages

This test can be combined with a leak test, cheap and simple to do.

Disadvantages

Not an on-line technique, The components have to be emptied and cleaned before and after the test.

Standards

BS 5430

Estimated Costs

Supplier/Products

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14.3.35 Radiography (X-Ray)

Conditions monitored

Surface and sub-surface discontinuities caused by fatigue, stress, inclusions, lack of penetration in welds, gas porosity, intergranular corrosion and stress corrosion.

Applications

Ferrous and non-ferrous materials, welds, steel structures, plastic-structures, metallic wear components of engines, compressors, gearboxes, pumps shaft, etc.

Technical Base

A radiograph is produced by passing x-rays or gamma rays through materials which are optically opaque. The absorption of the initial x-ray depends on thickness, nature of the material and intensity of the initial radiation. The areas exposed become dark when the film is developed. The degree of darkening depends on the amount of radiation reaching the film. It will be darkest where the object is thinnest. A crack, inclusion or a void is observed as a dark patch.

P-F Interval

Several months

Experience and knowledge necessary

Use of equipment: a suitably trained and skilled technician. To interpret the results: a highly skilled technician or engineer.

Advantages

Provides a permanent record, detects defects in parts or structures not visually accessible

Disadvantages

Sensitivity often low on crack-line defects: Two-sided access sometimes needed; external expert required; security precautions elevated; costly

Standards

Estimated Costs

USD 6'000 - 300'000

Supplier/Products

Specialised companies

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14.3.36 Linear Polarisation Resistance (Corrator)

Conditions monitored

The rate of corrosion in electrically conductive corrosive fluids

Applications

Cooling water systems, municipal water systems, heat exchanger, desalination plants and pulp and paper mills and in the plants where the measurement and/or the corrosion control is required in acid water systems

Technical Base

Corrosion rate is measured by the electro-chemical polarisation method with two or three probes and a measuring instrument. The principle is based on the fact that a small voltage applied between a metal specimen and a corrosive solution will produce a current. The ratio of voltage to current is inversely proportional to the corrosion rate so it provides a measure of the corrosion rate increase.

P-F Interval

Depends on the application and the corrosion rate. Usually several months

Experience and knowledge necessary

A suitably trained operator

Advantages

Provides a direct indication of the corrosion rate and pitting tendency: Measures corrosion as it occurs: Some instruments provide a record of the corrosion condition: Automatic detection and control available: Sensitive to very low corrosion rates: Portable equipment available: Rapid measurement: Interpretation normally easy

Disadvantages

Portable equipment does not provide a permanent record: Readings must be adjusted when taken in high sensitivity corrosive media: Gives no information on total corrosion.

Standards

ASTM D 2776

Estimated Costs

Supplier/Products

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14.3.37 Liquid Chromatography

Conditions monitored

Changes in lubricant properties such as alkalinity, acidity, ash, flash point, insoluble, viscosity, etc.

Applications

Enclosed oil systems such as transformers, engine sumps, compressor sumps, hydraulic systems, etc.

Technical Base

Liquids are selectively absorbed by passing through a column of finely divided absorbent material. The separate liquids are then set free by passing a mixture of two solvent liquids, with different polarities, through the column. Light liquids appear first from the column and complex liquids last. The analysis appears on a strip chart recorder, or a screen, and the area under each peak is measured to determine the respective liquid concentrations

P-F Interval

Depending on the degradation rate of the lubricant and the application, but usually several weeks.

Experience and knowledge necessary

Operating the equipment and interpretation of results: a laboratory technician who has passed a course of chromatography.

Advantages

High sensitivity: Quick sampling and analysis: Strip chart provides a permanent record

Disadvantages

Considerable skill is needed to interpret results: Equipment not portable: Wide range of applications required to justify purchase: Not an on-line monitoring technique: not widely used in maintenance

Standards

Estimated Costs

Supplier/Products

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14.3.38 Ultrasonic

Conditions monitored

Surface and below surface discontinuities caused by fatigue, heat treatment, inclusions, lack of penetration and gas porosity in weld, laminations, etc. as well as material thickness.

Applications

Ferrous and non-ferrous materials related to welds, steel structures, boilers, boiler tubes, plastic structures, shafts, compressor receivers, etc.

Technical Base

A transmitter sends an ultrasonic pulse into the test surface and a receiver amplifies the return pulse to an oscilloscope. The echo is a combination of return pulses from the opposite side of the test piece and from any intervening discontinuity. The time elapsed between the initial and return signals and the relative height indicate the location and severity of the discontinuity. A rough idea of the size and shape of the defect can be gained by checking the test piece from another location.

P-F Interval

From several weeks to several months

Experience and knowledge necessary

An experienced and suitably trained technician

Advantages

Applicable to the majority of materials, relatively low costs, no expensive preparations needed

Disadvantages

Difficult to differentiate types of defects, evaluation relatively difficult; Problems with complex geometrical pieces, superficie has to be machined.

Standards

Estimated Costs

USD 8'000

Supplier/Products

Krauträmer; Karl Deutsch

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14.3.39 Potential Monitoring

Conditions monitored

Corrosive states of plant (active or passive) such as stress-corrosion cracking, pitting corrosion, selective phase corrosion, impingement attack, etc.

Applications

Electrolyte environments such as chemical process plants, paper mills, electrical generating plant, pollution control plants, desalination plants, etc. best suited to materials of stainless steel, nickel-based alloys and titanium

Technical Base

This technique takes advantage of the fact that, from the point of view of corrosion, a metal which is in a passive state (low corrosion rate) has a noble corrosion potential, while the same metal in an active state (higher corrosion rate) has a much less noble potential. The potential changes when passivity breaks down, and measurements can be made using a voltmeter of about 10 megohm input impedance and full-scale deflection of 0.5 to 2 volts

P-F Interval

Depends on the material and the corrosion rate

Experience and knowledge necessary

Signal detection is normally easy, but sometimes an experienced engineer is needed to interpretate de results further

Advantages

Monitors localised attack: Fast response to change

Disadvantages

Small potential changes can be influenced by changes in temperature or acidity: Does not give a direct measure of corrosion rate or total corrosion: Not widely used: Expert assistance may be required for interpretation

Standards

Estimated Costs

Supplier/Products

Rohrback Instruments

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Quality Inspections for Cement Plant Equipment

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Summary:

Quality is not an invention of the twentieth century. Conscientious manufacturers have always been endeavored to assure the good quality of their products. But through the demand for substantial cost reduction, manufacturers reduced or outsourced their internal quality control.

Suppliers also place their orders to manufacturers in low cost countries where quality control is not yet sufficiently observed.

At the same time cement machines change to always larger dimensions and capacities. With this trend new problems appear as well.

Our statistic shows that during the last few years only 60% of the items inspected are without any shortcomings. Out of the 40% rejected due to non-conformances, 2% could not be repaired and had to be discarded and re-fabricated. If not detected during the inspection, this would have caused substantial loss of production due to the down time, which means a loss of money and in the worst case even loss of customers.

The Vision for our Quality Control Equipment was therefore clearly defined.

No break downs in the "HOLDERBANK" group due to components failures on purchases of new equipment

The objective of quality services is always the profit of the purchaser. The investment into quality will be paid back in lower production costs as the items are more reliable (less down time), lower maintenance costs (breakdown maintenance) and a longer live time (reduction of NOA).

One of the most important points from the very beginning is to include adequate quality requirements (specifications) in the purchase order/contract to assure a trouble free roll-out. Following this procedure the quality requests have to be included in the contract and the supplier has to follow it. Each deviations from the specifications must be communicated with the inspectors and needs an agreement. After a seriously review of operating requirements it can lead into longer guarantee time, price reduction or manufacturing of a new part.

Finally, from HMC experience we can say: Quality depends very much on the suppliers care – and regular quality inspections remind them for this responsibility.

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1. INTRODUCTION

For the purchaser of a production plant, new machine or component the mayor problem is normally the loss of production due to equipment break down and less the cost of a replacement. The failed item is sometimes still under guarantee. Due to production loss, substantial loss of sales or even loss of customers can occur.

To avoid unexpected down time due to manufacturing defects of items, companies started to contract independent inspectors to supervise the quality control of the manufacturer. The history of the quality inspection service at "HOLDERBANK" shows typically this evolution.

Recalculating the main items as girth gears, gear boxes and kiln tyres to influence its design was the first step. Anyhow, damages still occurred and it was found that some of the reasons were manufacturing defects. The experience with quality inspections showed that still nowadays it is necessary to supervise the fabrication through an independent body. This not only because the suppliers move their production into low price countries, but also because sometimes there is poor quality produced even in well-known companies with normally good products.

Consequently "HOLDERBANK"'s objectives were set:

- **Achieve high reliability of the equipment at adequate cost**
- **Maximize effective life time of assets**
- **Form the suppliers to deliver the expected quality**
- **Train our inspectors in supervising efficiently and precisely the quality inspections**

Full responsibility for the products remains with the supplier and the quality inspections have to be cost effective. Therefore our inspections are limited to important items as mentioned in **Annex 1.**

2. OBJECTIVE OF QUALITY INSPECTIONS

The objective of quality inspection services is always the profit of the purchaser. The investment into quality will be paid back in lower production costs as the items are more reliable (less down time), lower maintenance costs (breakdown maintenance) and have a longer life time (reduction of NOA). Further valuable information with regard to applied materials and procedures, which normally are not supplied by the manufacturer, are handed out.

3. EXAMPLES OF DAMAGES

In a cement plant the catalogue of possible examples of damages is quite long, e.g.

- Cracked / fractured girth gears and pinions
- Cracked / fractured kiln tyres, rollers, shafts
- Cracked kiln or mill shells
- Cracked mill end-castings and flanges
- Damaged roller mill bodies
- Damaged roller press rollers etc.

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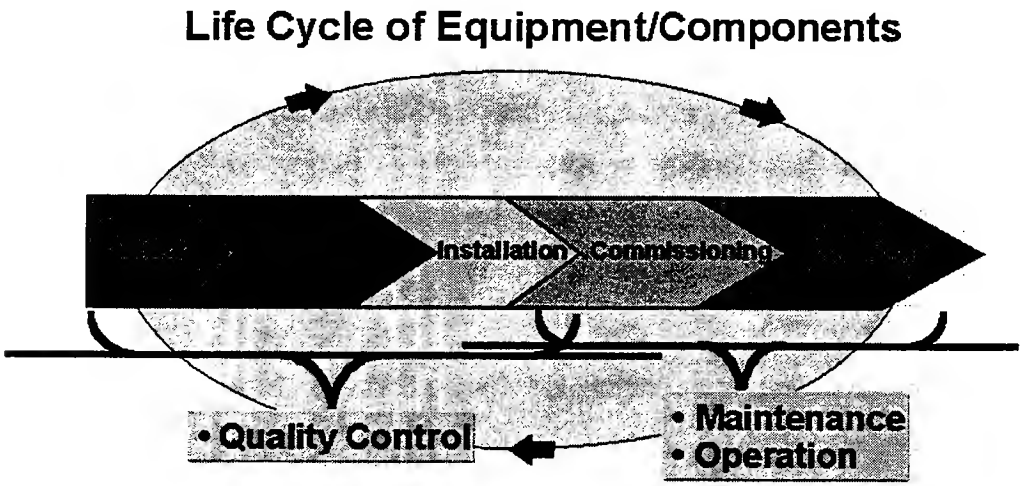
4. REASONS FOR DEFECTS

Reasons for defects are many fold. In some cases it is lack of knowledge, unfavorable design or a price that was too low for a good quality product. It may be added that the production of large castings not only demands excellent knowledge and modern facilities but also involves a great deal of luck. Errors, such as insufficient supervision or checks, a carefree attitude, fatalism and taking calculated risks up to deliberate attempts to cheat have all been observed.

The life cycle of an equipment/component explains very well that the quality of a part is "made" at different stages, starting with the design (specification) followed by the manufacturing (inspections) and finally at the installation.



The quality inspections focus on the manufacturing of the equipment/component, which is a very important step in its life. Defects can originate in every manufacturing step: casting, forging, welding, machining or assembling. The most common defects are:

- Material inhomogeneities
- Cracks
- Surface deficiencies
- Dimensional/Geometrical errors

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5. INSPECTION METHODS / TESTS

During the course of fabrication of components / machines quite a series of quality tests are carried out and different inspection methods are applied.

- Destructive Testing (Mechanical tests and Chemical analysis)
- Non-destructive testing
  - Ultrasonic test
  - Magnetic particle test
  - Dye penetration test
  - Hardness test
  - Dimensional check
- Verification of procedures / documents

Destructive Testing and Chemical Analysis

Materials samples taken from the component in question are tested for the ultimate strength, yield strength, elongation, reduction of area and impact energy. Their chemical composition is analysed as well. The values received are compared with current standards and specifications.

Visual Testing

Visual testing includes checking of general appearance, surface quality, dimensions, conformity with drawing, functions and is the most often employed method and the one which has proved its worth for acceptance inspections of cement equipment.

Ultrasonic Testing

The most difficult of the above mentioned test methods might be the check for internal defects with ultrasonic sound waves. The microstructure of the material, the geometry and surface quality of the test piece has an important influence on the sound waves. A broad and permanent experience is needed to interpret the respective signals on the screen.

A wide variety of ultrasonic probes is available, each of them has its special features. For the inspection of cement equipment it is not possible to take advantage of all technical possibilities (as for nuclear equipment) owing to the high cost. Therefore, a certain risk of failure during operation can not be excluded.

Magnetic Particle Testing

A test piece is magnetized and then sprayed with a solution containing very fine iron particles over the area to be inspected. If a crack exists, the iron particles will be attracted to the magnetic flux leaking from the area caused by the discontinuity and giving an indication. These leakage fields act as local magnets. The indication is then interpreted and evaluated. Fluorescent magnetic particle sprays provide greater sensitivity; the inspection should be carried out under ultra-violet light in a darkened room.

Dye Penetration Testing

The penetrant liquid is applied to the test surface and sufficient time is permitted for it to penetrate the surface discontinuity. The excess surface penetrant is removed. A developer is applied which draws the penetrant from the discontinuity to the test surface where it is interpreted and evaluated. The liquid penetrants are categorized according to dye type (visible dye, fluorescent or penetrates of double sensity) and the required procedure to eliminate them from the test surface (washable with water, post emulsified or solvent removed).

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Hardness Testing

A test body is accelerated and collides with the test surface. The resistance of the material against penetration of the body into the surface is an indication of the hardness. Two types of evaluations are used depending on the method applied: A typical dimension (i.e. diameter) of the trace of the test body on the surface is measured or the energy difference of the test body before and after the collision is measured. For both methods the value is converted in hardness and/or strength using tables.

Verification of Procedures / Documents

No quality inspection is complete without the verification of the welding and heat treatment procedures/parameters (if applicable) and the complete documentation. This documentation includes all forms completed during the manufacturing and the quality inspection. It has to come along with the part for further reference at the plant in case of a failure during the operation phase.

Inspectors Qualifications

The inspectors carrying out quality checks on components and machines during fabrication must have knowledge, sufficient training and practical experience in the different testing methods and management of such jobs.

Documents and Arrangements necessary for Quality Inspections

In order to be able to carry out tests, certain specific information and documents about each component must be available. Also the necessary arrangements for the inspectors must be provided by the supplier. These needs are listed in **Annex 2** which should become part of the purchasing contract in case HMC is entrusted with the quality control.

6. SUPPLIER'S / MANUFACTURER'S INSPECTIONS

In most cases the supplier is not or only up to a certain degree the manufacturer. He plans his order to a sub-supplier, e.g. a foundry, forge, gear maker etc. The manufacturer (sub-supplier) must perform quality inspections himself at various stages of manufacturing according to contractually agreed specifications, standards and procedures. He is, together with the supplier, fully responsible for ensuring that his products meet the specified qualities.

In practice however we experience too often that the manufacturer does not bear sufficiently the responsibility for the quality of his products. Statistical data collected by "HOLDERBANK" show that only about 60% of the items inspected were without any shortcomings.

Out of 40% rejected due to non-conformances, 2% could not be repaired and had to be discarded and re-fabricated **(see Annex 3).** If not detected during inspection, many of these cases would have later caused substantial loss of production due to downtime of the installation.

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7. STEPS OF QUALITY ASSURANCE

In order to avoid the above-mentioned shortcomings and non-conform behavior of manufacturers/suppliers it is recommended to engage an independent quality inspection company. This should, whenever possible, take place before signing the purchasing contract, specially when working with a new supplier.

The consultant can help to define and verify the expected quality level of the products as well as the inspection methods / procedures to be applied. The various steps of the proposed proceedings are shown in **Annex 4.1 and 4.2.**

If the inspection company is contracted late, the purchaser will not get the full profit of their experience. Also the work for the company becomes much more difficult, since the manufacturer will not accept any directions from them towards better quality. Especially if it would cause additional work for fabrication or testing and if such activities will add cost (reduce his margin).

In the worst cases experienced, at the time of involvement of the inspection company, fabrication had progressed so far that only a final inspection was possible. If there are any mayor deficiencies detected then, they can not be corrected anymore or only with a substantial loss of time and money. Both endanger the delivery and installation schedule of the item.

8. QUALITY INSPECTION PROGRAM

Not all components and machines are in the same way critical for failures or for the time when failures occur. HMC established a list of the most important items regarding

- Highest stress
- Influence on the operation reliability
- History of similar items
- Long delivery
- High cost items

The list is presented in **Annex 1.** For each component mentioned its priority for inspection is also indicated (1= high priority).

Depending on its type of material, way of fabrication and complexity every component has an individual inspection program. While some are only checked once at the end of fabrication, others are checked two or three times at different stages. In **Annex 5** a typical inspection program with three inspections is shown:

- After casting and pre-machining (before any welding repairs are carried out)
- After construction/repair welding and heat treatment
- Final inspection after completion

Certain tests are performed on a random basis others are full-scale tests. The extent of the non-destructive test amounts generally to 100%, but can be reduced according to requirements.

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9. HMC AS AN INSPECTION COMPANY – BENEFITS

HMC members of the inspection team can look back on a 20 years experience in machine techniques, maintenance and trouble shooting. Operational problems are therefore not new for them.

HMC is now on the way of establishing their own inspection specifications for the key components. These can then be used as a "bar" for new suppliers specifications or directly for placing an order.

Whether HMC sends their own or assigned inspector, all of them are highly trained and experienced inspectors. Training and knowledge exchange is a permanent issue also in the field of quality control.

Once a customer entrusts HMC with the quality assurance and inspection of their purchases, an independent party will watch his interests as if they were their own.

Some specific benefits are:

- The risk of unexpected sudden and premature failure is generally largely reduced. Such a failure could incur a downtime for carrying out of a (temporary) repair up to 3 weeks, resulting in a great loss of production with corresponding high monetary loss.
- If the quality inspection reveals shortcomings or deficiencies, action can be taken in time to avoid or shorten a delay in the delivery schedule of the item.
- Information that is not included in the documentation from the supplier will also be provided to the purchaser.
- An item unacceptable from a quality point of view will be disclosed to the client, the supplier will have to provide a replacement of acceptable quality.
- Due to the long time we have been in this business, we know most of the respective suppliers and their products very well; all the particularities and specialties, but, most of all, also their weak points. This allows for more effective inspections.
- Due to the well-structured documentation and the valuable hints given, lower maintenance cost can be expected and the lifetime of the parts may be increased.

Based on their knowledge, our inspectors can decide on the spot about the impact of any such problem. Furthermore, they have the full back-up support of our department. Additionally, we are always able to call in specialists within our organization for process matters. This makes the inspections very efficient and effective, i.e. less time (and money) is needed.

All this will help that erection, commissioning and operation will be more efficient and economic.

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10. CONCLUDING REMARKS

Quality inspections are not only a matter of costs but also an investment for the lifetime of new products and will be paid back by lower operation costs (reliability, lifetime). Besides the assurance of greater reliability in operation they also provide valuable information on the product. Therefore purchasers should look at it in the same manner as at an investment for a new line or an upgrade of a plant. However, no quality can completely exclude all risks.

The quality requirements have to be determined together with the supplier and set down in the acceptance specification before signing the contract. They should actually form an integral part of the purchase contract or order.

In the long run the collaboration between the customer (plant) and the manufacturer/supplier has to become closer, so that only good quality items are delivered, even if there is no third party carrying out additional inspections. Unfortunately we are still far away from that point.

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11. ANNEXES

Annex 1

Recommended Item to be Inspected	No. of Inspections	Inspection Priority
Kiln		
Kiln Roller	2	1
Kiln Roller Axle	1	1
Thrust Roller	1	2
Tyre	2-3	1
Tyre Shell Section	1	1
Shell Section (10 pieces)	1	2
Planetary Carrier	1	2
Tube Mill		
Mill End	2-3	1
Mill Flange	1-2	1
Mill Body	2	1
Mill Shell Section	1	1
Mill Shell with Tyre	2-3	1
Tube Mill Tyre (Bandage)	2	1
Slide Shoe	1	2
Vertical Mill		
Grinding Table	2	1
Roller Yoke (Polysius)	2	1
Roller Carrier (FLS, Pfeiffer, etc.)	1	1
Rocker Arm (Loesche, Onoda)	1	2
Fork (Loesche, Onoda)	1	2
Roller Body [if GGG]	1-2	2
Roller Shaft	1	2
Table Liner (Segment)	1	2
Roller Wear Segment/Tyre	1	2
Roller press		
Roller Body (Studded/ Welded Design)	1	1
Roller Segments	2	1
Roller Shaft	1	1
Bearing Block	1	2
Roller Press Frame	1	2
Drives		
Girth Gear (Cast)	3	1
Girth Gear (Welded)	2	1
Pinion	1	1
Pinion Shaft	1	1
Mill Gear Drive	2	1
Mill Gear Planetary Carrier	1	1
Kiln Gear Drive (Mechanic Standard)	1	2
Kiln Gear Drive (Mechanic Direct)	1	2
Kiln Gear Drive (Hydrostatic)	1	2
Roller Press Drive	1	1
Various		
Hammer Crusher Shaft	1	1
Main Process Fan	1	2
Reclaimer Chain	1	2

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Annex 2

Documents and Arrangements necessary for Quality Inspections

The following documents are to be provided to "Holderbank" at least one month before inspections:

- Fabrication program
- Inspection program of supplier
- Manufacturer for each item to be inspected (company address, telex, fax, telephone-No., responsible person for quality inspection, place of fabrication)
- Acceptance specifications for each item to be inspected (acceptance criteria, state of item when inspections take place, scope, etc.)
- Applied standards for non-destructive testing and other inspections
- Drawings (fabrication and assembly if available)
- Guarantee clauses of contract for items to be inspected

Documents and certificates to be provided to "Holderbank" during inspections or latest at the final inspection (two sets):

- Non-destructive testing
- Surface quality
- Hardness test
- Materials (chemical composition, mechanical properties)
- Record of dimensions
- Heat treatment (indicating temperatures, heating, holding and cooling time)
- Welding (indicating properties of electrodes, preheating and post heating procedures, type of welding, welder legitimation)
- Map of defects of repair welds (indicating place and size of defects/repair welds)

All the above mentioned documents have to be delivered in the language specified in the contract. This also applies to documents from **sub-suppliers.**

Arrangements of Inspection

The HMC's inspectors and additional inspectors from the client must be given access to manufacture at any time (time to be co-ordinated in advance).

The HMC's inspector must be given due notice (at least 5 working days in advance) of the individual inspection stages by the supplier. Provision should be made by the supplier to allow the examination of several items at the same time (if possible) to minimize traveling expenses.

All inspections will be carried out together with the supplier whereby all tests and measurements are to be performed by the manufacturers' personal using their own instruments and equipment.

In case of suspect results of inspections the HMC inspector has the right to demand additional suitable inspections.

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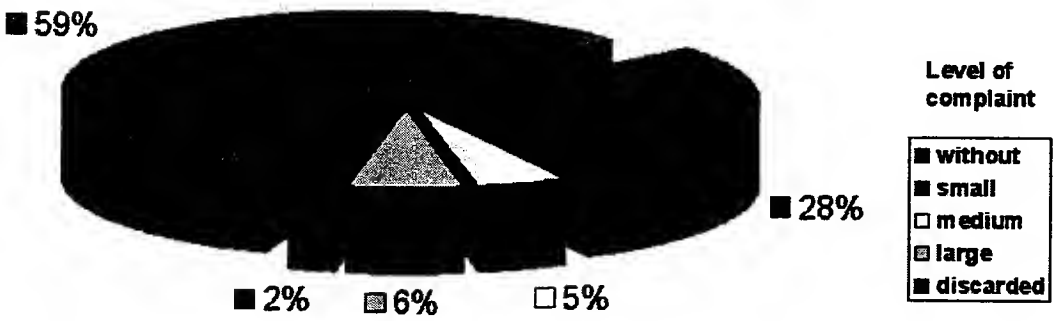
Annex 3

QCE-Statistics on Quality Complaints



Revision Date:
31.12.98

Total number of pieces checked: 209

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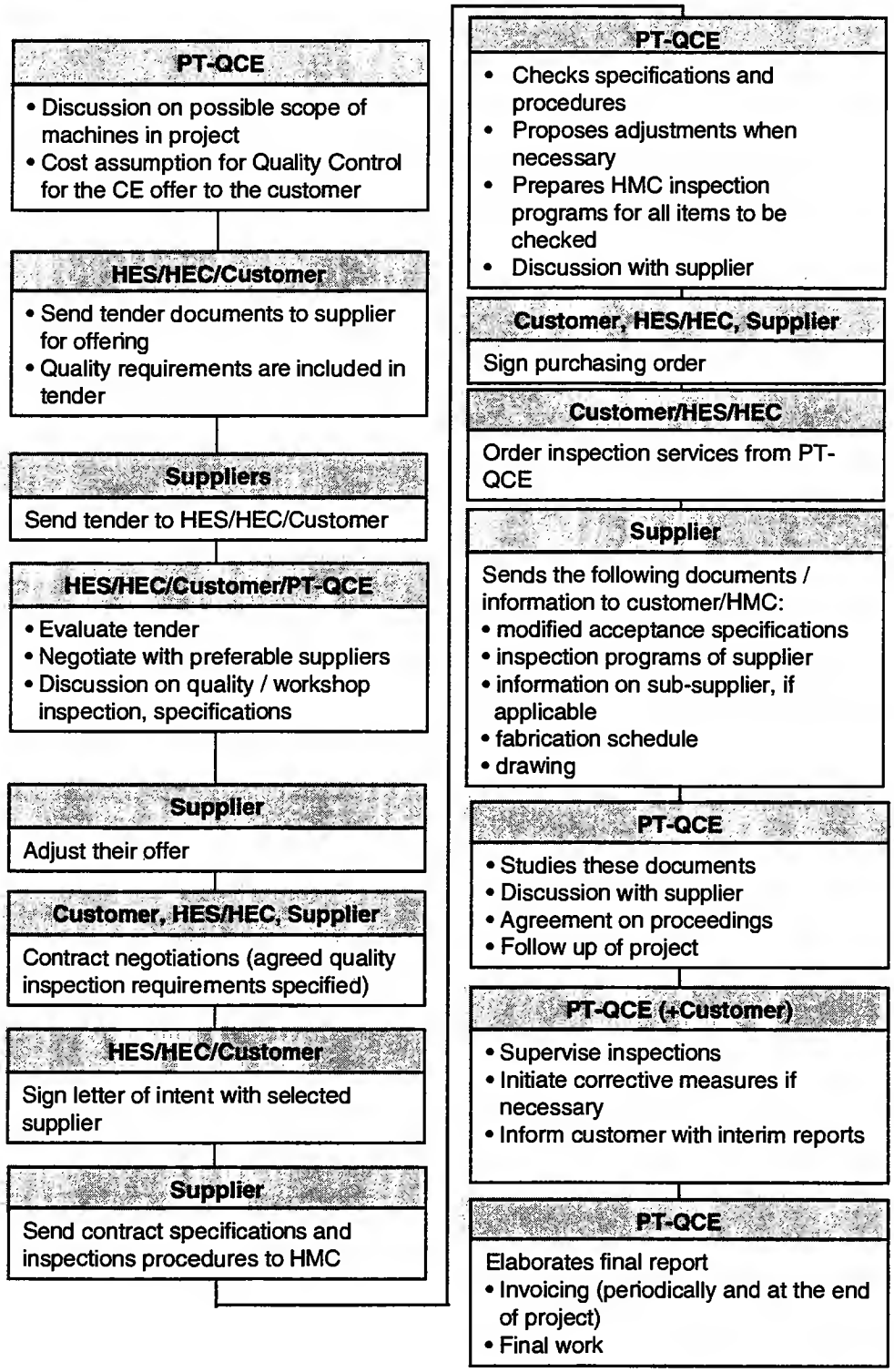
Annex 4.1

Quality Control of New Equipment



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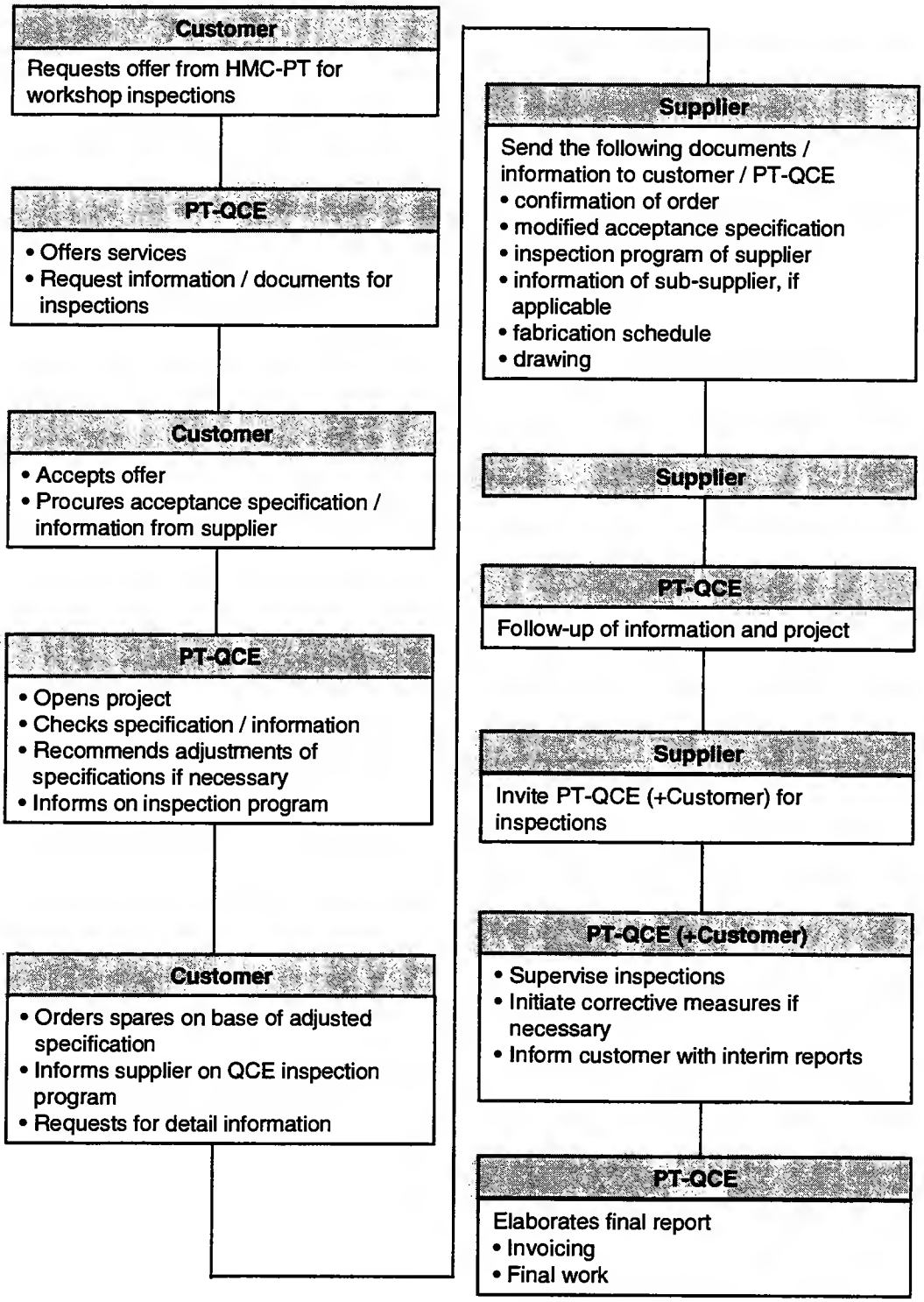
Annex 4.2

Quality Control of Individual Spare Parts



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Annex 5

GIRTH GEAR (CAST)

INSPECTION PROGRAM

For the quality assessment of a cast girth gear three inspections are to be carried out at different stages of the manufacturing.

First Inspection

After casting and pre-machining (before any welding repairs are carried out)

- Visual inspection
- Ultrasonic test of the internal material homogeneity (100%)
- Magnetic particle test of the surface in general and the areas of casting defects (100%)
- Dimensions
- Review of repair welding and heat treatment procedure

Second Inspection

After repair welding and heat treatment

- Visual inspection
- Magnetic particle test of repaired areas
- Ultrasonic test of the repaired areas
- Witness of material strength test (tensile and yield strength, elongation and reduction of area, impact work)
- Hardness test across repair welds and at gear rim
- Dimensions

The second inspection can be dispensed if only minor repair welding is necessary. In this case the material strength test would be carried out either during the first or the last inspection.

Third Inspection

Final inspection after completion. The two gear halves must be assembled.

- Visual inspection
- Magnetic particle test of tooth (100%) and remaining areas
- Hardness test of tooth flanks
- Surface roughness test of teeth
- Tooth rectilinearity (if straight teeth)
- Tooth contact of pinion with girth gear (if possible)
- Dimensions in general and of toothing in particular
- Review of documents and certificates

The extent of the **Non Destructive Tests** amounts generally to **100%.** It can be reduced on the HMC inspector's decision.

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MAINTENANCE

18th International
Cement Seminar 2000
CTS / MTC
May 17, 2000

Who are we?

Maintenance Team "Holderbank"

- To be the partner of "HOLDERBANK" group companies in the area of maintenance.
- Support the plants / companies in their goals of optimizing overall asset effectiveness

Presenting at the Cement Seminar today:

- **Hans Burger**
- **Walter Bürgi**
- **René Burkhalter**

Objective of Today

Objectives:

- Give an idea of the current strategies and trends in maintenance
- See some practical examples about maintenance
- Share experiences in the field of maintenance

Program of Today

Morning (8:00 - 12:30)

➢ Introduction to maintenance	Hans Burger
➢ Practical Example - Ball Mill	Walter Bürgi
• Case Study	

Afternoon (13:30 - 17:05)

➢ Condition Monitoring	René Burkhalter
➢ Quality Assurance	René Burkhalter
➢ Summary and Conclusions	Hans Burger / René Burkhalter

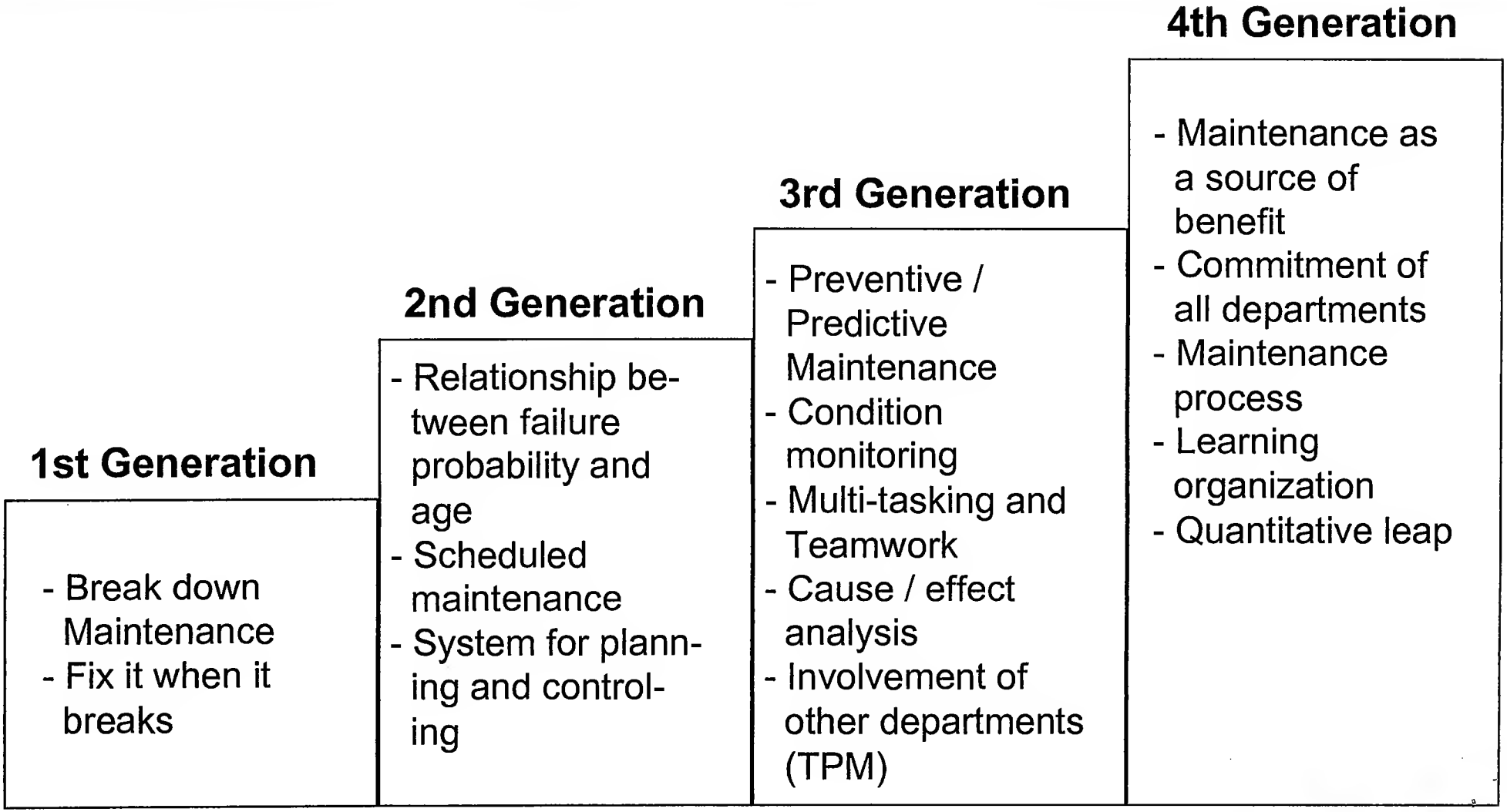
Breaks: 9:35 and 15:05

Definition of Maintenance

- To maintain and restore the desired status
- To define and assess the actual status

..... of technical assets (Machines, Equipment & Components, Buildings, Spare Parts)

Generations of maintenance



Strategy for Quantitative Leap

Profitability

3rd Generation

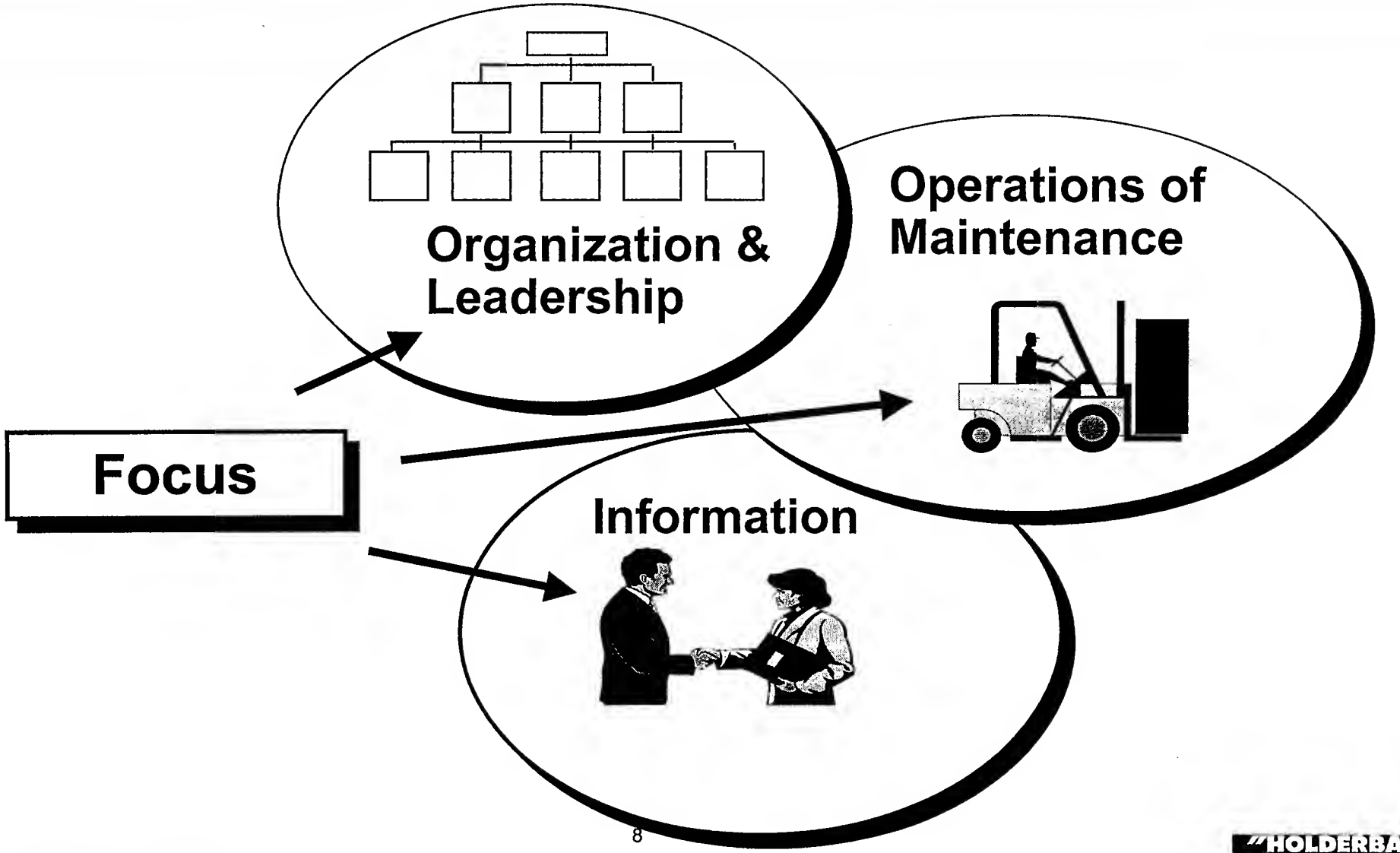
- Preventive maintenance
- Condition monitoring
- Multi-tasking and Team-work
- Cause/Effect analysis
- Involvement of other departments (TPM)

4th Generation

- Maintenance as a source of benefit
- Commitment of all departments
- Maintenance process
- Learning organization
- Quantitative leap

Time

Focus on Key Areas



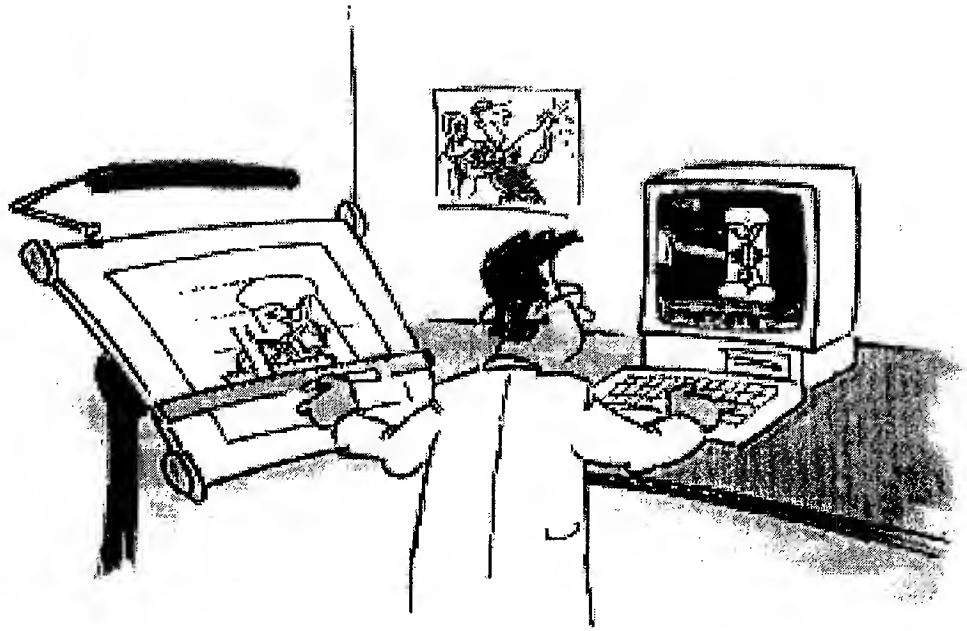
Focus on Key Areas

Overall Equipment Efficiency

The world of motor car racing clearly

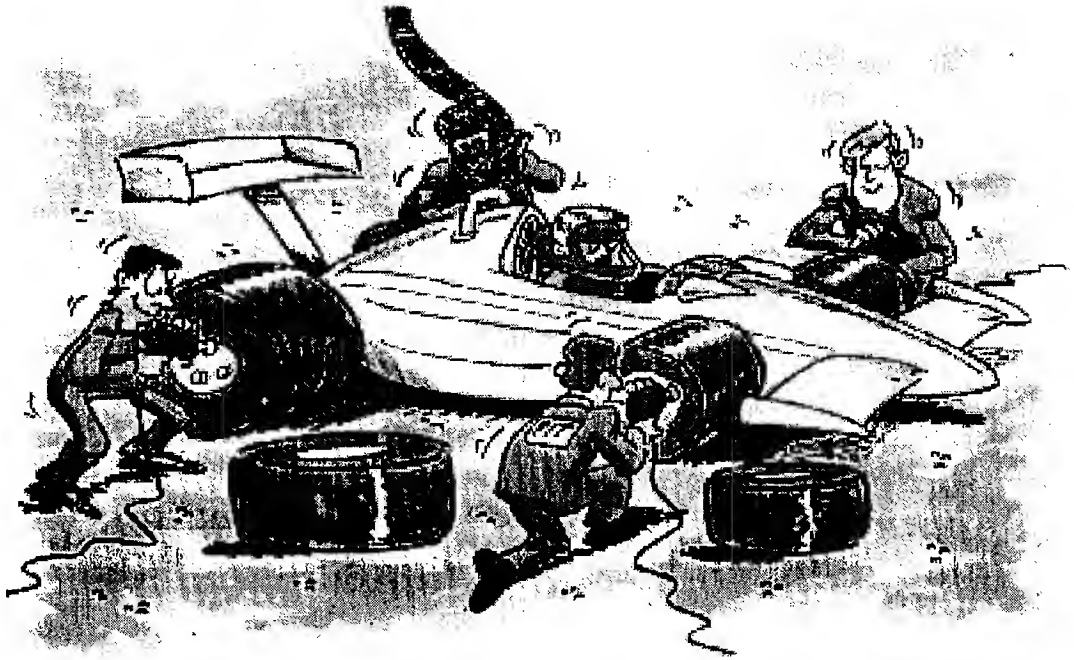
illustrates the concept

Design engineers have the skill to design the car ...



The driver competes on the track ...

And the pit team prepare and maintain the car .....

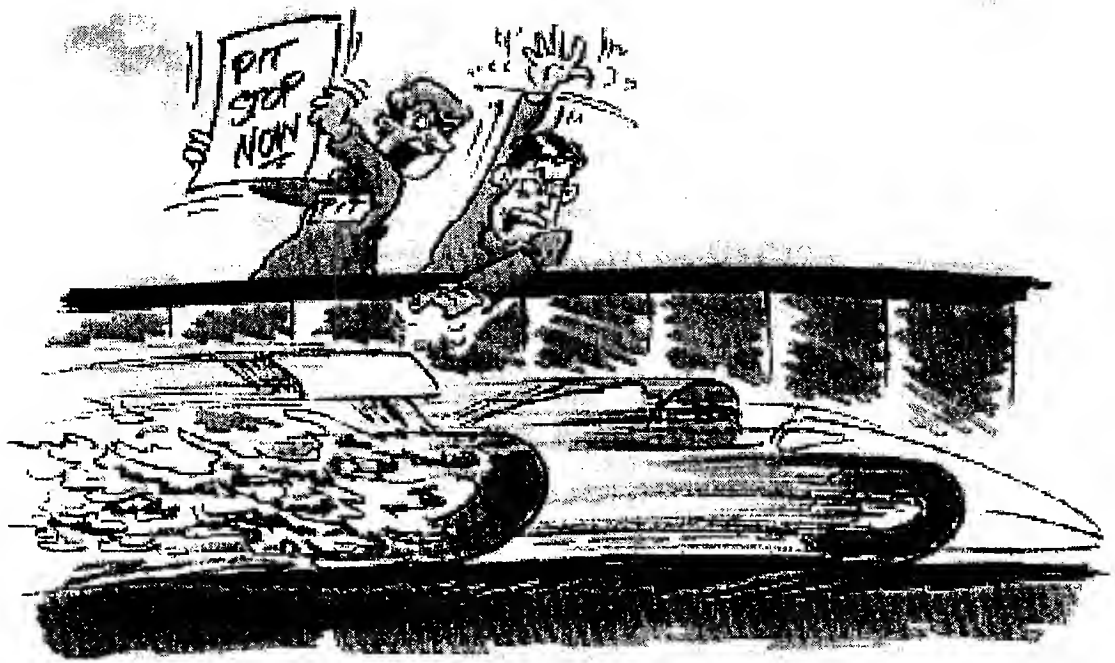


The team cannot be successful unless all co-operate and are involved in attaining the goal -

Imagine the chances of winning if this

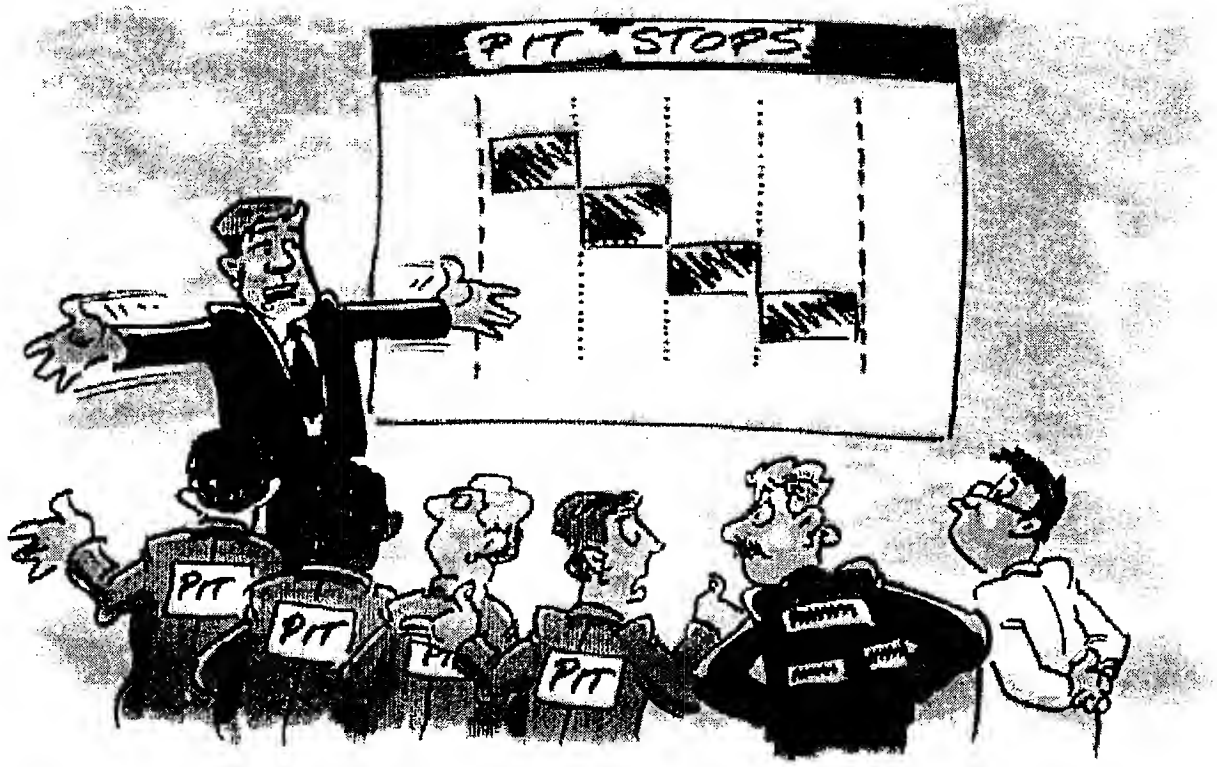
co-operation didn't take place .... if Design ignored input ...

The driver ignored the pit ....



And the pit team only focused on their needs ....

Successful and winning teams work <u>together</u>

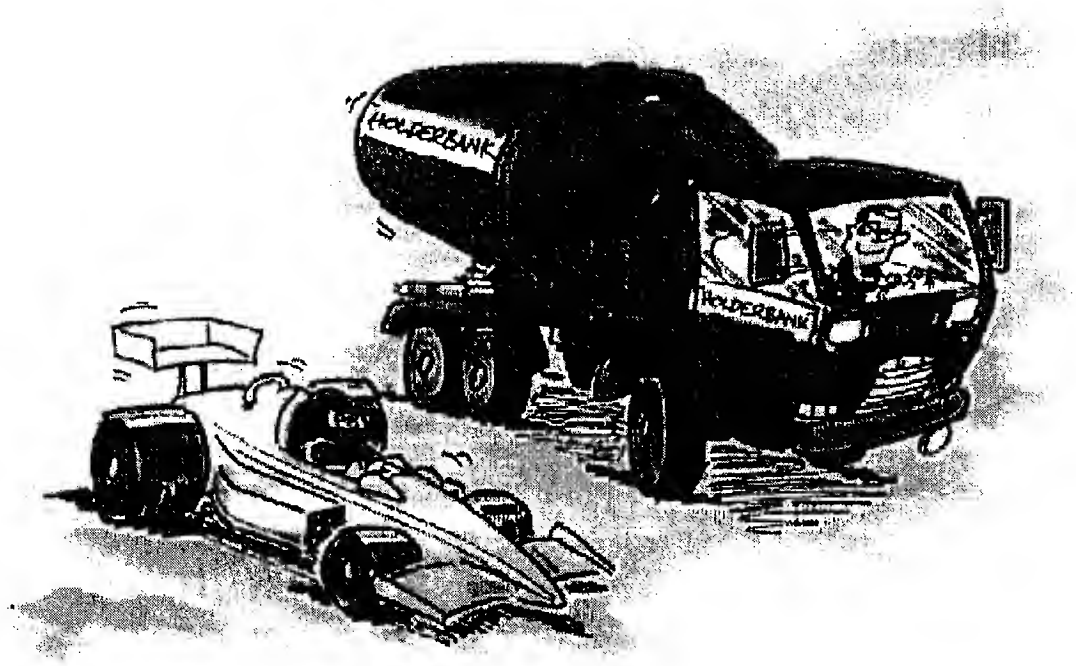


Both the driver and the pit team have input into design -

As the car is developed there are constant reviews of progress

It is not difficult to draw the parallel

to cement production ...



Motor racing uses indicators to measure performance ....

It does not require complex

measures of performance ...

A common unit of measure helps establish where one is and where one wants to be.

O.E.E.

Overall Equipment Efficiency

Overall Equipment Efficiency

O.E.E = Availability x Rate x Quality

How much time is it running	x	How well is it running	x	How much of the output is good product

Availability

Availability is impacted by:

- Planned Major Overhauls
- Major breakdowns
- Short stoppages
  - planned
  - unplanned

Rate

Rate is impacted by:

- Process Control
- Condition of Equipment
- Product Specifications
- Fuel Quality

Quality

Quality is impacted by:

Customer Requirement

↓

Product Specifications

↓

Process Specifications

The Challenge is to quantify the Cost of Non-Quality

Focus of Today

A key success factor of improving Availability:

- Planned Major Overhauls
- Major breakdowns
- Short stoppages
  - planned
  - unplanned

Know the Equipment and its requirements

Cement Seminar 2000

Maintenance

Topics of the Lecture

Mill Components (partially repetition)

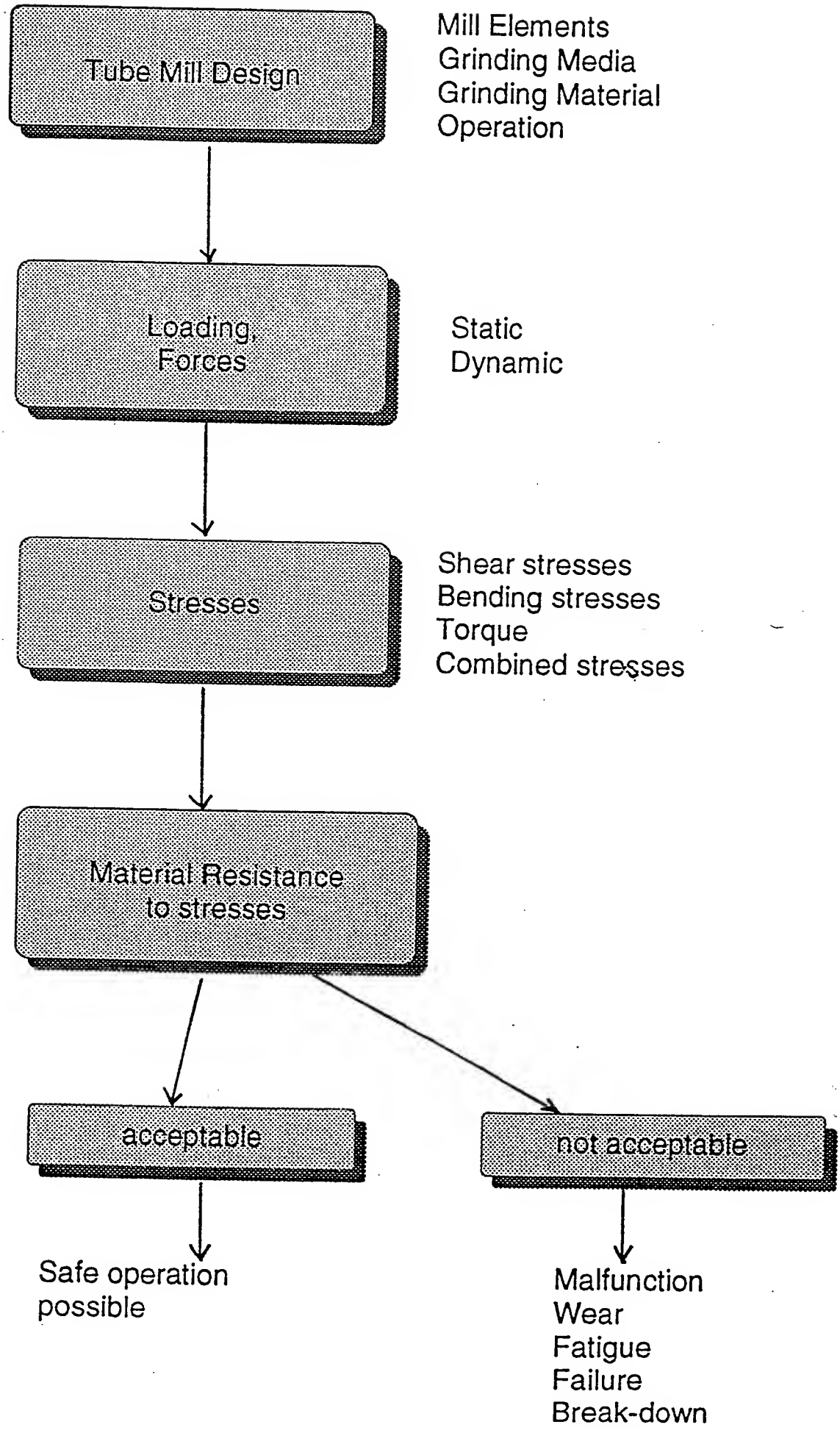
Operating Load, Forces, Stresses

Defects, Damages, Reasons

Case Study: How to prevent / minimize Defects, Damages, Shut Down

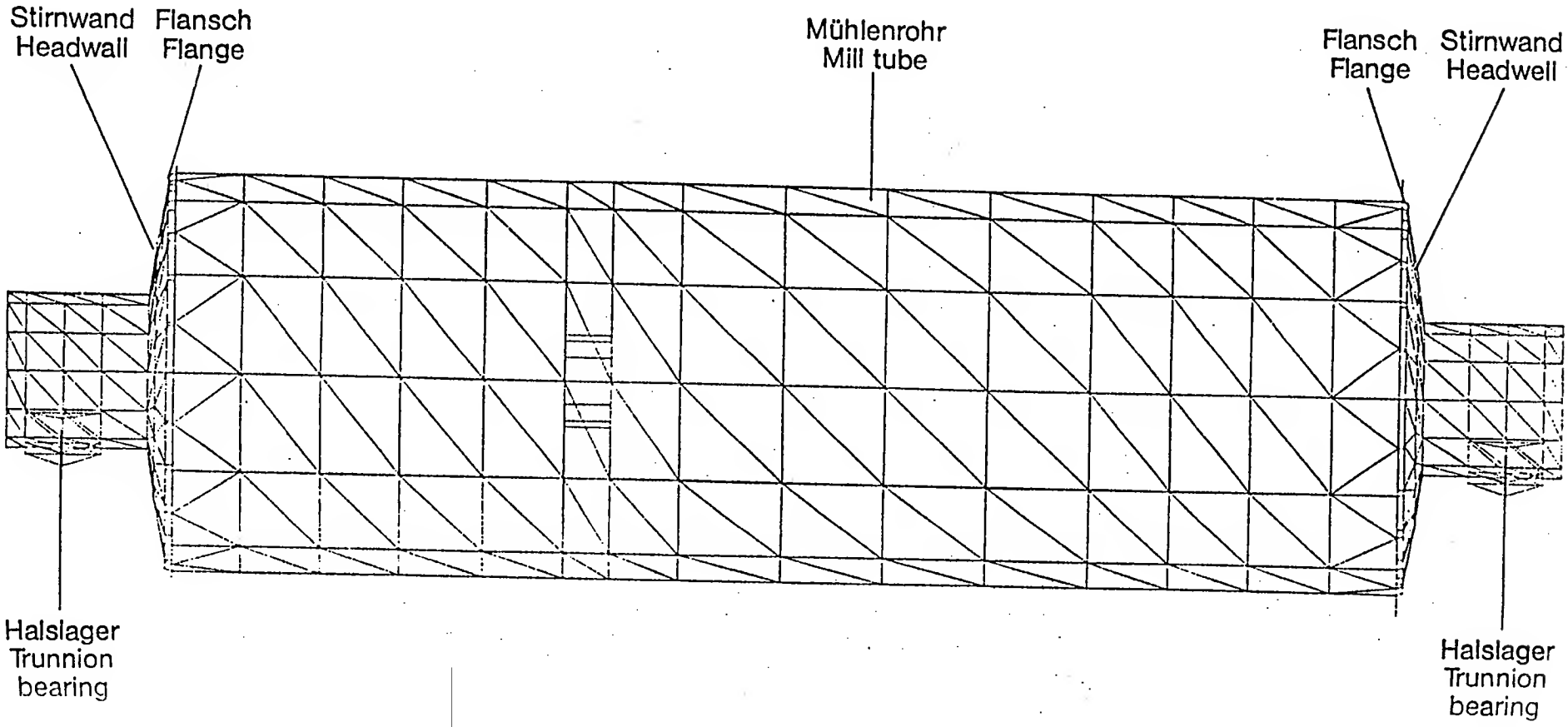
Discussion, Questions

TUBE MILLS


Tube Mill Design
Mill Elements
Grinding Media
Grinding Material
Operation
Loading, Forces
Static
Dynamic
Stresses
Shear stresses
Bending stresses
Torque
Combined stresses
Material Resistance to stresses
acceptable
not acceptable
Safe operation possible
Malfunction
Wear
Fatigue
Failure
Break-down

Rechenmodell einer Zweikammermühle mit Halslagerung
1326 Elemente, 3749 Freiheitsgrade,
die Stirnwandverschraubung ist durch Balken angenähert

Calculation model of a two-chamber mill with trunnion bearings
1326 elements, 3749 degree of freedom
Headwell bolting has been simulated by beans

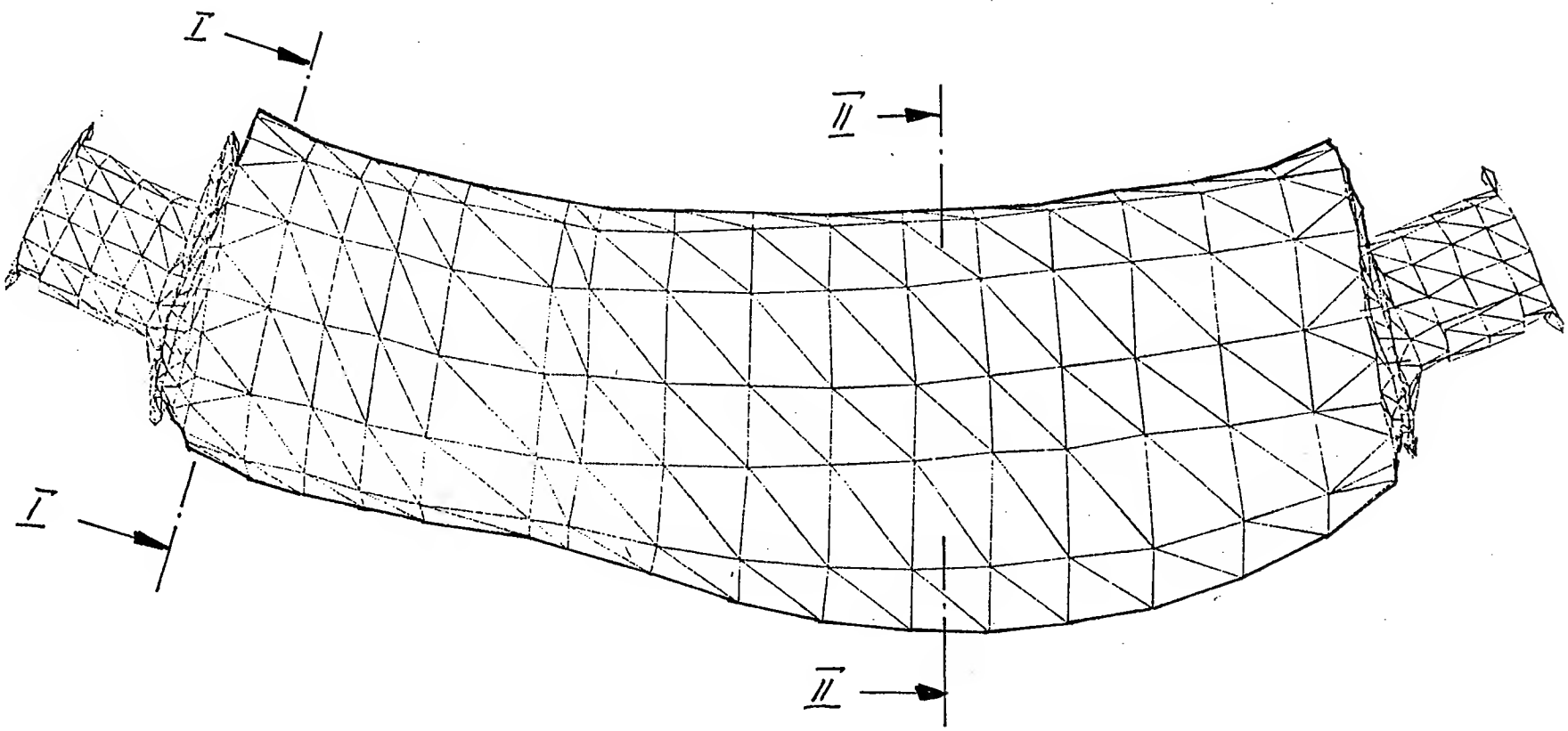


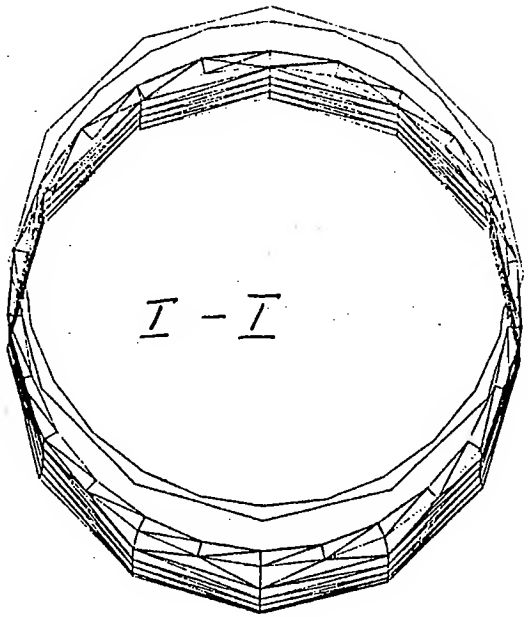
Ermittlung des versteifenden Einflusses der Übertragswand

Zweikammermühle:
Durchmesser 4200 mm, Lagerabstand 15350 mm
Kugelfüllung:
Kammer I = 64 t, Kammer II = 166 t

Ascertaining the strengthening effect of the intermediate diaphragm

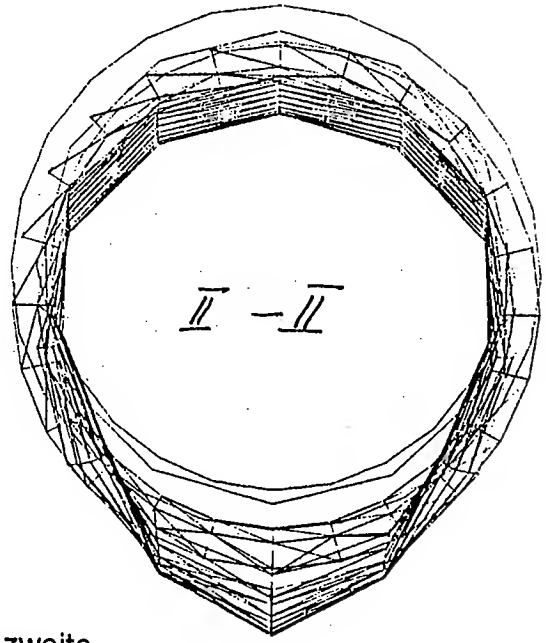
Two-chamber mill:
diameter 4200 mm, bearing distance 15350 mm
Ball filling:
chamber I = 64 t, chamber II = 166 t





Blick in die erste Mahlkammer von der Übertragswand aus betrachtet

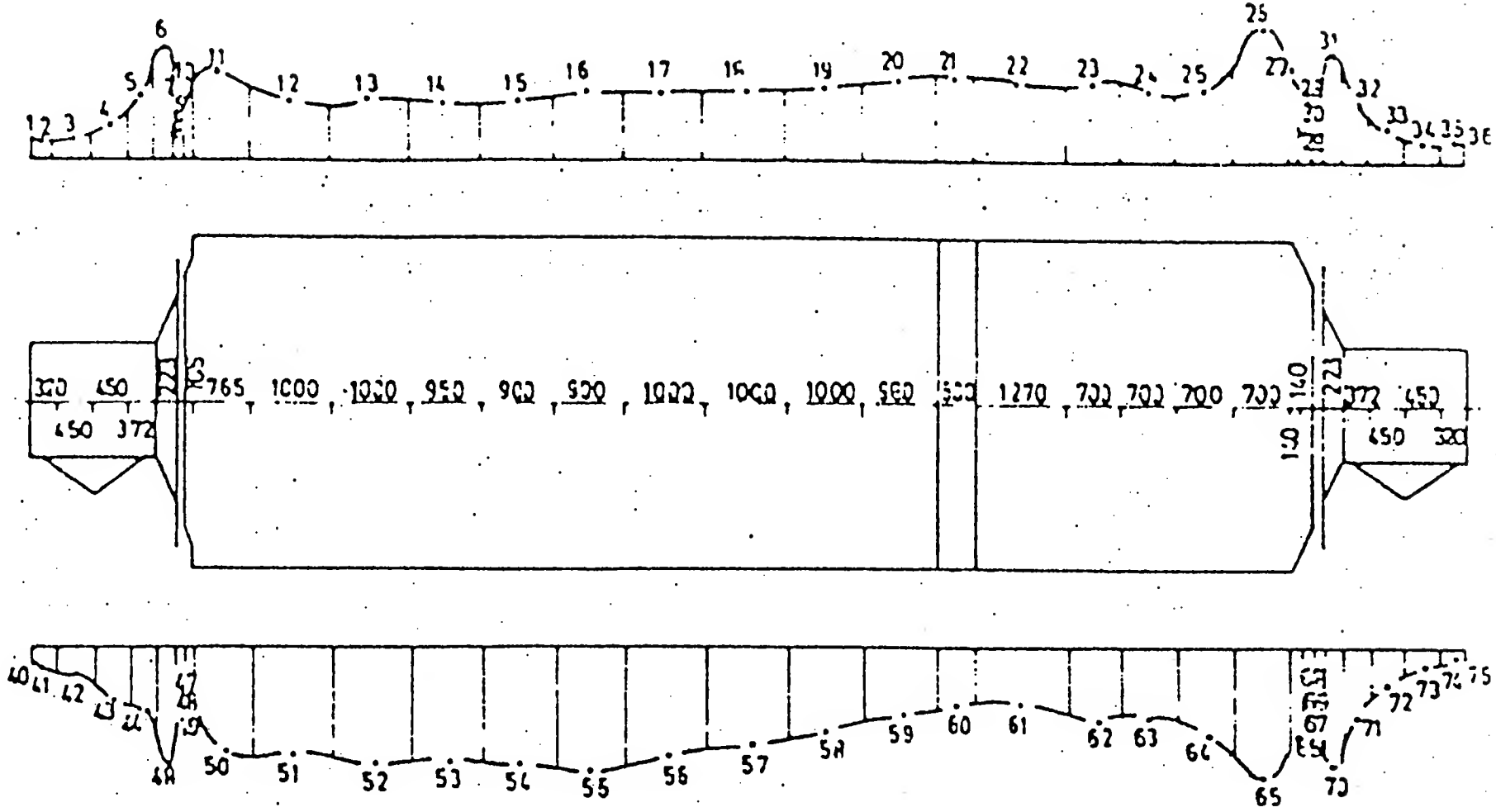
View into the first grinding chamber seen from the intermediate diaphragm



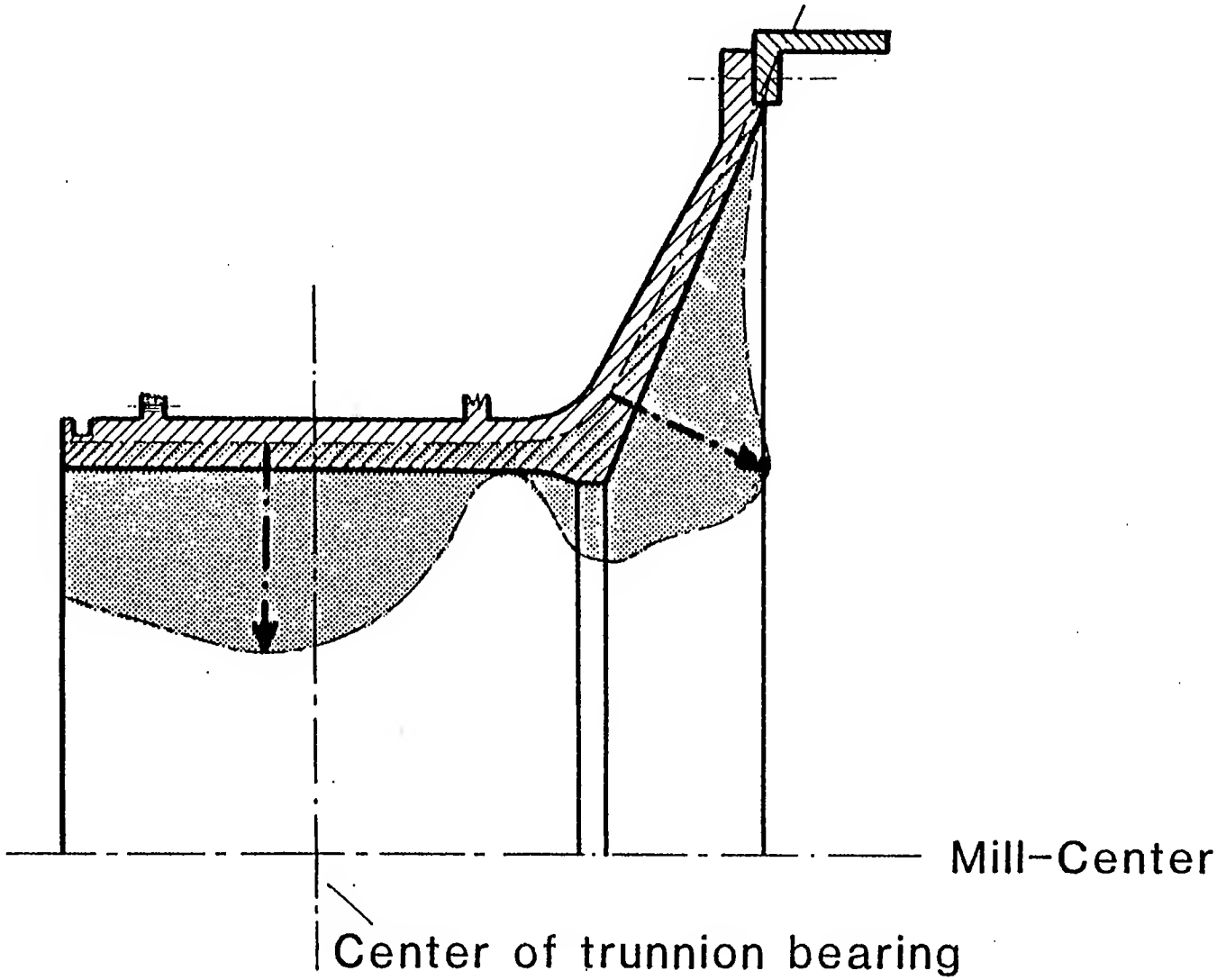
Blick in die zweite Mahlkammer von der Übertragswand aus betrachtet

View into the second grinding chamber seen from the intermediate diaphragm

Ideal stress diagram plotted vs. topside and underside of the mill at varying flange configuration

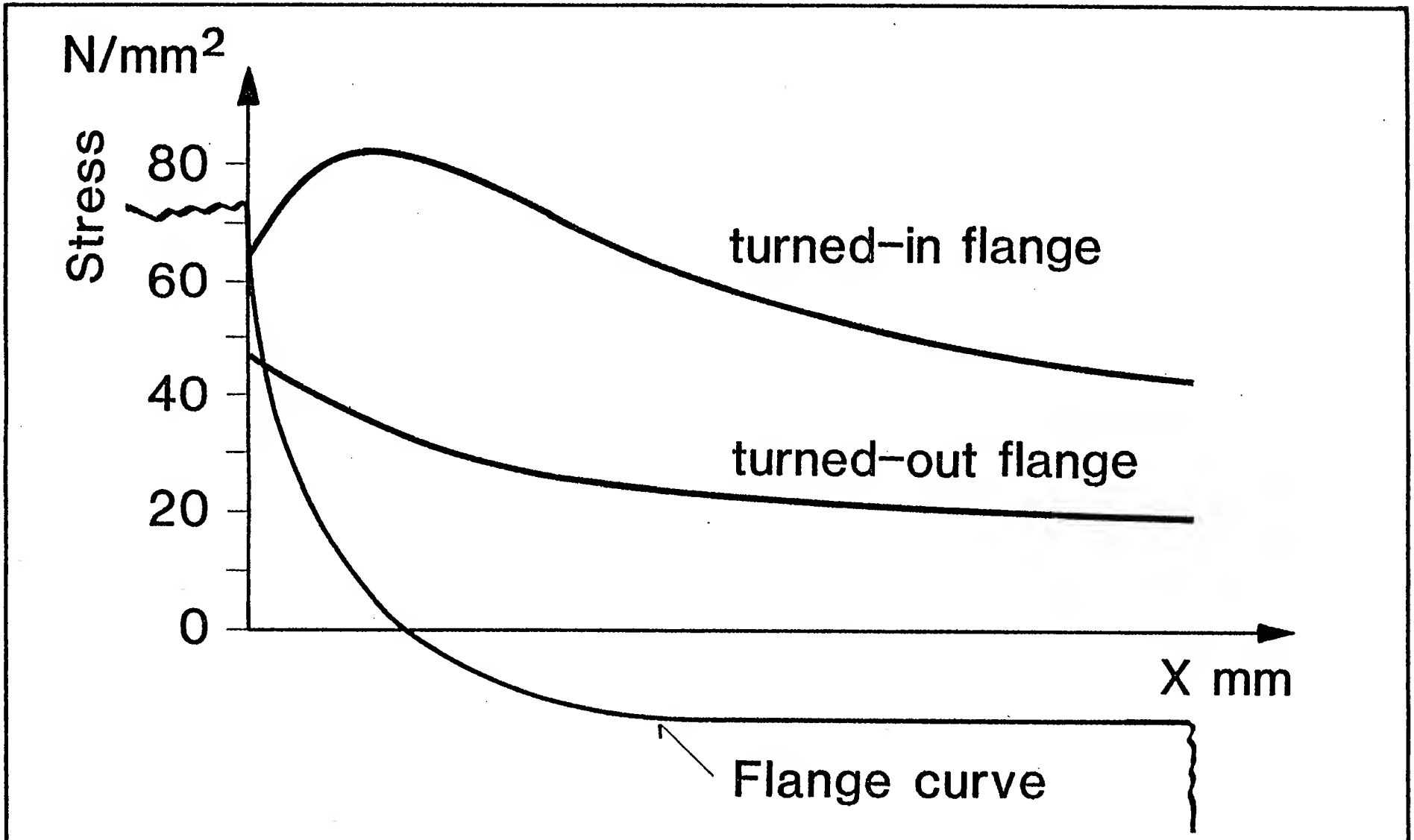


innenliegender Flansch
Flange drawn inside the mill



COMPUTER-CALCULATED STRESS PATTERN OF A MILL END-CASTING

Fig.



	STRESS PATTERN OF TURNED-IN AND TURNED-OUT FLANGES	Fig.

ISO/DIN Gear Rating Concept

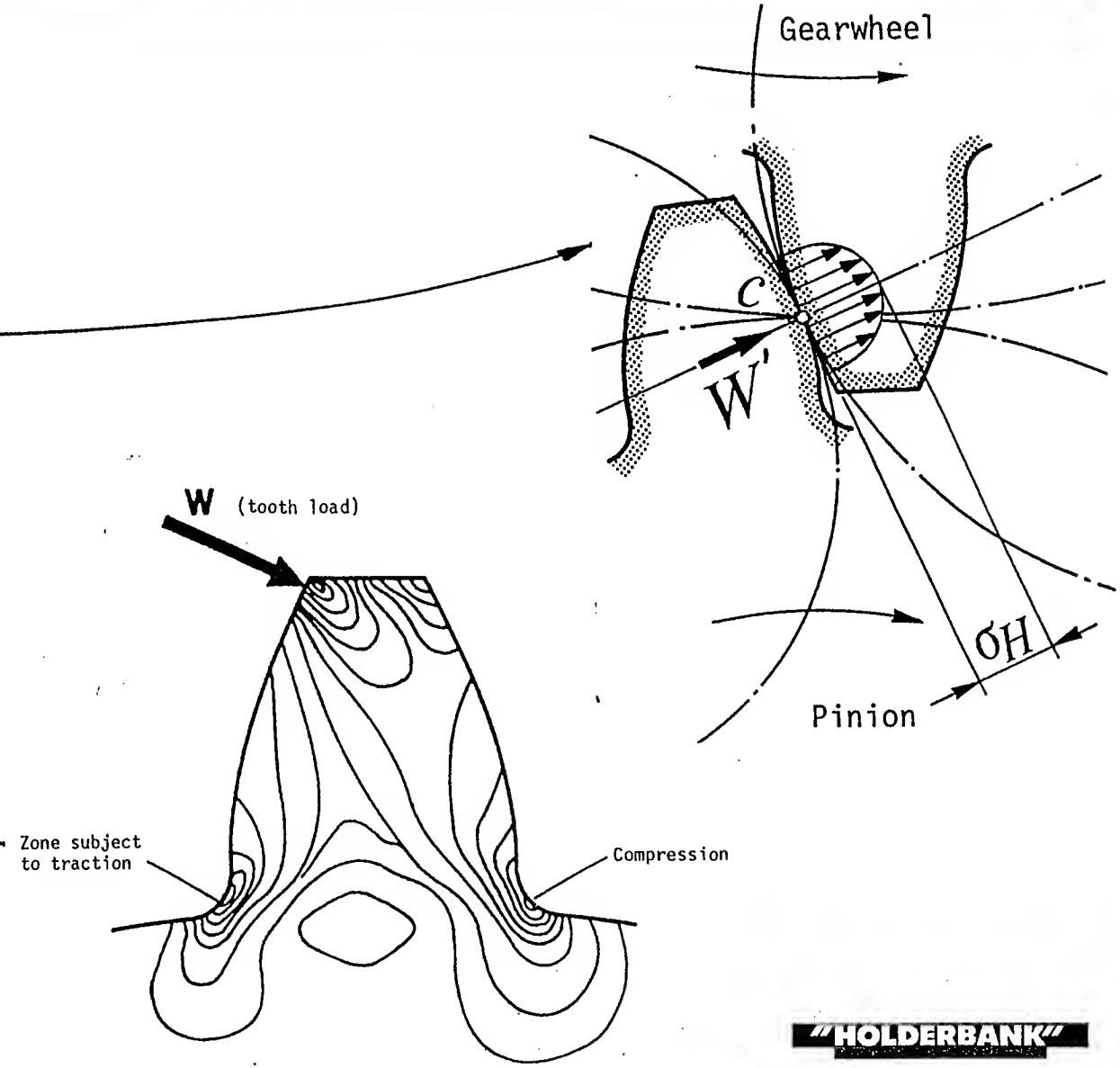
- **Calculation of safety factors with respect to ...**

Flank durability (pitting safety)

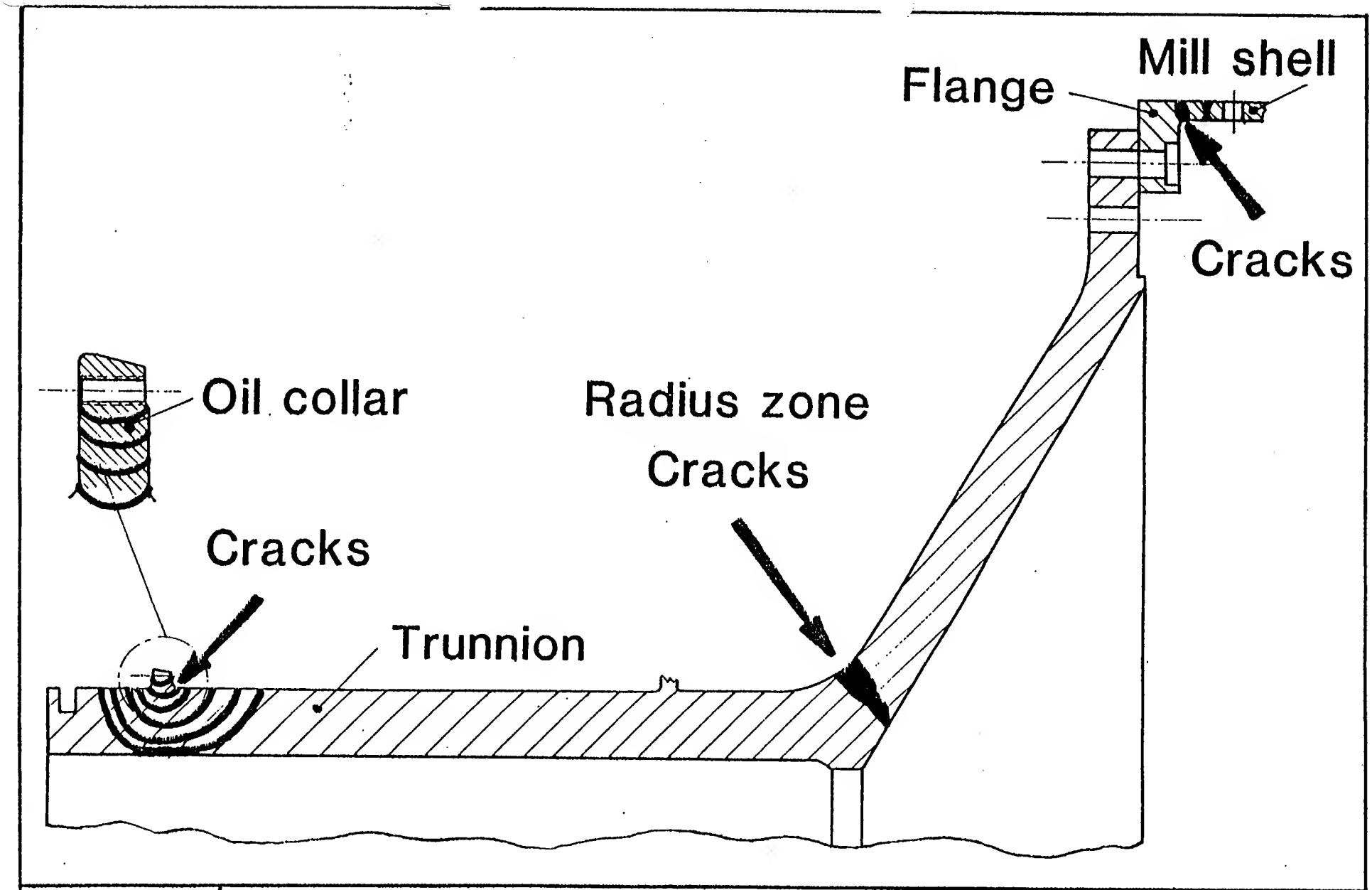
$$S_H = \frac{\text{Fatigue contact stress (MPa)}}{\text{Eff. contact stress (MPa)}}$$

Tooth strength (breakage safety)

$$S_F = \frac{\text{Fatigue root stress (MPa)}}{\text{Eff. root stress (MPa)}}$$

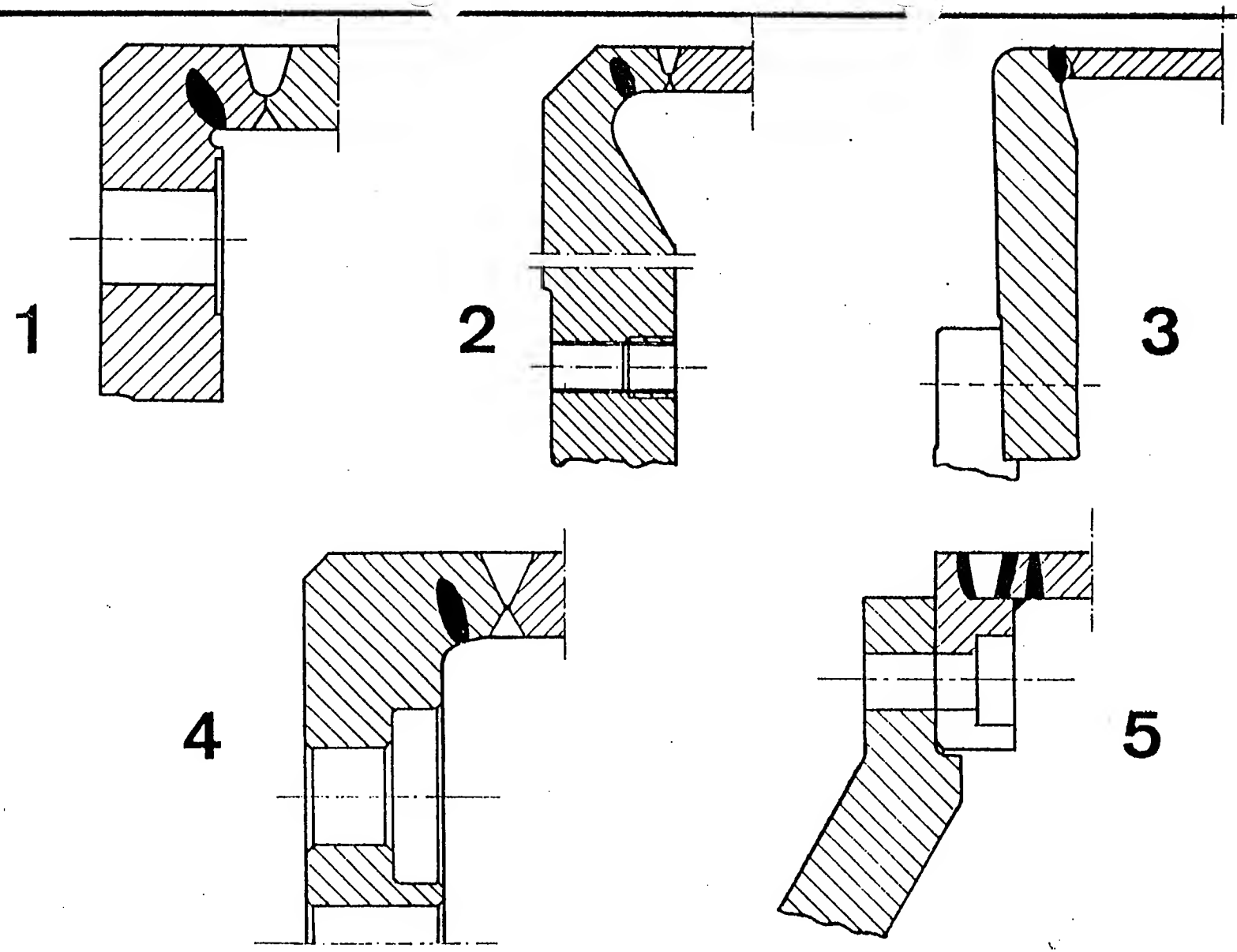


"HOLDERBANK"



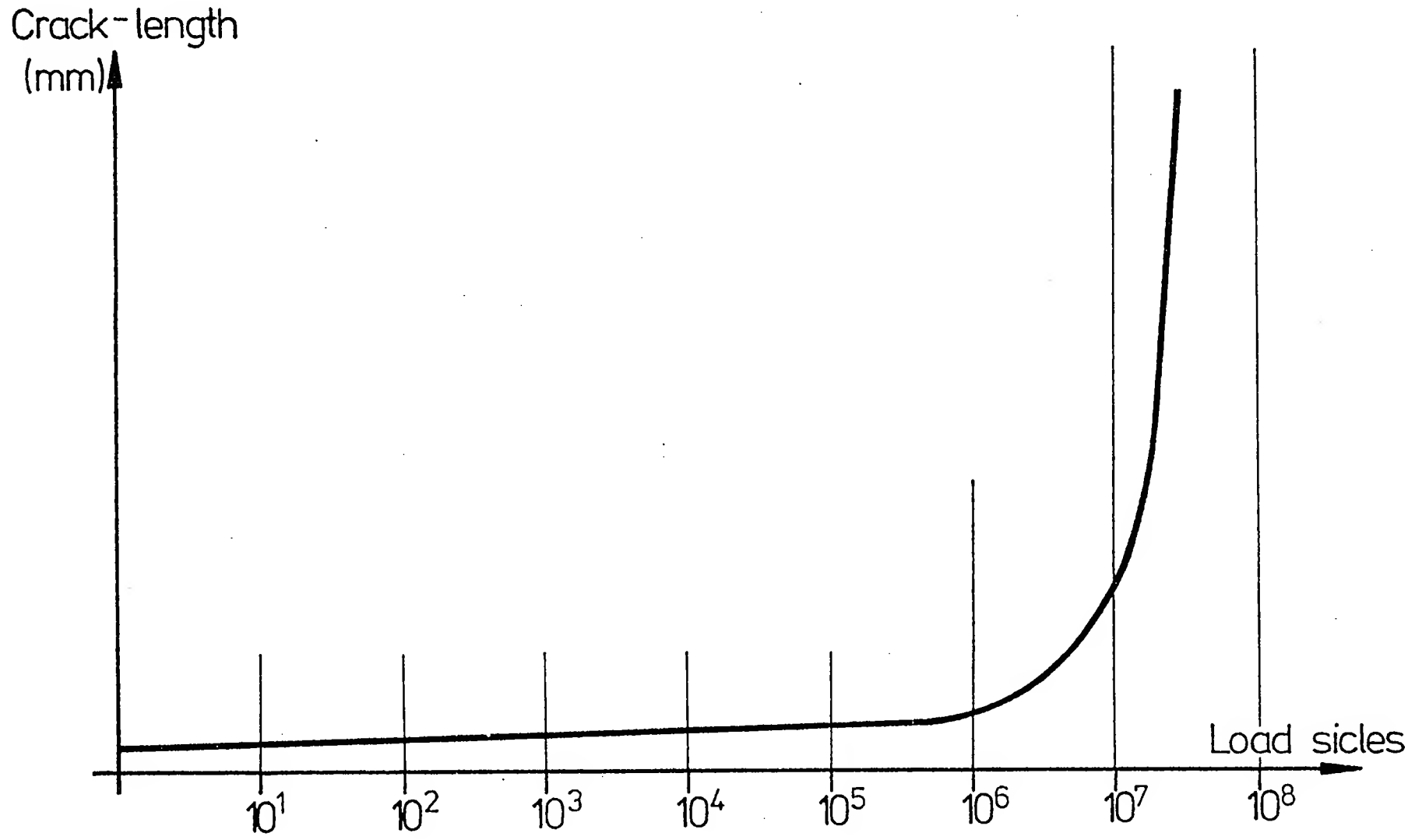
MILL END-CASTING AND FLANGE WITH ENDANGERED ZONES

Fig.



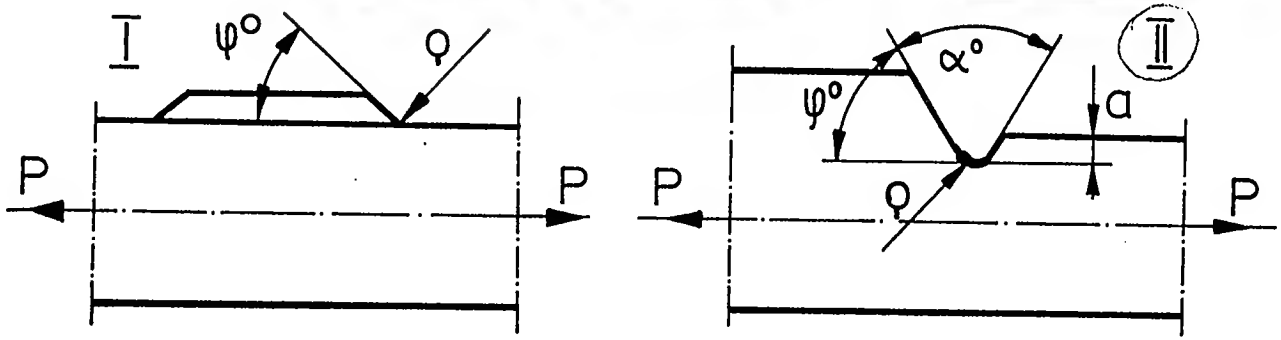
5 DIFFERENT MILL FLANGE JOINTS WITH INCLINATION TO CRACK FORMATION

Fig.

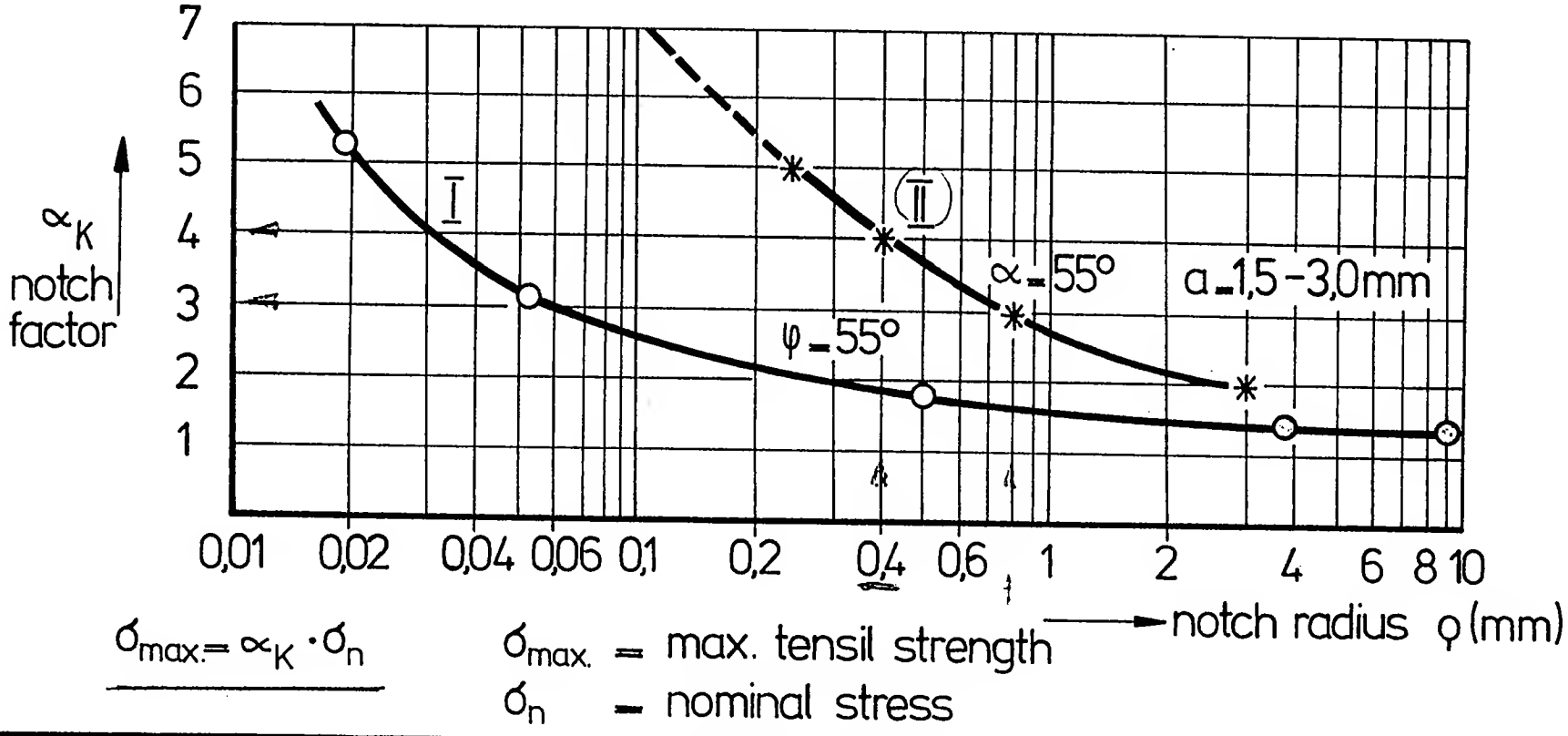


CRACK DEVELOPMENT AS A FUNCTION OF THE NUMBER OF LOAD CYCLES

Fig.



φ° = flank angle
ϱ = notch radius (mm)
a = notch depth
α° = notch angle

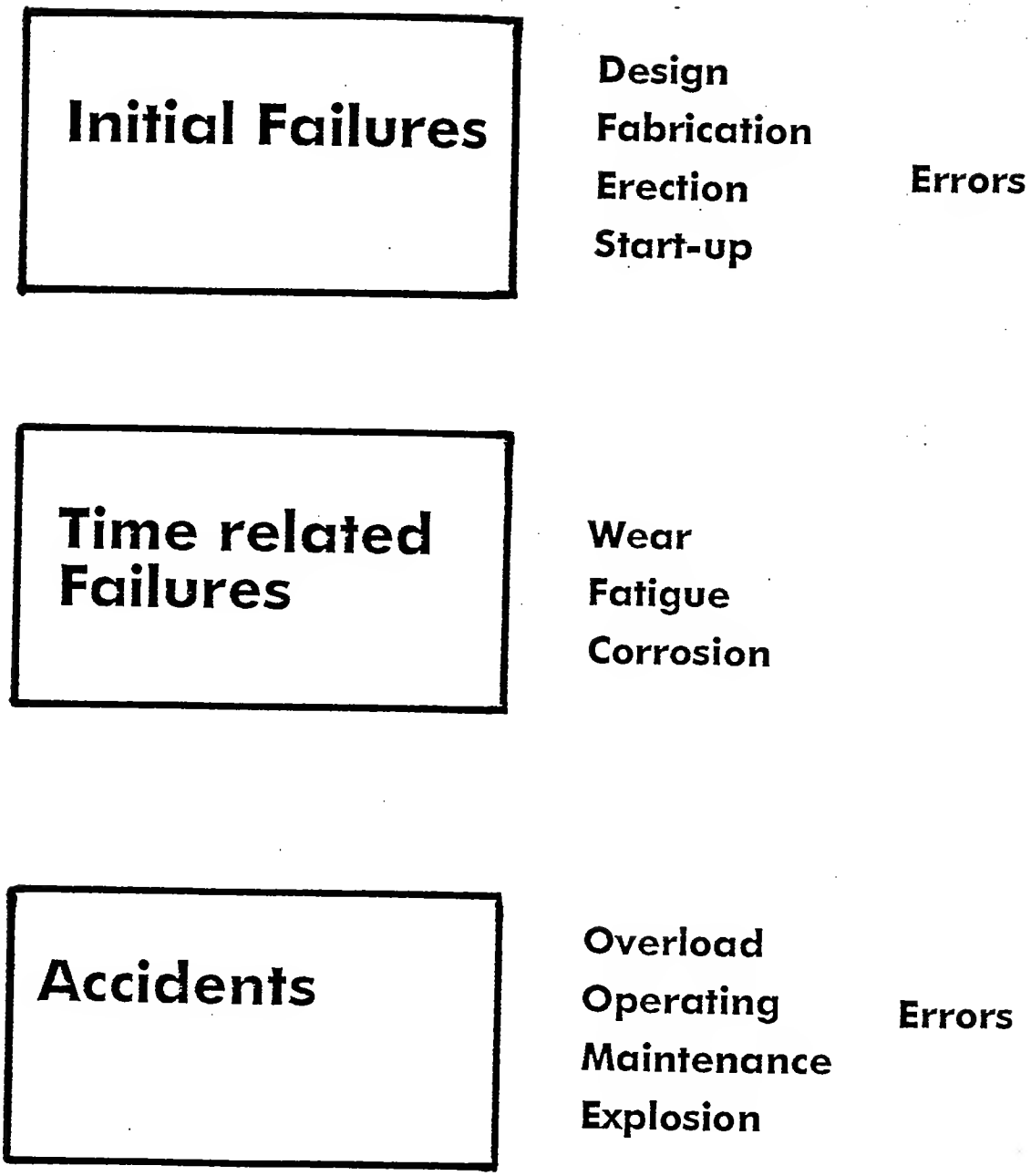


$\sigma_{max.} = \alpha_K \cdot \sigma_n$

$\sigma_{max.}$ = max. tensil strength
σ_n = nominal stress

MAX. TENSILE STRENGTH AS A FUNCTION OF THE NOTCH-RADIUS

Fig.



Failure Categories

Chapter 4

Gas Analysing Systems

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PRACTICAL GAS ANALYSIS IN CEMENT WORKS

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SUMMARY

The measurement of individual process gas components in connection with emissions and for the purpose of process optimisation is increasingly becoming one of the most important procedures in cement works. Measuring techniques have advanced to the extent that there is now virtually no such thing as a problem that cannot be solved technically with the necessary specialised knowledge and means.

In principle, distinction is to be made between analysers that measure (in-situ) directly in the measuring channel (process gas flow) and extractive measuring instruments. Extractive measuring instruments measure in a partial gas flow that is prepared prior to entering the analysers according to specifications (sampling). Extractive measuring instruments are mainly used in process measurement, as the conditions with respect to temperature and dust loading does not permit the use of the in-situ technique.

The point at which measurements are carried out is mainly a question of the measuring problem involved. The extent of sampling increases with increasing process temperature at the sampling point. For gas sampling at the kiln inlet and up to gas temperatures in excess of 500°C, cooling and an emergency cooling or withdrawal system is required. Withdrawal devices with compressed-air drives are more reliable for emergency use, so that preference is given to electric drives.

The separation of dust from the sample gas preferably takes place as close as possible to the entrance of the sampling probe. Sintered metal filters (alloyed steel) clog less quickly than the usual ceramic filters. Compared to the filters outside the sampling probe, internal filters require less maintenance and are smaller in volume. Wet sampling probes are considered out-dated.

Three systems for gas sampling at the kiln inlet are described here in brief: Siemens type FLK, Hartmann & Braun type 60 and type H (Holderbank probe). The latter has proved highly reliable in use worldwide. For gas sampling below 500°C, in addition to standard products, proposals are submitted for the specific construction of sampling probes.

The purpose of sample gas preparation is to supply the analysers with qualitatively and quantitatively specified sample gas. The apparatus used for this purpose must be generously dimensioned and equipped. This requirement applies in particular to the sample gas pump, sample gas cooler and sample gas distribution. The temperature in the measuring cabinet should be between 15° and 25°C. A monitoring system must inform the control system of any irregularities such as "insufficient sample gas", "system being adjusted", etc.

The analyser room must provide the ambient conditions for the perfect functioning of the instruments, i.e. free from vibrations, as close to the sampling point as possible, free from dust and between 15° and 25°C. As it also serves as a workroom for maintenance personnel, it must offer sufficient space, good ventilation and a low noise level as possible.

The analysers used for the determination of individual components in the sample gas are based on different physical effects such as light absorption. Paramagnetism, flame ionisation and solid-state electrolysis. The instrument functions and theoretical relationships will be dealt with only insofar as they are of significance for practical application. When choosing analysers, the manufacturing company is not as important as the specifications and local services offered.

The majority of gas components (CO, NO, SO_2, CO_2) can be measured with NDIR (non-dispersive infrared analysers). They are simple, sturdy, robust, durable and maintenance-friendly. Disturbing cross-sensitivities to CO_2 and water vapour are appropriately corrected.

When measuring SO_2 with NDIR analysers by the classic method, undefinable quantities of SO_2 are absorbed in the sample gas cooler. Corrections are possible by acidifying the

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condensate in the sample gas cooler or using hot NDIR analysers. Preference is given to the use of in-situ analysers.

Oxygen cannot be measured with light-sensitive analysers. The most reliable analysers are still considered to be those that exploit the magnetic properties of oxygen. The life of solid-state electrolytic systems such as zirconium dioxide probes is limited and depends on the gas composition. Improved electrochemical measuring cells have been more widely for the past few years and have a useful life of three years. They are small, lightweight, require a minimum degree of evaluation and are therefore used more and more in extractive multi-component analysers.

The flame ionisation detector is used for measurement of the total hydrocarbons (VOC = Volatile Organic Compounds). The increasing requirements for continuous measurement of the VOC - in connection with alternative fuels - are maintenance-intensive.

Microprocessor technology makes it possible to use well-known and new measuring methods and therefore to measure several components simultaneously. For the installation or replacement of measuring instruments, only the use of multi-component analysers is possible.

The GM 31 in-situ analyser of Messrs Sick enables three gas components to be measured simultaneously. Compared with the earlier GM 21 (SO_2 and dust) and GM 30 (additionally NO) series, the GM 31 does not offer dust measurement.

The AR 620 and AR 650 series of Opsis are modern emission measuring instruments. Depending on the type and number of components to be measured, infrared or ultra-violet light used to the two together. The analyser or analysers are connected to the receiver at the measuring channel via glass-fibre cable (light conductor). Control, monitoring and evaluation is carried out by a standard PC. The analysers have been used several times with good experience.

The new gas analysers with lasers are expected to have a promising future. These are in-situ analysers that offer the benefits of high sensitivity, short response times and high stability. They could provide the ideal preconditions for CO monitoring, but are (still) relatively expensive.

Of the extractive measuring instruments, good experiences have been made with the CEMAS FTIR analyser of Hartmann and Braun. The analyser can be programmed with all IR selectable components. In order not to alter the originality of the sample gas, all gas conducting parts are heated.

Recording the measured values with strip chart recorders should still suffice for a general overview. However, in the majority of countries, digital evaluation of emissions is required according to specific criteria. In the field of process optimisation, the measured values are also only useful when available in the required form, e.g. as mean values, spreadsheet for further evaluation, etc. At the kiln inlet, averaging or damping for lump fuel (tyres) feeding in precalcining is absolutely necessary. In modern plants, the measured values can be partly or wholly processed by the process control system.

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The reliable functioning of each measuring instrument depends largely on the maintenance provided. Visual checks and function tests must be carried out according to the age of the measuring instruments and the specifications of the supplier.

Personnel must be suitably trained. Critical spare parts such as the sample gas pump, filters valves, seals, etc., must be carried in stock.

The gas analysers must be checked according to the manufacturer's instructions and emission measurements carried out in compliance with the local requirements and corrections made as necessary. If test gas should (still) be required, this must be obtained in good time. Any fault signalling devices must be checked for proper functioning.

Theoretically, maintenance can be reduced by appropriate automation. Adjustment via internal analyser reference devices has proved highly disadvantageous in this respect. However, if solenoid valves should be required for the application of test gas, their usefulness is negligible or negative. Solenoid valves in sample gas are highly susceptible to faults. For this reason, sample gas changeover from two or more measuring points to the same analyser group is not considered worthwhile.

The accuracy of gas analysis as a whole is often overestimated, whereby the analysers are least to blame in this respect. Significant errors occur when sampling (representative) due to sorptions and reactions in sample gas preparation.

Gas analysis also has special monitoring functions for protection against explosions in the electrostatic filter and coal pulverising mills. Simple monitoring of the CO content is usually not sufficient for the protection of the electrostatic filter. The consequences are unnecessary shutdowns of the filter. In addition, reference is made to a new analyser of Messrs Sick, which has been designed specially for this task.

Finally, a table is provided showing the various conversion factors from ppm to mg/m^3 for the relevant gas components.

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1. INTRODUCTION

Gas analysis is among the most important, but also the most sophisticated methods of measurement in cement works. This is due to the constantly growing demands in the field of gaseous emissions and related requirements. Gas analysis fulfils an equally important function in the field of automation, where the reliable and exact measurement of certain gas components plays an essential role.

Advances in the field of gas analysis have taken place parallel to the trend of demands, so that one can hardly speak of "technically unsolvable problems" any longer. If problems should nevertheless occur, they are mainly due to the following

- Incorrect choice of equipment
- Lack of specialised knowledge
- Underestimation of expenditure involved (investment and/or maintenance expenditure)
- The aim of this report is to prevent problems of this kind occurring when renewing and adding gas analysing apparatus.

2. PURPOSE

The continuous measurement of the constituents of certain components in the gas flows (gas analysis) in cement works consists of three areas of activity:

a) Emission measurements (pollutants, e.g. NO, SO_2, CO, etc.).

b) Safety measuring systems (e.g. CO monitoring in filtering plants, monitoring of coal dust silos, etc.).

c) Process measuring systems (e.g. O_2, CO, NO) for optimisation of the combustion process.

Important for all these activities is a high availability of the measuring equipment and sufficient accuracy of the measured values. The report therefore attaches particular importance to sample gas sampling and preparation, maintenance and quality assurance as well as possible measuring errors. Finally, hints and practical advice are given, based on many years of experience.

3. DEFINITIONS

Indication delay (90% time) of an analyser or a measuring instrument: The time that elapses from a sudden change in the gas concentration at the entrance of the analyser or probe up to indication of 90% of this change.

Downtime: The time during which faulty conditions of the measuring instrument and their correction unforeseeably make the generation of measuring or adjusting signals impossible.

Accompanying gas: Sample gas without the gas components to be measured.

Extractive sampling: The sample gas is extracted as a partial gas flow from the process gas prior to analysis and conditioned.

Gradient monitoring: Monitoring the gas concentration change gradient (e.g. control of CO cut-off).

In-situ: The measuring instrument is located in or on the process gas duct; analysis is carried out in the process gas (e.g. optical dust content measuring instrument or zirconium oxygen probe).

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Adjustment: Setting or adjustment of the gas analyser with test or zero gas with the aim of adjusting the indication of the gas analyser as close as possible to the gas concentration to be measured in the sample gas. With newer analysers, this can also take place with built-in gas-filled glass cuvettes.

Calibration: For a given measuring instrument, determination of a valid relationship between the indicated measured value or the value of the output signal and a reference value that comes closest to the true measured quantity. Determination of the scale divisions of a measuring instrument with the aid of a reference measuring method (generally required only for officially specified continuous emission measurements).

Life zero: A specific quantity (normally mA) prevails in the electric output signal when the analyser physically measures "zero". The advantage of this system is that the output signal is electrically set to "zero" only if the electrical measuring circuit fails.

Sample gas: That part of the process gas extracted from the process gas and passed to the analyser for measurement.

Sample gas preparation: The entire equipment used for the qualitative and quantitative preparation of the sample gas to ensure that the same conforms to the specifications of the analysers.

Measuring equipment: All devices and instruments required for the measurement of gas component concentrations.

Measured quantity: Physical quantity, the numerical value of which is to be determined by measurement.

Measured component: The gas component in sample gas, the concentration of which is to be measured.

Zero gas: Test gas used for the adjustment or readjustment of the zero of a gas analyser.

On-Line: The measured quantity is coupled directly with the process.

Test gas: Gas with a known composition for adjustment or readjustment of gas analysers. It consists of the measured component and one or several accompanying gases (e.g. nitrogen/ carbonic acid + measuring component).

Cross-sensitivity: Influence of the concentration of a disturbing component on the measuring result of another gas component under identical measuring conditions.

Representativity: The correspondence of the measured signal with the definition of the measured quantity, e.g. the composition of the gas at the kiln inlet measured at a point in relation to the average gas composition over the entire cross-section.

Disturbing component: Component in the sample gas that falsifies the measured value, e.g. if the gas analyser has a cross-sensitivity or chemical reactions distort measurements.

Dead time of an analyser or measuring instrument: The time that elapses from a sudden change in the gas concentration at the entrance of the analyser or probe up to the start of the indication of this change.

Availability: The time during which the measuring instrument generates usable signals.

Supply pressure: The sample gas pressure necessary for supplying the analysers.

VOC Volatile Organic Compounds: Volatile organic hydrocarbons.

Maintenance time: Time required for maintenance of the measuring instrument (planned preventative maintenance).

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4. MEASURING METHODS

4.1 In-situ measurement

In-situ measuring methods mainly offer the following advantages:

- No gas sampling
- Sampling forms a path in the measuring channel (as opposed to a point in extractive methods)
- Undelayed indication
- Less maintenance
- Good long-term stability

In-situ measuring methods are used increasingly in modern measuring techniques.

Two different measuring principles are in the foreground:

- Solid-state electrolytic and
- Optical systems

Stabilised zirconium dioxide is a typical solid-state electrolyte with whose properties oxygen can be measured directly in the process (similar to thermocouples) (see chapter 8.5.1). This measuring method has proved reliable in the cement industry. In individual cases, the useful life of the zirconium sensors was unsatisfactory, as their activity was destroyed by certain accompanying gases.

Optical instruments are flanged directly on to the gas channel to be measured (e.g. chimney). These consist of a transmitter (normally light) and a receiver or reflector arranged opposite. While passing through the channel, the light beam emitted from the transmitter (infrared, ultraviolet or laser) is changed by certain gas components. After appropriate processing, this change results in the measured quantity (see chapter 8.6).

Optical measuring instruments (8.6.1, 8.6.2, 8.6.3) are used mainly for the measurement of pollutant emissions, e.g. dust content, SO_2, NO, etc. The range of instruments available for measuring further gas components is increasing in line with technological advances. The application of optical measuring in-situ instruments is restricted by two factors; the dust content and the given length of the measuring path, which is in direct relation to the sensitivity. For use in the cement works, these restrictions mean that they only function in pure gas and their application is limited solely to the measurement of emissions, although good experience has been made in this respect. Thanks to improved representativity, in-situ measuring methods are technically superior to extractive methods.

Investments in-situ instruments are generally higher compared with extractive measuring instruments. As will be shown later, additional investments in in-situ measuring methods are justified under certain conditions. The level of maintenance is normally less, but partly more sophisticated than extractive measuring instruments, from a technical point of view.

The measured values of in-situ instruments are always based on gas in an operational condition. However, values in a standard condition are usually required, .i.e. at 0°C, 1013mbar, dry. For conversion to the standard condition, the parameters pressure, temperature and moisture must be known or additionally measured.

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4.2 Extractive measurement

The measuring procedure is subdivided into different stages:

- Sample gas sampling
- Sample gas supply
- Sample gas preparation
- Analysers

With the exception of the dust content, the majority of components can be measured by the extractive method. In-situ measuring instruments offer the following advantages:

- Several analysers can be connected to the same sample gas preparation apparatus.
- The positioning of dust and heat-sensitive analysers is more flexible.
- Adjustment and calibration is simpler (with built-in reference cuvettes or test gas).

Drying the gases during gas preparation by means of cooling to a dew point of about 3°C can be disturbing for certain components. These are logically water vapour (H_2O) sulphur dioxide (SO_2) and certain hydrocarbon compounds (VOC) (see chapter 8.4).

To exclude disturbing influences during sample gas drying, there are instruments with operating temperatures that are far above the dew point (up to 250°C). The components of these instruments are subject to increased load when exposed to these temperatures, which inevitably leads to an increased susceptibility to faults. Experiences made with an instrument of this type are described in chapter 8.6.4.

The multiple advantages of extractive measuring instruments, compared with in-situ measuring methods, are confronted with a slightly higher level of maintenance, although the maintenance necessary for the extractive measuring method is less demanding than that required for in-situ measuring instruments.

4.3 Measuring points

Figures 1 to 3 show the typical measuring points at various kiln systems (cyclone preheaters, grate preheaters and wet kilns). The associated tables show the usually measured components as well as the purpose of their measurement. The choice of measuring point at the kiln system at which measurements are to be made, depends primarily on the purpose of measurement.

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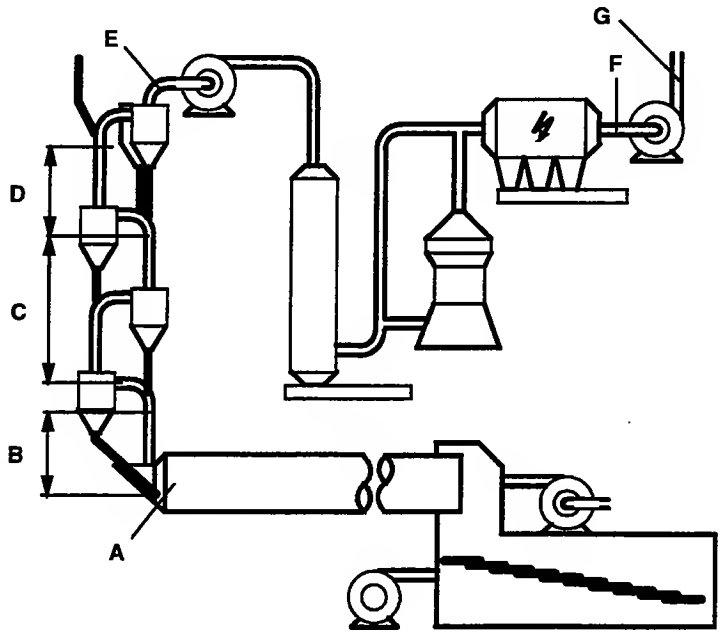
Figure 1: Measuring Points at Cyclone Preheater Kilns



Purpose	Components				A	B	C	D	E	F	G	Remarks
Process Control	O2	CO	NO	CO2	●	◑	◑	⊘	⊘	○	○	Without Precalcination
Process Control	O2	CO	NO	CO2	●	◑	○	●	●	⊘	○	With Precalcination
CO-Monitoring	O2	CO	NO		○	◕	◑	●	◑	○	○	Without Precalcination
CO-Monitoring	O2	CO	NO		○	○	◕	●	●	○	○	With Precalcination
Emissions	O2	CO	NO		○	○	◕	◕	◕	◑	●	Without Precalcination
Emissions	O2	CO	NO	CO2	○	○	○	◕	◕	◑	●	With Precalcination
Emissions	SO2	NO	HC	etc	○	○	○	○	○	◑	●	All Kiln Systems

Suitability:

- ● Good
- ◑ Satisfactory
- ◕ Bad
- ⊘ Standby (Reserve)
- ○ Unsuitable

Measuring Points:

- A Kiln Inlet (Probe in rotating Part)
- B Inlet Chamber, Riser Duct (Probe not in rotating Part)
- C Lower Cyclon stages (Calcinator)
- D Upper Cyclon stages
- E After Cyclon preheater
- F After the Filter before Fan
- G Stack (After Filter and Fan)

Table 1: Cyclon-preheater Kilns

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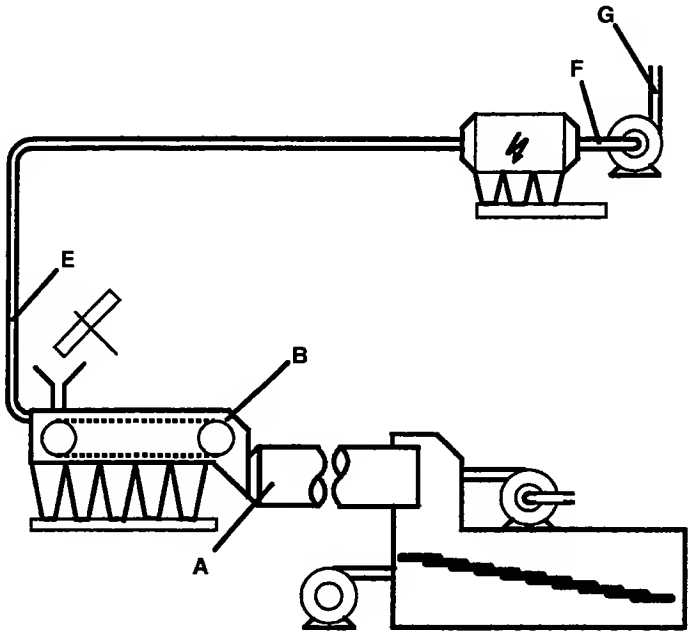
Figure 2: Measuring points at the grate preheater kilns



Purpose	Components				A	B	C	D	E	F	G	Remarks
Process control	O2	CO	NO	CO2	●	◑			◑	○	○	
CO-monitoring	O2	CO	NO		◕	◑			●	○	○	
CO-monitoring	O2	CO	NO		○	○			●	○	○	
Emissions	O2	CO	NO		◕	◕			◕	◑	●	
Emissions	O2	CO	NO	CO2	○	○			◕	◑	●	
Emissions	SO2	NO	HC	etc	○	○			○	◑	●	

Suitability:
● Good
◑ Satisfactory
◕ Bad
○ Unsuitable

Measuring points:
A Kiln inlet (probe in rotating part)
B Kiln inlet (probe not in rotating part)
E After the grate preheater
F After the filter, before the fan
G Stack, after the filter and the fan

Table 2 Grate preheater kilns

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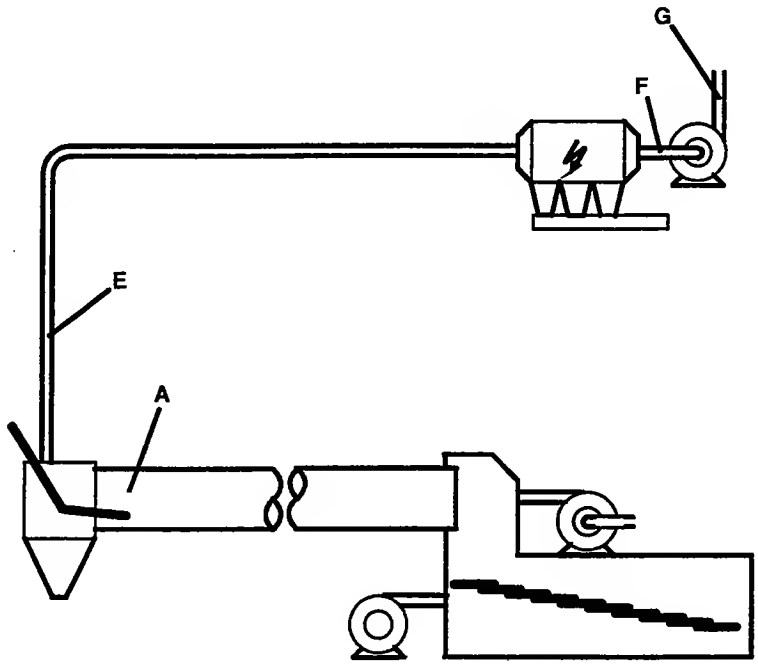
Figure 3: Measuring points at wet kilns and long dry kilns



Purpose	Components				A	B	C	D	E	F	G	Remarks
Process optimization	O2	CO	NO	CO2	●				◑	○	○	
CO-monitoring	O2	CO	NO		●				◑	○	○	
Emissions	O2	CO	NO	CO2	◕				◑	◑	●	
Emissions	SO2	NO	HC	etc	◕				◑	◑	●	

Suitability:

- ● Good
- ◑ Satisfactory
- ◕ Bad
- ○ Unsuitable

Measuring points:

- **A** Kiln inlet (probe in rotating part)
- **E** before the filter
- **F** After the filter, before the fan
- **G** Stack, after the filter and the fan

Table 3 Wet and long dry kilns

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5. GAS SAMPLING

5.1 Kiln inlet (measuring point A)

At the rotary kiln itself, only the measuring point at the feed end of the kiln is realisable with reasonable effort. The components normally measured there are needed as reference quantities for an optimal combustion process (usually O_2, CO, NO).

To prevent distortion of the measured signals due to false air from the kiln inlet seal, the probe tip must be located distinctly within the rotating part. The sampling conditions at the kiln inlet vary considerably, depending on the type of kiln (preheater kilns, long dry kilns and wet kilns).

5.1.1 Preheater kilns (cyclone preheater and grate preheater kilns)

Process conditions at the sampling point	
Gas temperature	1000 to 1300°C
Dust content	>100g/m^3 (cyclone preheater)
Dust content	>5/g/m^3 (grate preheater)
Dew point temperature	35 to 40°C

The process gas contains large proportions of alkalis, chlorides and sulphates that are subject to high local and time variations. Depending on the process and raw material situation, the alkali compounds combine with the dust to form strongly adherent deposits. To counteract their effects to make gas sampling possible at all, elaborate measures are necessary from a measuring point of view.

Owing to the high temperatures, the provision of external cooling is necessary for all types of sampling probes in the area of the kiln inlet.

Mounting sampling probes

The principle mounting arrangement is shown in Figure 4. Owing to the wide range of kiln systems available, it is impossible to define an exact mounting point for all types of kilns in advance. The following criteria play a dominant role:

- Available space
- Accessibility
- Direction of rotation of the kiln
- Internals in the kiln inlet chamber

According to experience, the gas composition at the kiln inlet is not homogeneous. In consequence, the sample is only representative of the average gas composition to a limited degree, irrespective of where the probe is positioned. The measuring point or sampling point in the rotating part of the kiln, as will be shown later, cannot be determined primarily on the basis of measuring criteria. An individually adapted interpretation of the measured values (averaging), correlation with disturbance variables, etc.) is far more important here than the position of the probe (see chapter 9).

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The mounting position shown in Figure 4 is for reference only. Positions and dimensions can vary considerably from plant to plant. The undisturbed, reliable function of the probe and the warranty that no false air will be drawn in from the kiln inlet seal always has the highest priority.

The probe must be easily accessible. If necessary, special platforms must be erected. A lifting device placed above the probe will considerably facilitate maintenance. The minimum distance between the platform railing and probe axis should be about 1.3m (at least on one side).

As the probe consists of relatively long sections, the access way must not be too narrow. The mounting position should be accessible via steps not ladders.

Pokeholes, which must be operated regularly, are located in the immediate vicinity of the probe. It must be ensured that the sampling probe does not interfere with the working space provided for maintenance personnel.

In principle, the probe must be mounted on the side where the direction of rotation of the kiln shell is directed downwards. If conditions do not permit, the other side can also be selected, provided the probe can be positioned at a sufficient distance to the material bed.

Meal inlet pipes, pokeholes, measuring connections, etc., mostly restrict free selection of the mounting position, the main aim is to minimise hindrance of maintenance personnel.

The inevitably cooled probe surfaces encourage the formation of deposits in the kiln inlet chamber. These deposits not only hinder subsequent removal of a probe, they can negatively influence the geometry of the kiln inlet chamber and also cause pressure loss. To prevent this happening, it is recommended to regularly blow-out the gap between the pipe and probe by means of a mounted air cannon. The pipe must be dimensioned, so that a gap of about 25mm is produced between the probe and inner pipe wall.

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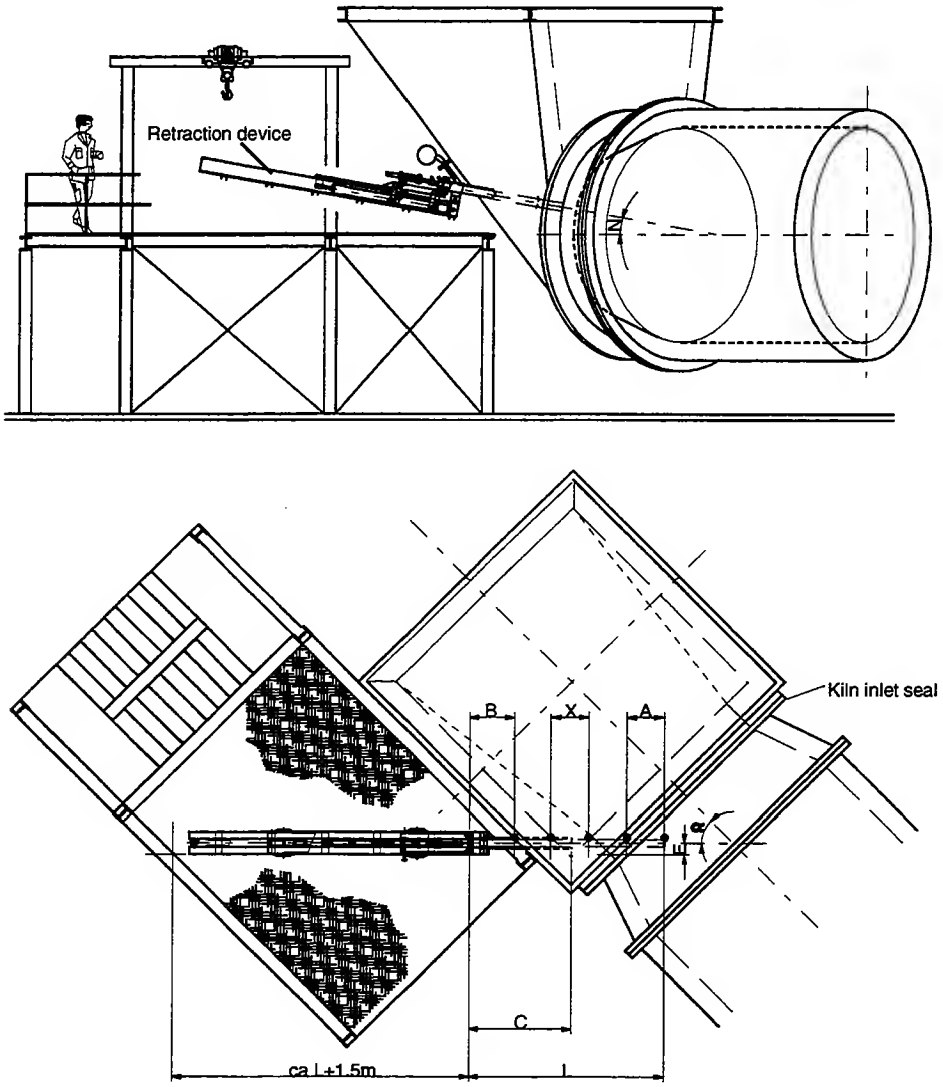
Figure 4: Mounting of the gas sampling probe at kiln inlet



A = 0....300 mm	α = 30°....45 °
B = Min. 500 mm	L = Nominal length of the probe
C = B + wall thickness	N = Inclination from horizontal axis
E = Min 200 mm	X = As short as possible

5.1.2 Wet kilns and long dry kilns

Process conditions at the sampling point:	
Temperature wet kilns	150 to 200°C
Temperature long dry kilns	250 to 500°C
Dust content wet kilns	about 40g/Nm^3
Dust content long dry kilns	about 300g/Nm^3
Dew point temperature	60 to 70°C
Dew point temperature long dry kilns	35 to 40°C

Under these conditions, gas sampling is much simpler than for kilns with preheaters. No cooling of the probe is necessary and the measured signal has improved representativity, as better intermixture of the gases takes place due to the internals (chains, crosses). With regard to positioning, mounting conditions, etc. roughly the same conditions apply as for kilns with preheaters.

5.1.3 Special instructions for gas sampling at the kiln inlet

Cyclone and grate preheater kilns

The above process conditions show that gas sampling from this difficult environment requires particular measures:

- Owing to the high temperature, the sampling probe must be cooled.
- A filter system must ensure that the large amounts of diversely structured dust is removed prior to sample gas preparation
- Very often, deposits build up at the sampling system and clog the probe entrance, so that measures must be taken for their prevention.

Probe cooling

Liquid cooling is used for all known probes. The cooling medium, which is usually water, transfers the heat in a closed system to water-air or water-water heat exchanger to prevent calcareous deposits. The energy produced varies considerably. Depending on the condition of the probe surface, a large amount (clean surface) or small amount (dust or deposits on the probe) must be removed. Based on experience, an energy flow of maximum 30kW per probe meter must be removed within the kiln (probe with 10cm outside diameter). In extreme cases, the energy flow varies between 1kW and 30kW per probe meter.

According to experience, the risk of deposit formation increases with reducing probe or cooling circuit temperature. As a countermeasure, some probes are operated at higher temperatures. This takes place by controlling the cooling circuit or by internally heating the probe.

If a water-air heat exchanger is used, it must be taken into account in dimensioning that the maximum cooling capacity is still sufficient even with heavy clogging of the heat exchanger surfaces. It is also recommended to add an anti-freeze to the water in cold regions.

In order to better control incrustations and deposit build-up in and on the probes, cooling systems were developed which enable the probes to be operated at higher temperatures. An example is the steam jacket probe. This probe functions as a heat conducting tube, whereby the heat of evaporation of water is used as a heat carrier to the directly flanged-on heat exchanger (condenser).

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The probe temperature is 120 to 150°C, the internal pressure is 1.5 to 5 bar. The probe only functions properly if it can be mounted almost vertically (less than 30° from the vertical axis). This requirement virtually excludes the mounting method shown in Figure 4 in an almost horizontal position, so that its possible applications are very restricted.

If a synthetic heat carrying liquid is used instead of water, the probes can be operated at even higher temperatures (up to 200°C) (see chapter 6.1.1).

Safety and maintenance equipment

In the event of failure of the cooling system, the probe would be damaged within a few minutes. Countermeasures are essential. For emergency cooling purposes, water supplied from storage tanks can be used and feed directly into the probe circuit if required. Emergency cooling systems of this kind must also function in the event of a total power failure. Pneumatically operated withdrawal devices have proved more reliable than emergency cooling systems (see Figure 5). Since compressed air can be stored without problems, pneumatic operation has the advantage that the probe can be withdrawn from the danger, zone even in the event of a total power failure, within a reasonable time. The automatic insertion and withdrawal device also has further significant advantages:

- Regular insertion and withdrawal prevents the formation of deposits on the probe and their build-up.
- Maintenance is considerably facilitated.

All connections to the probe (hoses, cables, etc.) must be flexible.

Dust filtration in the probes

With dry filtration, the sample gas is withdrawn from the process by suction in an unpurified condition. Dust separation takes place inside the probe or at its outer end. The produced dust must be removed from the filter from time to time. Whether this takes place manually by replacing the filter element or automatically by blowing back into the sampling space, the sample gas flow is always interrupted. This interruption has a negative effect on the availability of the measuring instrument. To prevent condensation of any kind, the filter must have a minimum operating temperature of 150°C.

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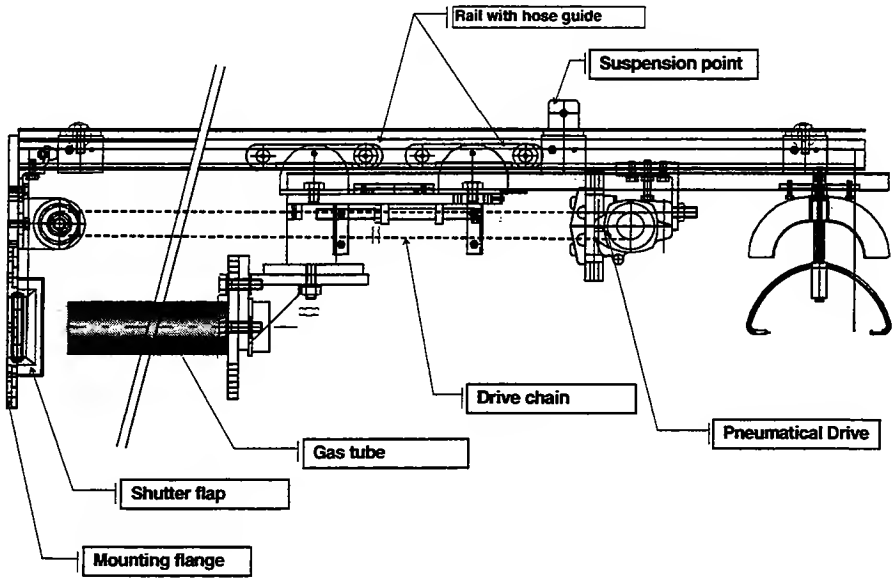
Figure 5: Probe extraction device



With wet filtration, the sample gas flows through a water curtain at the probe tip. This produces a slurry of water and dust, which together with the sample gas discharges through the wet probe, which in contrast to the dry probe is inclined outwards. The slurry discharges, the gas is dried and passed to the analysers.

Wet filtration has the disadvantage that various gas components are also dissolved in water. SO_2 is completely washed out and the CO_2 concentration reduces to an uncontrollable degree. As a result of this change in the total composition, the relative contents of the other gas components increase, so that the composition of the sample gas is no longer correct. If the injector water circulates in a closed system, the circulated water becomes acidic and behaves virtually inert towards CO_2. However, susceptibility to corrosion of all parts coming into contact with the water must be observed.

The level of maintenance necessary with wet filtration is slightly less and above all technically less demanding than with dry filtration. However, in view of the distortion of measured values and the reliable dry systems that are currently available, its application is no longer recommended.

Filter cleaning

The dust separated in the dry filter must be removed from time to time. The degree of clogging of the filter, including the gas sampling probe, can be determined by means of a vacuum gauge arranged between the filter and sample gas pump. The maximum permissible value for cleaning differs from plant to plant and must be individually determined.

Compressed air is normally used for cleaning and removing the dust from the filter. If the filter is arranged at the outer end of the probe, cleaning should take place in two stages.

1) Loosening the dust via the sample gas pipe.

2) Blowing out the dust via a separate compressed-air pipe.

The compressed air must be free from oil and water. Oil in the purging air produces incorrect CO in the filter and excessive moisture encourages encrustation of the filter.

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With regard to gas sampling, the conditions for the wet kiln and long dry kiln are considerably simpler than for preheater kilns. The lower temperatures usually obviate the need of probe cooling. The lower dust load enables the use of simpler filter systems without special cleaning, as used following the heat exchanger in the case of dry kilns (see chapter 6.3).

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5.2 Other measuring points (B to G)

5.2.1 Kiln inlet chamber and riser duct (measuring point B)

Process conditions at the sampling point:	
Gas temperature	820 to 1300°C
Dust content	200 to 1200g/m^3 (N.tr.)
Dew point temperature	35 to 40°C

In the kiln inlet chamber and riser duct, as at the rotary kiln inlet, local and time-related concentration differences can be expected. Above the meal inlet of the lowest cyclone, there is no longer any danger of a deposit build-up through circulated materials. However, the gas composition no longer corresponds at this point with the gas composition in the rotary kiln inlet. It is distorted by the reduction of CO in the riser duct and through dilution with false air and CO_2 from deacidification of the burned product.

When burning secondary fuel, the measuring point must be arranged below the secondary fuel inlet.

With regard to gas sampling, virtually the same conditions apply as at the kiln inlet. This applies similarly to a large extent to sampling from the calcining chambers of grate preheater kilns.

5.2.2 Lower cyclone stages (measuring point C)

From a measurement point of view, gas analysis in this area is virtually pointless, particularly in plants with pre-calcining. However, if measurements are still carried on a frequent basis, these are not for measurement reasons, but the more favourable conditions with regard to sampling (deposit formation and temperature) than further below, e.g. in the rotary kiln inlet. A cooling system is necessary in any event.

5.2.3 Upper cyclone stages (measuring point D)

Process conditions at the sampling point:	
Gas temperature	400 to 550°C
Dust content	200 to 1200g/m^3 (N.tr.)
Dew point temperature	35 to 45°C

Protection of the electrostatic filter against explosions due to impermissible CO concentrations, presupposes a rapid reaction of the gas analysis (see chapter 12.1). For this task, gas sampling below the uppermost cyclone stage is preferable, as the residence time of the main gas flow in the uppermost cyclone stage prolongs the reaction time of the monitoring device. Two-section preheater systems should always be provided with two complete measuring instruments (one for each section).

The sampling conditions are relatively simple. No cooling is necessary. In order to achieve rapid reaction times, the maximum amount of gas must be drawn in (60 to 300 l/h). However, as the amount of gas increases, so too does the dust content in the sampling system, so that, depending on the particular system, it must be separated by the filter and blown back again. More suitable for this purpose are probes whose filters are located

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directly in the gas flow (internal filter probes) than those with external filters (see chapter 6.3).

5.2.4 Downstream of cyclone or grate preheaters (measuring point E)

Process conditions at the sampling point:	
Gas temperature for rotary kilns with cyclone preheaters	300 to 400°C
Gas temperature for rotary kilns with grate preheaters	100 to 150°C
Dust content for rotary kilns with cyclone preheaters	20 to 70gm/^{3} (N.tr.)
Dust content for rotary kilns with grate preheaters	2 to 10g/m^3 (N.tr.)
Dew point temperature for rotary kilns with cyclone preheaters	35 to 45°C
Dew point temperature for rotary kilns with grate preheaters	50 to 65°C

At the point, similar conditions prevail for gas sampling as in the riser duct (measuring point D). In the case of grate preheaters, this point approximately corresponds with the conditions downstream of the grate, but less dust loading. To be taken into account for CO monitoring in grate preheater kilns is the fact that no cooling tower is usually available to delay the residence time of the gases in a positive sense. For this, the gas quantity can be increased on account of the low dust loading.

5.2.5 Downstream of electrostatic filter (measuring points F, G)

Process conditions at the sampling point:

Gas temperature	100 to 150°C
Dust content	>50mg/m^3 (N.tr.)
Dew point temperature	50 to 65°C

In the majority of cases, the components relevant for emission are measured. Preference must be given to the point downstream of the fan, as the gas composition is no longer subject to change before the exhaust gas discharges from the stack. Owing to the turbulence of the process gas in the plant sections and fans, it can generally be assumed that the process gas is well intermixed.

Owing to the probability of filter shutdowns, gas sampling requires a dust filter as provided at the measuring points in the raw gas flow. The high dew point makes it necessary for the probe to be heated at the critical points. No differences exist with regard to the type of kiln (cyclones, grate preheaters or wet kilns).

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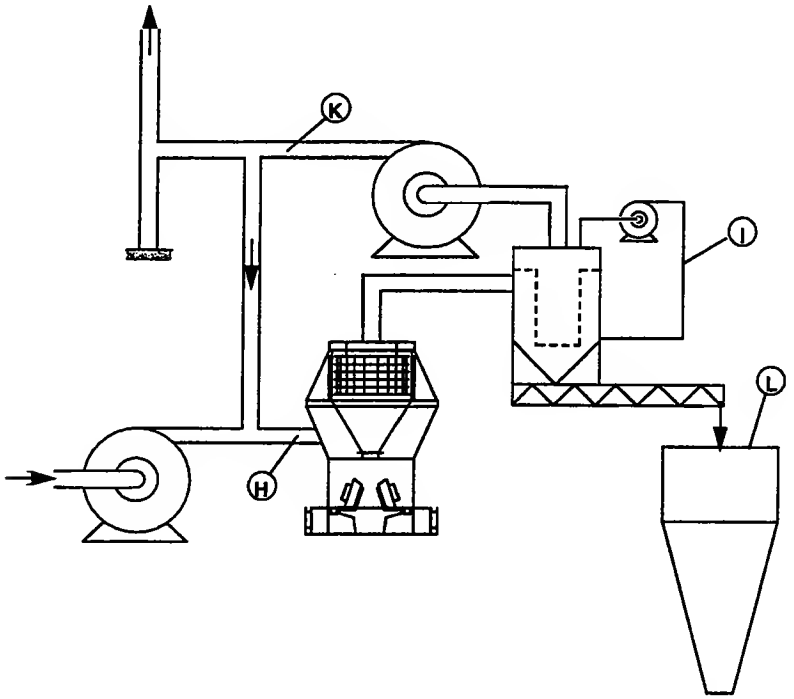
5.2.6 Coal pulverising mill

Gas analysis in the area of the coal pulverising mill serves solely for the purpose of quickly detecting smouldering fires and/or potentially explosive gas concentrations. The possible measuring points are shown in Figure 6.

Figure 6: Measuring points at the coal grinding plant



Upstream of coal pulverising mill (measuring point H)

Process conditions at the sampling point:	
Gas temperature	250 to 350°C
Dust content	20 to 70g/m^3 (N.tr.)
Dew point temperature	35 to 45°C

In filter (measuring point I)

Process conditions at the sampling point:	
Gas temperature	90 to 110°C
Dust content	>50mg/m^3 (N.tr.)
Dew point temperature	50 to 70°C

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The measuring point in the filter serves especially for monitoring glow nests when the coal pulverising mill is stationary. Owing to the large filter volume, glow nests can only be detected quickly enough with sufficient circulation of the process gas. For this reason, 200-300m^3/h process gas should be drawn out by suction via a separate gas pipe and fed back again at the filter inlet when the coal pulverising mill is stationary. The gas sampling probe is then arranged in the gas pipe. When the coal pulverising mill is in operation, the gas pipe is closed via a valve.

Downstream of coal pulverising mill fan (measuring point E)

Process conditions at the sampling point:	
Gas temperature	90 to 100°C
Dust content	>50mg/m^3 (N.tr.)
Dew point temperature	50 to 70°C

Coal dust silo (measuring point L)

Process conditions at the sampling point:	
Gas temperature	>60°C
Dust content	1 to 1000g/m^3 (N.tr.)
Dew point temperature	>35°C

The gas sample is preferably positioned in the silo roof.

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6. GAS SAMPLING PROBES

In principle, gas sampling probes can be divided into three categories according to temperature range:

- Sampling probes in the temperature range 900 to 1500°C
- Sampling probes in the temperature range 500 to 900°C
- Sampling probes in the temperature range up to 500°C

The following assessment of various gas sampling probes is based on the manufacturer's documents as well as our own experience and information from the works, if available.

6.1 Sampling probes in the temperature range 900 to 1500°C

This category includes gas sampling at the inlet of preheater kilns with and without pre-calcining.

6.1.1 Siemens type FLK

Brief description

Compact gas sampling system with special cooling fluid permitting probe operating temperatures up to 230°C. The condensation of alkali vapours and in turn the danger of incrustations is drastically reduced. The suction opening is arranged on the side of the probe tube and should be located in the flow slipstream. This arrangement offers selective dust separation. An electrically or pneumatically operated insertion / withdrawal device is an integral part of the probe.

Design:	Extremely compact and professional design. Oval stainless steel sampling tube.
Suction opening:	Lateral, about 30mm in diameter
Mounting tube:	325mm diameter (relatively large)
Dust removal filter:	External, heated, blow-back with compressed air, large volume
Maximum probe length:	3200mm

Comment

Despite effective measures, stubborn blockages can occur, particularly when kiln operation is loaded by high alkali circuits. Owing to the lateral arrangement of the suction opening, the gases within the probe tube flow through an elbow. The removal of incrusted deposits from this elbow is far more difficult due to poor access than if the gases were drawn out by suction at the end of the stem and through a straight tube. In view of the turbulent flow conditions at the sampling point, it is doubtful whether lateral suction results in selective dust separation.

The large overall volume of the dust filter permits a relatively large gas throughput. The resulting delay and damping of the indication is of little significance from a measurement point of view.

Available for the drive of the withdrawal device is either an electric motor or compressed-air drive. As compressed air can be stored without problems, the compressed-air drive offers more reliability than an electric motor in the event of a power failure. Emergency operation with crank handle is also provided.

With difficult kiln operation, alkali condensation can occur and block the entrance of the probe, as already mentioned, despite increased operating temperature.

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6.1.2 Harman & Braun type 60S (formerly type 13)

Brief description

Gas sampling probe with closed cooling system (cooling medium water) and temperature control. Integrated emergency cooling system with fresh water. Lateral arrangement of two suction openings (Figure 7).

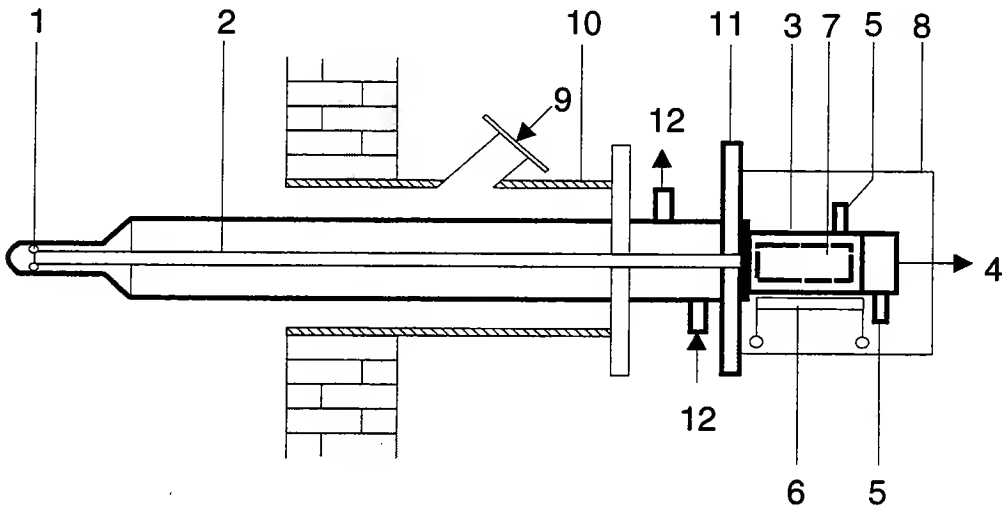
Comment

Widely used. Functions well in non-extreme alkali conditions. The same remarks apply here with regard to the lateral arrangement of the suction openings as for the Siemens system.

With large amounts of dust, the filter must be blown-out at brief intervals, The resulting loss of signal can have a critical effect on the availability of the measuring signal.

The operating temperature is too low to completely prevent alkali condensation. An automatic insertion and withdrawal device is optionally available.

Figure 7:



1 Gas inlet holes
2 Gas tube
3 Filtering unit
4 Gas outlet
5 Compressed air inlet
6 Ring heating element
7 Ceramic fliter
8 Protection case
9 Connecting for air blaster
10 Built in tube
11 Mounting flange
12 Cooling water in/outlet

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6.1.3 Hartmann & Braun type H ("Holderbank" probe)

Brief description

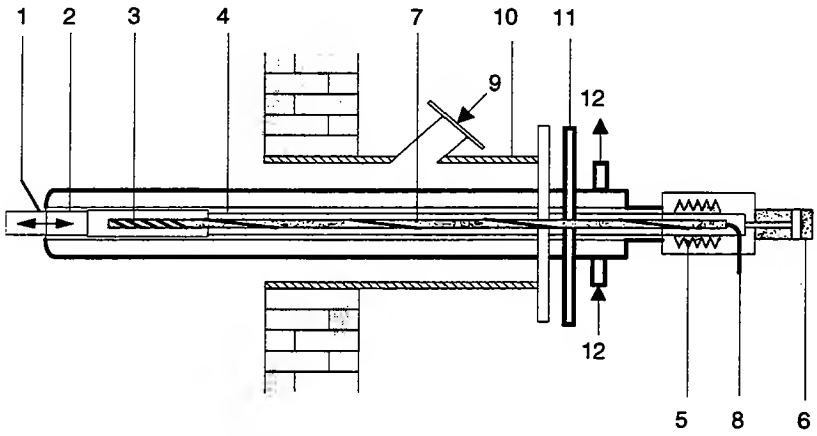
The main features of this probe are the automatically functioning mechanical removal of any incrustations at the probe inlet as well as a new type of filter system (see Figure 8). Cooling takes place in a closed system (cooling medium water) with temperature controller.

About 20 probes of this type are in use worldwide and are functioning efficiently. Marketing takes place via Messrs ELSAG BAILEY Hartmann & Braun in Frankfurt under the designation type H.

Design:	Probe body of alloyed steel
Suction opening:	At end of stem, about 40mm in diameter
Probe temperature:	External, 30 to 85°C, internal up to 250°C
Mounting tube:	150mm in diameter
Dust removal filter:	Internal, heated filter of sintered metal, cleaning with compressed air
Maximum probe length:	3000mm

Thanks to its modular concept, the probe can be provided with the same insertion and withdrawal device as type 60.

Figure 8:



1 Measuring gas inlet
2 Piston
3 Filter
4 Tube
5 Expansion seal (Stainless steel)
6 Air cylinder
7 Heated gas tube
8 Protection case
9 Anschluss für Luftkanone
10 Montagestutzen
11 Montageflansch
12 Kühlwasseranschlüsse

Comment

This sampling system has proved highly reliable in practical use, even under difficult conditions. The availability satisfies the high requirements of "high-level control systems". The maintenance costs are low thanks to the simple and maintenance-friendly design of the system.

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6.2 Sampling probes in the temperature range 500 to 900°C

With regard to the sampling probes, similar conditions prevail in this temperature as at higher temperatures (e.g. at the kiln inlet of preheater kilns).

However, the danger of incrustations is lower. Cooling of the probes is essential down to 500°C, so that in principle, the same sampling probes can be used as in the higher temperature range.

Temperatures between 500 and 900°C are typical for cyclone preheater kilns in the area of the lower cyclone stages (measuring point C). As already noted, measurements there are almost pointless, particularly in plants with precalcining. If measurements are still to be carried out despite this, steam jack probes are considered ideal for this purpose when mounted vertically in the cyclone roof.

6.3 Sampling probes in the temperature range below 500°C

Typical for the application of such probes are the measuring points downstream of the heat exchanger or grate preheater, upstream or downstream of the electrostatic filter as well as at the kiln inlet of wet or long dry kilns. Cooling is unnecessary. With regard to the place of use or purpose of measurement, there are certain differences in the design of the probes (e.g. mounting length). Decisive for the quality of these probes is the dust filtration. The discovery of fine-pore filter tubes of sintered stainless steel brought about great advances. The level of maintenance is low, a block-back device is unnecessary. The sample gas is clean and available for weeks to months without interruption. Probes with such filters are not available from established suppliers. They still use fine ceramic filters that enable a comparatively long useful life. However, it must be ensured that the gas conducting tube is heated at the penetration point (Figure 9).

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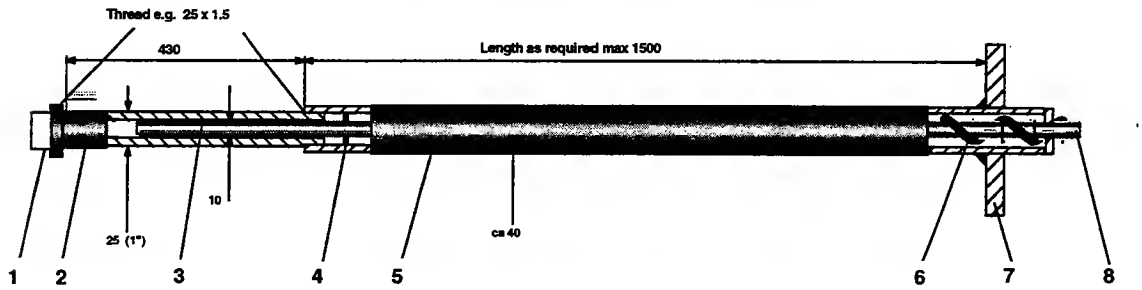
The optimal porosity of sintered metal filters is only 0.5m. As the main proportion of the dust particles is larger, the pores clog at a correspondingly slow rate. According to experience, they have a useful life of four weeks to several months, depending on the properties of the dust. If the pressure drop becomes excessive due to clogging, the filter tubes can be cleaned with acid.

With wet kilns and long dry kilns, the probes must be lengthened, depending on the size of the smoke chamber. In order to ensure the necessary mechanical strength, supporting tube of a larger diameter must be mounted, into which the probe is inserted.

A further important element is the heating in the penetration zone from the measuring channel to the outside. In order to prevent any kind of condensation, the temperature must not fall below 100°C at any point.

For all gas conducting parts, stainless steel, e.g. No. 1.4541 or 1.4571 must be used.

Figure 9: Gas sampling probe, up to 500°C



1	Stainless steel closing
2	Filter but: Typ 1400.1.000-.18-0.5 threaded on both sides e.g. M 25 Manufacturer: Mott Metallurgical Corp. Farmington Industrial Park Farmington CT 06032 Fax No (USA) 203 674
3	Gas condcuting tube, Outside diam. 10 mm, inner diam. 6 mm, stainless
4	Centering discs approx. every 500
5	Carrier tube. Outside diam. 40 mm, wall thichness 2 mm, stainless
6	Heating coil Heated lenght approx. 300 mm ca. 200
7	Mounting flame
8	Sample gas connection

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7. SAMPLE GAS PREPARATION

The purpose of sample gas preparation is to supply the downstream gas analysers with sample gas, so that it conforms qualitatively and quantitatively with the specifications of the analysers. Such an arrangement normally consists of the following elements:

- Supply > Sample gas pipe
- Drying > Sample gas cooler
- Delivery > Sample gas pump
- Dosage > Controller
- Distribution > Pressure control valve
- Calibrating option > Manual or automatic

Experience has shown that the availability of a measuring instrument depends essentially on a properly functioning sample gas sampling and preparation. Insufficient sample gas preparation leads to fouling in the gas analysers and not infrequently to expensive damage.

7.1 Sample gas pipe

The probe itself as well as the sample gas pipe between the sampling probe and sample gas cooler should be heated to between 100 and 150°C (for measurements with FID devices up to 200°C). This heating has the following purpose:

- To prevent condensation and freezing of the condensate at low temperatures, because:
  - Condensation in the sample gas pipe can distort certain measured values.
  - Incrustations and blockage of the sample gas pipe are accelerated by the formation of condensate.
- To delay the conversion of NO into NO_2 during emission measurement.
  Heated sample gas pipes are available in various qualities:
- Low price pipes with a self-regulating heating band and
- Pipes with resistance heating and separate temperature controller

For selection purposes, it should be noted that with heating band controlled pipes, the temperature is limited to a maximum of 105°C. This temperature does not normally suffice for process measurements but for emission measurements. The following specifications must be observed:

- Inner tube material: Teflon
- Inner tube dimensions: Outside diameter 6mm
  Inside diameter 4mm
- Connection ends: Clamping ring tube fittings
- Control: Thermocouple with controller
- Protective sleeve: PVC or metal

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Heated pipes are relatively expensive. When planning a measuring system, the sample gas pipes must therefore be as short as possible. The following points must be observed:

a) If only O_2, CO or NO is measured, heating can be dispensed with if the sample gas pipe (several metres) is short and can be laid with uninterrupted gradient from the sampling point to the ample gas cooler (no siphons).

b) Possible installation of the sample gas cooler in the vicinity of the sampling point.

For automatic compressed-air cleaning via the sample gas pipe, it must be ensured that this has sufficient pressure resistance.

For connections, high-quality stainless steel clamping ring fittings, e.g. Swagelok, Serto or similar, must generally be used.

7.2 Sample gas coolers

Sample gas coolers serve to remove most of the water vapour from the sample gas at temperatures of about 3°C and keep the residual content at a constant value. The latter is important particularly for emission measurement. The temperature indication of the sample gas cooler normally shows the operating temperature of the cooling medium. If the sample gas cooler is subject to excessive load or ambient temperature, the sample gas temperature at the cooler outlet can be considerably higher than the indication. The dew point temperature then no longer corresponds with the cooling medium indication.

The sample gas cooler is usually accommodated in a cabinet together with the analysers and the remaining sample gas preparation apparatus. Instead of using long heated sample gas pipes, it may be advantageous to install the sample gas cooler immediately downstream of the sample gas probe in a separate equipment cabinet.

Condensate collecting tanks have the disadvantage that indication of the measured values is greatly delayed due to the additional volume. The produced condensate should therefore be pumped out using automatic hose pumps or diaphragm liquid pumps.

7.2.1 Specifications of a suitable sample gas cooler

Dew point at outlet	+5°C (±1°C)
Dew point stability	±0.25°C
Temperature at sample gas inlet	max. 180°C
Dew point at sample gas inlet	max. 80°C
Gas flow	max 100l/h
Ambient temperature	+5 to +45°C
Cooling capacity	860kJ/h, 25°C
Material of gas conducting parts	Teflon, PVDF
Permissible gas pressure	min. 3 bar
Volume	max. 100cm^3
Sample gas connections	6mm or 1/4"
Time until ready for measurement	max. 30 minutes

Also available for the majority of sample gas coolers are moisture monitoring devices at the sample gas outlet. Such devices are urgently recommended.

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7.3 Sample gas pump

Mainly diaphragm and compressor pumps are used. The sample gas volume necessary for the gas analysers is between 30 and 100l/h. In order to keep indication delays to a minimum, a larger sample gas volume is often extracted from the process gas. The excess sample gas can be diverted via a bypass shortly upstream of the analyser. If possible, a large part of the measuring instrument should be operated under pressure, as false air can infiltrate during suction phase.

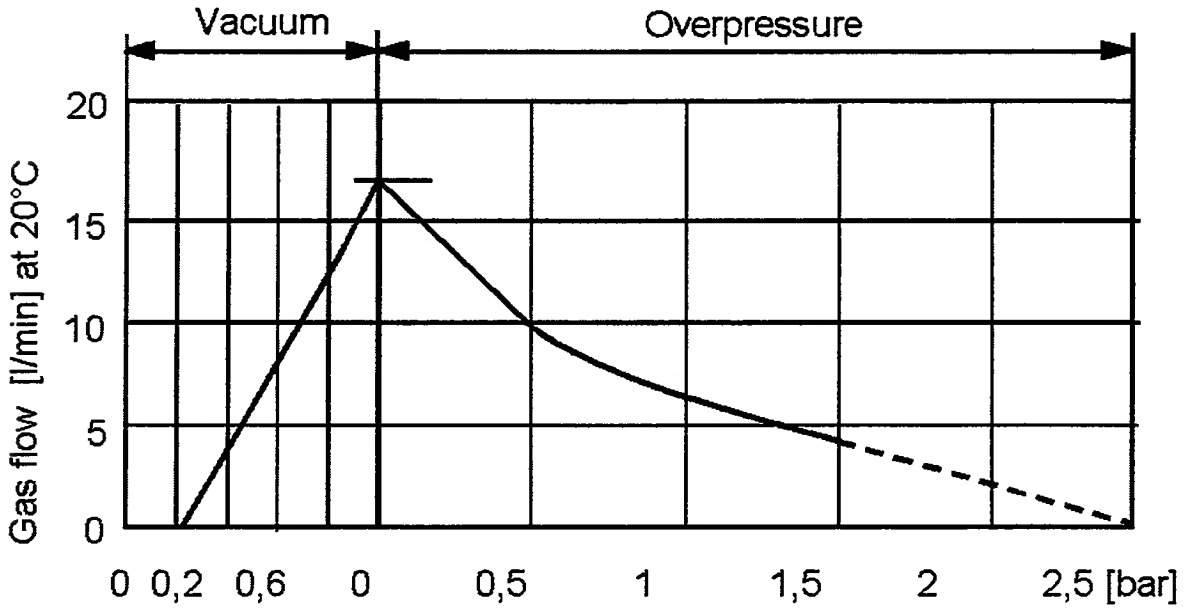
Sample gas pumps installed upstream of the sample gas cooler must be heated. This arrangement has the advantage that no false air is sucked in the event of leaks in the sample gas cooler. However, according to experience, heated pumps are more susceptible to faults than cold operated pumps, so that arrangement downstream of the sample gas cooler is still more advantageous.

The pump capacity should be generously dimensioned in respect of the suction pressure. The flow rate must be dimensioned, so that the available gas flow about 1.5 to 2 times higher than that required by the analysers according to specifications. Throttling of the flow preferably takes place on the low-pressure side.

With throttling on the pressure side, a pressure control valve must ensure that the pressure downstream of the pump does not increase to an unnecessary degree. Significant pressure drops in the sample gas encourages the formation of powdery salts (white powder). The following diagram shows the characteristics of a suitable sample gas pump.

- Vacuum
- Excess pressure
- Gas flow [l/min] at 20°C

Figure 10:

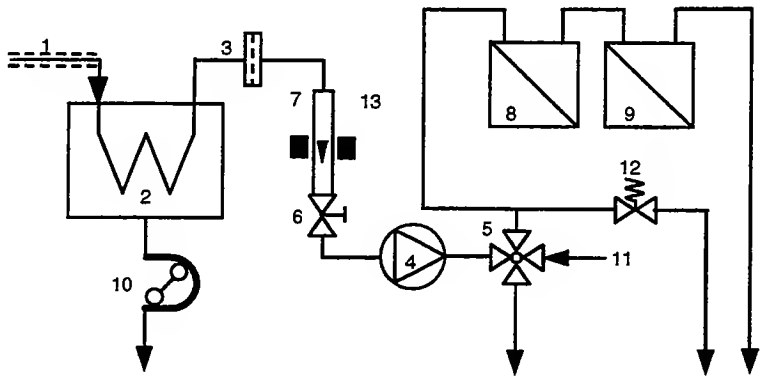


7.4 Flow control and distribution

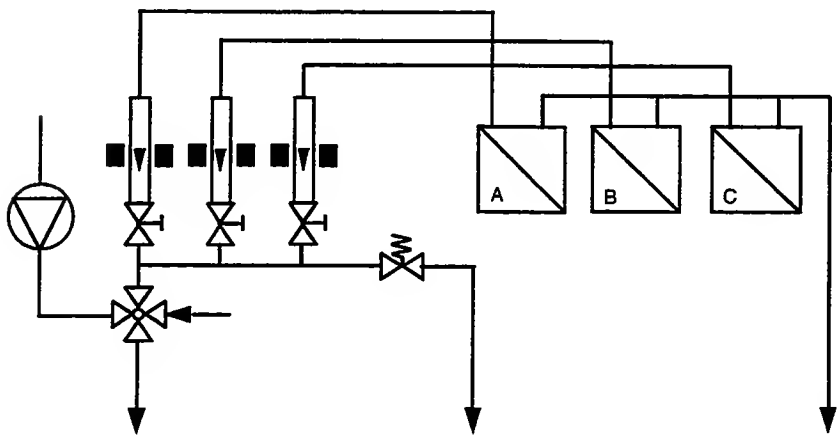
The arrangement for control and distribution of the sample gas depends on the number and specifications of the analysers to be supplied. Multi-component analysers with only connection solve the problem of gas distribution internally. However, as soon as two or more analysers have to be supplied with gas, the inlet pressure must be taken into account in distribution. At a low supply pressure, the analysers can be connected in series. If analysers have to be connected with varying supply pressure, they must be connected in parallel via appropriate gas distribution.

7.4.1 Simple gas control

Figure 11:



1 Sample gas inlet
2 Sample gas cooler
3 Fine filter combined with wetness detector
4 Sample gas pump
5 Changeover valve measurement / Calibration
6 Throttle valve, Gasflow setting
7 Floating body flow meter
8 Analyzer 1
9 Analyzer 2
10 Condensate pump
11 Calibration gas input
12 Overpressure valve
13 Min. flow contact



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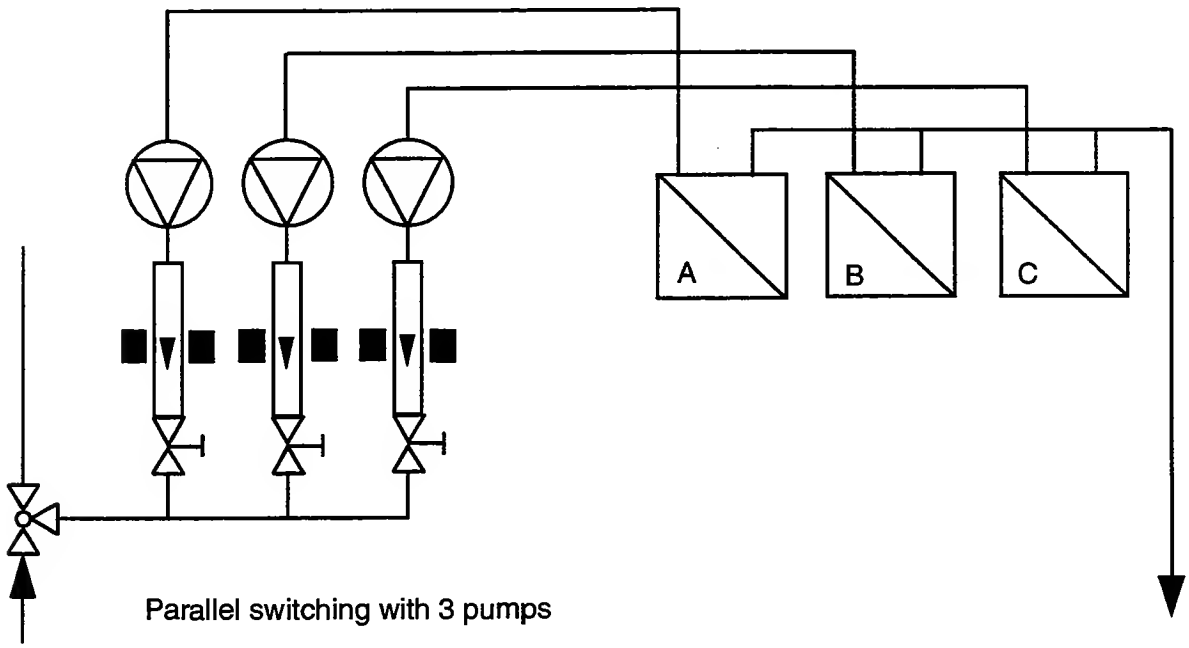
The flowmeter monitors and controls the set sample and test gas volume. Rotameters fitted with variable limit transmitters are normally used for setting the flow rate. The signal transmitters should signal especially when the necessary flow is not reached. This is particularly important for gas analysers with safety functions.

As already mentioned, not all gas analysers have the same conditions with respect to supply pressures. Optical analysers normally require less pressure than oxygen analysers, for example, which use the paramagnetic measuring principle. However, special attention must be given to the supply pressure of the analysers when dimensioning the sample gas preparation apparatus. As soon as several analysers have to be operated, parallel connection is recommended, despite the additional work involved. For this purpose, the sample gas flow must be divided appropriate to the number of gas analysers to be supplied. The following flow diagram shows how this division can take place. The sample gas pump supplies a common supply pipe, which is under increased sample gas pressure. A spring-loaded pressure control valve controls the pressure in this pipe. The excess sample gas is diverted. A further, slightly more elaborate option, is to use a separate sample gas pump for each analyser.

Parallel connection with pressure control valve

Figure 12:



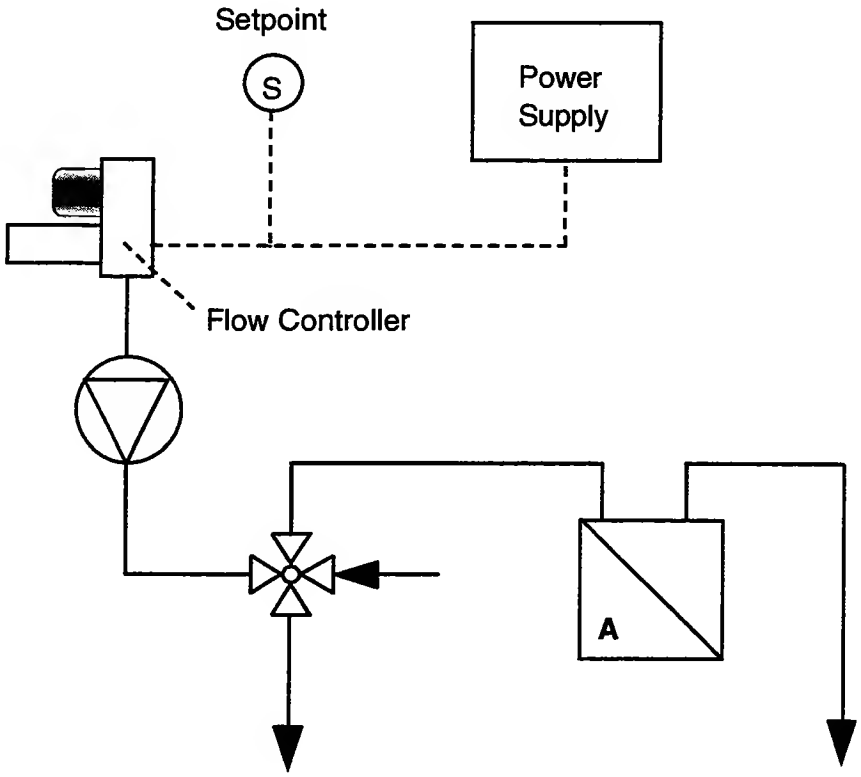
Parallel switching with 3 pumps

7.4.2 Electronic gas volume control

The available electronic flowmeters are reliable and should be used to an increasing extent for automatic gas volume control purposes.

These are flowmeters whose measuring principle is based on a thermal measuring bridge, whereby the transported heat of a flowing substance is used as an indicator. The calibrated reference quantity is normally air. For other gas compositions, appropriate correction values (e.g. CO_2) must be used for conversion.

Figure 13:



Tests with automatic control have confirmed that the reliability of the gas supply can be considerably improved with this system. It offers decisive functional advantages compared with rotameters and is less susceptible to faults. A constant gas flow positively affects overall gas analysis in every respect. Since the controllers have a throttling effect on the gas flow, they are preferably arranged on the suction side of the pump. For flow indication purposes, a volume-proportional signal is available. This enables ideal monitoring via the control system. In addition, a rotameter (without needle valve) can still be used for monitoring locally.

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7.5 Sample gas filters

7.5.1 Coarse filters

Coarse filters are normally installed inside or downstream of the probe, so that a large proportion of the dust is already separated before the sample gas enters the sample gas preparation apparatus. If sintered metal filters with a porosity of <0.5m are already used in the probe, the sample would be sufficiently clean. Nonetheless, fine afterfiltration directly upstream of the analysers is recommended, as impurities can also infiltrate the system downstream of the probe. Ceramic filters of silicon carbide, mineral fibres or filter casings of borosilicate glass fibres are also frequently used. Silicone carbide composites have a high separation efficiency of about 99.9% at an average grain diameter of 1.2m and are temperature resistant up to 660°C.

The disadvantage of all these filters compared to sintered metal filters, apart from their lower separation efficiency is that they have to be cleaned and blown out or replaced at shorter intervals

7.5.2 Fine filters

For separation of the dust particles and sublimated salts left behind in the sample gas, membrane filter, e.g. of glass fibres or PTFE should be provided.

Recommended is a combination of membrane filter and condensate monitor. The condensate monitor measures a change in the electrical conductivity between two electrodes. If a limiting value is exceeded, the sample gas pump is stopped and the fault is indicated in the control room.

7.5.3 Valves

Changeover valves

If automatic blowing out of the filter and/or probes is required, the gas path must be able to be diverted automatically. For this purpose, stainless steel solenoid valves or mechanically/ pneumatically operated ball sliding valves of Teflon (housing) and stainless steel (ball) are used.

Valves installed in moist sample gas (upstream of the sample gas cooler) must be mounted on a temperature-controlled valve plate in order to prevent corrosion as a consequence of the dew point not being reached.

Valves, particularly solenoid valves, have the tendency to leak, even when slightly fouled. They enable a high degree of automation of the measuring instrument, e.g. automatic calibration, however, experience has shown that their susceptibility to faults almost cancels out the convenience they are expected to provide (see chapter 11.7).

7.5.4 Pressure control valve

The purpose of the pressure control valve is to ensure that the supply pressure of the analysers remains fairly constant. The excess gas delivered by the pump discharges via a spring-loaded body. The nominal pressure of the valve must be slightly higher than the highest specified supply pressure of all parallel connected analysers.

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7.6 Adjustment

The necessity of regular adjustments depends on the purpose of the analysis and the stability of the analysers. For emission measurements, a rotation of one to two weeks is sufficient, provided the regulations in the country concerned do not contain any tighter requirements. During adjustment, a preferably manually-operated two-way directional control valve switches the gas supply to a bottle containing test gas. Newer gas analysers are normally provided with an automatic adjusting device. By means of an internal or external command a glass cuvette filled with the respective gas is inserted into the sensor and the analyser is automatically readjusted. During this time, the last measured value remains indicated, so that the subsequent signal processing system is not disturbed by the procedure. With older analysers that do not have this type of automatic device, operating personnel or the process control system must be informed of the adjustment. With CO monitoring systems, activation of the automatic filter shutdown system must be prevented.

Analysers with so-called calibrating cuvettes have the big advantage that no more test gas is necessary. Manufacturers and most nationally customary regulations accept this type of adjustment over a period of 2...3 years. After this time, the analysers must be overhauled and inspected and readjusted by an appropriately licensed institute.

7.7 Further information on sample gas preparation

7.7.1 Sample gas discharge

No sample gas must discharge into the analyser room. Every gas outlet must be connected to a pipe and discharged to the atmosphere or returned to the main gas flow. Pipes with large cross-sections must be used in order to prevent any back pressure.

7.7.2 Tightness test

An option should be provided to easily connect the sample gas pipe gas-tight to a nitrogen bottle directly downstream of the sampling probe.

The complete measuring apparatus is purged with nitrogen with the gas preparation apparatus operating normally. The oxygen analyser shows "zero" indication soon afterwards. After several minutes, the nitrogen supply is interrupted, so that a vacuum is produced in the system. Caution! The regulating valves on the test bottles are often not vacuum-tight. If the measured value of the oxygen analyser should increase again within a space of about 20 seconds, this means that there is a leak in the system. An increase of several per cent within several minutes is considered normal.

7.7.3 Filter condition monitoring

The suction pressure is a good indicator of the condition of the filter or filters between the probe and sample gas pump. By measuring this pressure with a pressure transmitter or contact pressure gauge, the automatic cleaning process can be initiated when a certain level is reached. As a result of this optimising measure, the measured signal is not interrupted unnecessarily and the filters are cleaned or blown back before being irreversibly clogged.

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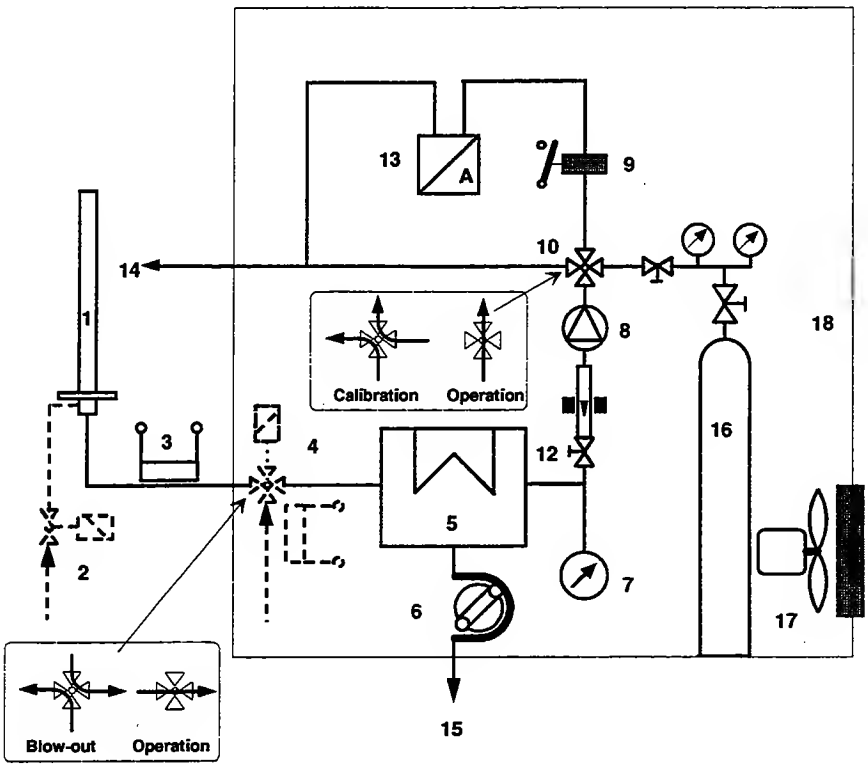
7.7.4 Analyser room

The accuracy and reliability of gas analysis essentially depends on temperature variations, vibrations, draughts, dust, moisture and electrical interference fields being avoided. For this reason, the gas analysers including sample gas preparation apparatus should be installed in a closed room or equipment cabinet that can be locked to prevent unauthorised access. In the analyser room, the individual instruments and apparatus must be arranged with an emphasis on clarity and easy accessibility, so that rational operation and maintenance of the measuring equipment is ensured.

Figure 14:



1 Gas sampling probe
2 If necessary: Blow-out device for outside of coarse filter)
3 Heated sample gas pipe
4 If necessary: Heated magnetic valve for blow-out device inside the filter
5 Sample gas cooler
6 Condensate pump
7 Suction pressure measurement (Manometer or pressure transmitter)
8 Sample gas pump
9 Fine filter with humidity monitoring
10 Changeover valve Operation/ Calibration (Manual actuation)
12 Control valves with floating body flow monitoring
13 Gas analyzers
14 Exit of superfluous sample gas
15 Condensate discharge
16 Calibrating gas
17 Cabinet air conditioning with suction filter
18 Cabinet

7.8 Space requirement

For the analysers and sample gas preparation apparatus, a double 19" equipment cabinet with one field respectively for the gas preparation apparatus and analysers is normally sufficient. However, other equipment is often installed in this clean, air-conditioned analyser room. In such cases, it must be ensured that sufficient room is provided to enable maintenance personnel to move freely and that there is adequate room for material (e.g. test gas bottles, tools, etc.) and additional space.

7.9 Location

In principle, the analyser room should be located as close as possible to the measuring point. This aspect is particularly important for CO monitoring, where fast indication is necessary. Unheated sample gas pipes must be laid descending to the sample gas cooler. It is therefore advantageous when the analyser room is located one floor below the sampling point.

If space is available at a suitable location that is not exposed to excessive dust, heat radiation and noise, a switchgear cabinet is sufficient for accommodating the measuring equipment. A separate, closed room is unnecessary under such conditions. The volume to be air-conditioned can be considerably reduced as a result.

7.10 Climatic conditions

The temperature in the analyser room or cabinet should be able to be regulated between 15 and 25°C. For this purpose, heating and ventilation is necessary; a cooling unit must be provided to counteract heat radiation. The ventilation fans must be provided with efficient dust filters. All filters produced during gas analysis must be discharged to the atmosphere.

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8. ANALYSERS

The determination of individual components in the sample gas is based on various physical basic principles, such as:

- Light absorption
- Paramagnetism
- Flame ionisation and
- Solid-state electrolytic systems

All newer analysers process the analogue signals output by the sensors digitally. Digitalisation has provided the following advantages:

- Increased operating convenience (menu guidance)
- Multi-channel technology
- Automatic adjustment
- More compact design
- Increased stability
- Remote monitoring and control via modem and/or data bus

The internal functions of the analysers will not be dealt with here. Sufficient information in this respect is provided in the manufacturer's documents and technical literature. Important for practical application is knowing how to use the various measuring methods effectively.

The manufacturer plays a secondary role in analyser selection from a qualitative point of view. Measuring equipment faults are rarely attributed to faulty analysers. It goes without saying that only types designed for industrial purposes are suitable for use in cement works and certainly no laboratory equipment. Suitability for industrial purposes means, for example:

- Solid, totally enclosed housing (degree of protection IP 65 or IP 54)
- Insensitivity to vibrations
- Minimum temperature application range 15 to 35°C
- Insensitivity to system disturbances
- Isolated (floating) output signals

The majority of established manufacturers are in a position to meet these specifications. Service and spare parts availability are therefore more important than the purchase price. If measuring equipment is already available and the above criteria are met, there is no reason to change manufacturers when purchasing new equipment or additions.

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8.1 Infrared absorption

With the exception of oxygen, the majority of interesting gas components can be measured by infrared absorption. Available are analysers for extractive and in-situ application. In the main, infrared is used for the extractive method. The following advantages are offered by these NDIR (non-dispersive infrared) analysers:

- Broad application spectrum
- Relatively simple method, favourable price
- Robustness
- Low wear, long useful life
- Good stability

The not particularly high sensitivity of the NDIR analysers does not normally pose a problem in the cement industry. The smallest measuring range, e.g. for CO is about 100ppm, for NO about 500ppm and for CO_2 about 20 %. Cross-sensitivities to water vapour and CO_2 are present. Water vapour as a disturbing component plays a secondary role when the gases in the sample gas cooler are dried with sufficient stability. The cross-sensitivity of CO_2 is usually within the tolerance when compensated in the analyser. The influence can be further reduced when an amount of CO_2 corresponding to the average value of the sample gas is mixed with the test gas. As the CO_2 content in the cement process is extremely high, unestablished analysers should be tested in this respect prior to their use.

The most well known manufacturers today produce multi-channel NDIR analysers, enabling the prices per measured component to be distinctly reduced. The majority of manufacturers also offer analysers in different quality classes (low-cost analysers). As a rule, the low-cost analysers are not temperature-stabilised, but temperature-compensated and are therefore not as stable as the analysers at the top end of the price scale. However, if external fluctuations are kept to a minimum, the temperature-compensated analysers meet the requirements in the majority of cases.

8.1.1 Space case, sulphur dioxide (SO_2)

In principle, NDIR analysers are suitable for the measurement of SO_2. However, tests on kiln systems with cyclone preheaters have shown that insufficient SO_2 is measured with extractive sample gas sampling and preparation (in accordance with DIN 2462, page 4). The magnitude of the deficiency can vary considerably and can therefore not be calibrated. These measuring errors are attributed to chemical reactions of the SO_2 with NH_3 (ammonia) in the sample gas cooler (see chapter 11.2). The following options are available to eliminate these errors:

- Acidification of the condensate by addition of phosphoric acid in the sample gas cooler (TÜV) tested method developed by and available from Hartmann & Braun).
- Use of a "hot analyser" (measurement with moist sample gas).
- In-situ measuring method

With the addition of about 8 to 12ml phosphoric acid (5%) in the sample gas cooler, the pH of the condensate reduces to a value below 1.5. The reaction of the SO_2 with ammonia is accordingly prevented.

It is obvious that hot analysers, whose entire measuring system must function at temperatures between 150°C and 250°C, are more susceptible than cold analysers. According to experience, the heated sample gas pump causes the greatest problems.

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With in-situ measuring instruments, no distortion through chemical reaction is noticeable. As part of comparative tests, they showed to be more reliable than measuring instruments with extractive sampling and sample gas cooler. However, absolute measured values in mg/Nm^3 are initially available after calibration using a reference measuring procedure. In addition, if "dry" standard values are required, the correction must be corrected with the moisture and temperature of the sample gas.

8.2 Ultra-violet absorption

The difference of the NDUV analysers compared with NDIR is mainly that they are less cross-sensitive to water vapour and CO_2 and higher sensitivities can be achieved. In the cement industry, NDUV analysers were initially used for the measurement of NO components. Meanwhile, the less expensive NDIR analysers are being more widely used for NO and have proved reliable.

8.3 Paramagnetism

Oxygen does not have a usable spectrum either in IR or UV light. In future, laser analysers /chapter 8.6.3) will be the first to be able to measure oxygen with light in addition to other components. The paramagnetic properties of oxygen (oxygen molecules are strongly attracted in a magnetic field) are still used for oxygen measurement, however with a declining trend. In practice, two methods are widely used:

- The magnetic torsion balance
- Thermomagnetic oxygen measurement

8.3.1 Oxygen analysers

In recent years, analysers operating on the torsion balance or also dumb bell principle have become increasingly popular.

The measuring effect is based solely on the magnetic forces of the oxygen molecule and is therefore comparatively less cross-sensitive to other gas components. However, the measuring chamber reacts sensitively to dirt and condensate.

The paramagnetism of oxygen reduces with increasing temperature. This thermomagnetic effect is used in so-called annual chamber or hot wire analysers for O_2 measurement. The advantage of these analysers is that the measuring chambers contain no sensitive internals, so that they are less sensitive to dirt and are also easier to clean than analysers that operate on the torsion balance principle. However, the measuring method is cross-sensitive to other gases and only functions accurately with a known or more constant accompanying gas composition.

8.4 Flame ionisation (FID)

This measuring method is used for measurement of the volatile content of unburnt hydrocarbons (VOC) (Volatile Organic Compounds). Organic carbon compounds contained in the sample gas are ionised in a hydrogen flame. The ion quantity is almost proportional to the number of carbon atoms involved in combustion. The ionic voltage is detected with the aid of an electrode and brought to indication via a high-resistance amplifier. Propane gas in nitrogen is normally used for calibration purposes.

Continuous measurements with the flame ionisation detector (FID) are increasingly prescribed, as soon as alternative fuels are burned. FID analysers are unfortunately susceptible to faults and require considerable maintenance by suitably trained personnel.

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The entire sample gas preparation system must not fall at any point below the sample gas temperature, i.e. the sampler, sample gas pipe and pump must be heated. The temperature in the ionisation chamber of the FID analyser is normally about 200°C.

Modern multi-component systems such as OPSIS are capable of measuring the majority of interesting VOC compounds as individual components. Programs for direct measurement of the volatile hydrocarbons (VOC) have also been developed. Initial tests have shown almost one hundred per cent correlation with FID values. This means that in countries where non-explicit FID measurements are required, an OPSIS or perhaps also some other type of multi-component analyser will preferably be used at a later time.

8.5 Solid-state electrolytic systems

8.5.1 Zirconium dioxide

Zirconium dioxide (ZrO_2) has the characteristic of building up a differential voltage at varying partial pressures and temperatures between 500 and 1000°C due to the flow of oxygen ions. This effect is used for the purpose of oxygen measurement by exposing both sides of a ZrO_2 membrane heated to about 800°C to varying oxygen concentrations (sample gas and reference gas). The supplied voltage is tapped and measured on the two surfaces of the membrane with thin, porous platinum electrodes. The measured potential difference increases exponentially with the oxygen concentration difference. To be noted is that the indicated measured values are based on gas in a moist condition.

In addition to the advantageous zirconium dioxide based in-situ measuring instruments, there are also extractive types that are hardly worth mentioning.

8.5.2 Other electrochemical measuring methods

Electrochemical measuring cells (EC cells) have the characteristic of supplying a voltage in the presence of certain components proportional to the concentration. As they are small and lightweight and require only a small number of peripheral devices, they are used mainly for portable measuring equipment (e.g. flue gas controls), although they had proved unsuitable for use in the cement industry in the past. However, in recent years, suitable EC cells have been developed that offer advantages mainly for oxygen measurement. The oxygen sensor may not be used when the accompanying gas contains H_2S, chlorine or fluorine containing compounds, as well as heavy metals and aerosols. The guaranteed useful life is three years. The many years of good experience prompted the majority of manufacturers to complement the NDIR analysers with EC cells for oxygen measurement.

8.6 Multi-component measuring systems

Development in the field of analysis has witnessed significant advances. Thanks to modern microprocessor technology, large computer capacities and high speeds, measuring methods are employed that enable disturbing secondary effects to be compensated by calculation and new techniques to be used, for example:

- Gas filter correlation technique (GFC)
- Fastfourier transformation technique (FTIR, FTUV) Differential Optical Absorption Spectroscopy (DOAS)
- Laser Diode Spectrometer (LDS), etc.

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Without going into details about the individual techniques, the following points are of importance:

- Multi-component analysers are less expensive for simple measuring tasks with more than one component than individual analysers in all respects.
- When using alternative fuels, the necessity may arise that new, particularly critical pollutant components have to be measured. This aspect justifies the use of flexible, programmable measuring instruments.

The suitability of multi-component measuring instruments in the cement industry has been demonstrated by many years of reliable use. In the following, three instruments will be presented that have proved reliable and are state of the art. These are both extractive and in-situ measuring instruments.

8.6.1 Sick GM 31

Sick, Waldkirch (D), is a pioneer of the design and construction of in-situ measuring systems. The instruments GM 21 for dust, and later, the GM 30 for NO SO_2 and dust, have been used successfully for more than ten years. The successor instrument, GM 31, differs considerably from its predecessors in two ways:

- Dust can no longer be measured with this instrument.

The flanged design as a so-called lance instrument does not measure the whole channel cross-section.

The GM 31 is designed for the components SO_2, NO, NH_3 and NO. A maximum of three of these components can also be measured simultaneously. Despite the lack of the benefit of dust measurement, the GM 31 is a low-cost emission measuring instrument, which complemented with a dust measuring instrument for systems not subject to particular requirements, meets all specific requirements.

8.6.2 OPSIS AR 620; ER 650

Within the Opsis measuring system, a light beam is projected on to a receiver and then passed through a glass fibre cable to the Opsis analyser.

In emission measurement, the light beam or measuring path continues through the interior of a stack or exhaust gas duct. Every gas absorbs light in known spectral regions of the total light spectrum, in a manner that is characteristic of the respective gas. This enables the analyser to detect and measure gases defined by the user. The results can be indicated in real-time or used for statistical and further processing purposes.

Opsis uses a scientifically researched principle for the identification and measurement of concentrations of various gases: Differential optical absorption spectroscopy (DOAS) which is based on the Lambert-Beer Law. It describes the correlation between the absorbed quantity of light and the number of molecules in the light path..

As every gas has its own unmistakable absorption spectrum, the so-called "finger print", the concentrations of several different gases can be determined simultaneously in the light path. With the DOAS method, a light beam is sent from a special light source - a Xenon high-pressure lamp - over a specific path; by means of complex, computer-aided calculations, the light losses through molecular absorption along the path are evaluated and analysed. The light of the Xenon lamp contains both the visible spectrum as well as the ultra-violet and infrared wavelengths. The light is detected by a receiver and passed on via a fibre optic cable to the analyser. This fibre optic cable makes it possible to install the analyser at a sufficient distance from any harmful environmental influences at the measuring point.

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The analyser consists, among other things, of a high-performance spectrometer, a computer and the associated control unit. The spectrometer breaks down the light with the aid of an optical grid in narrow wavelength bands. This optical grid can be adjusted for examining an optimal wavelength region.

The light is converted into electrical signals. A narrow slot moves at high speed across the detector; this combines a large number of instantaneous values that provide an image of the spectrum in the relevant wavelength region This scanning procedure is repeated a hundred times per second; the recorded spectrums are added up in the multi-channel memory of the evaluation unit until their evaluation.

Evaluation is carried out individually for each wavelength region and is based on the comparison of absorption curves. The respectively last recorded absorption spectrum is compared with a computer calculated spectrum. The calculated spectrum consists of a summation of the reference spectrums for the respective evaluation. The computer alters the size factors for each reference spectrum until optimal correspondence is achieved, so that the various gas concentrations can be calculated with high accuracy.

In order to be able to measure a diverse range of gas components as possible, the light spectrums must be divided into IR and UV. Measurement is similarly divided into a UV spectrometer and an IR interferometer. Depending on the components to be measured, both or one of the two instruments is used. Both instruments are usually necessary for comprehensive measurement of cement kiln exhaust gases.

The system is approved as a recognised emission measuring instrument worldwide.

For calibration: In principle, each component must be calibrated at last once (as part of the commissioning procedure). This takes place either via a convention method or by inserting cuvettes circulated by test gas in the light path. In this simple way, the work involved in the normally usual convention method is avoided. In two works of the ChB, these measuring instruments have proved highly reliable and may be referred to as standard-setting technology.

8.6.3 LDS 3000 diode laser of AltOptronic (Sweden)

The design is basically comparable with OPSIS. However, as a light source, a semiconductor laser is used. The LDS 3000 enables measurements to also be carried out in aggressive atmospheres with a high dust content; this takes place in a way that the special features of the diode laser are utilised in conjunction with a patented signal evaluation method. The instrument can distinguish whether a change in transmission is due to dust or a change in the gases to be measured.

The LDS 300 can be used for measurements in atmospheres with a varying dust content up to 50g/m3, depending on the size of the dust particles.

The light spectrum of the semiconductor laser can be adjusted to the absorption lines of the gas component to be measured via temperature and current. This ensures that measurement can only take place on a selected absorption line of the respective gas. The absorption lines of the remaining gas components are on other wavelengths and do not influence measurement.

In the three years in which the LDS has been used, continuous tests were carried out in order to determine the stability of the system. It emerged that no recalibration of either the zero or measuring range were necessary.

The diode laser is located in the main unit, from where the laser beam is passed via an optical fibre to the sensors at the measuring points. The distance between the instrument and the measuring point can be up to 1000m. In the main unit, the laser beam is scattered

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in an optical distributor and so enables simultaneous measurements on three different process measuring levels.

The short measuring time of the LDS 3000 provides direct information (t_{90}<1 sec). This is made possible by in-situ measurement, the high efficiency of the diode laser and appropriate signal evaluation. The short resonance time depends on the number of measuring points, as the laser light is scattered in a fibre-optic distributor and passed on to all measuring points simultaneously. Each measuring point can therefore function independently of the other measuring channels.

Thanks to its unusually short reaction time, this measuring system was to be able to considerably improve the CO monitoring of electrostatic filters. Measurement directly following WT would perhaps be possible, since the manufacturer mentions that up to 50g/m^3 dust has no influence. Unfortunately, the measuring distance this value applies to is unavailable.

The following gases can be measured at the present time:

Gas	ppm, at 1m	mg/m3, at 1m
NH_3	0.6	0.4
O_2	200	260
H_2O/NH_3	200/1	150/0.7
H_2O	0.3	0.2
Hcl	0.3	0.4
HF	0.3	0.4

Also other gas components can be measured on request.

The LDS 300 is one of the first laser analysers to appear on the market. At present, the price is (still) relatively high. Development is absolutely in line with the trend and could also be used advantageously in the cement industry in the near future.

8.6.4 Advance Cemas FTRI of Hartmann & Braun

Development has also continued with extractive measuring instruments. Hartmann and Braun builds in addition to the maximum four-component measuring NDIR systems a modern FTIR (Fastfourier Transformation Infrared) spectrometer, which has identical functions to those of the OPSIS, but is designed as an extractive measuring instrument. It offers high selectivity as well as easy upgrading to additional infrared components. Based on the measuring principle and automatic zero correction, calibration is only necessary twice a year to maintain the smallest measuring ranges. All parts coming into contact with the sample gas: Sampling system, sample gas preparation system and measuring cuvette are heated to 100°C. The instrument features a self-diagnosis system and, as with all modern instruments of this type, can be monitored by H & B Service and faults diagnosed via an integrated modem.

A German cement works has had good experience over a two-year operating period. The instrument is overhauled twice a year as part of a maintenance contract. The weak point, the hot gas pump, is changed each time or the diaphragms replaced as a precautionary measure.

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9. RECORDING AND EVALUATION

Following the measured values on the basis of recording strip charts is advantageous for the general assessment of a process sequence. In the area of emissions, the majority of countries require that measured values are prepared according to certain, adapted criteria; hourly and daily average value, criteria relating to the observance or non-observance of limiting values, etc. In such cases, digital processing of the measured values is unavoidable.

9.1 Trend curves, recorders

For the purpose of clarity, not too many components should be recorded simultaneously. Strongly varying signals must be appropriately dampened. Monitors or line recorders are better suited for this purpose than dotted-line recorders. Scaling should correspond with the actual physical values. This applies particularly to life-zero signals. For strip chart recorders, a paper feed rate of 20mm/h is standard.

9.2 Averaging computers

When lump fuel (e.g. tyres) is fed into the precalcining zone, the measured values at the kiln inlet are subject to extreme fluctuations. Without damping of averaging, the signals are difficult to interpret. It has emerged that sliding, linear averaging is easier to interpret than logarithmic damping, as individual values are less important in linear averaging. The more favourable intervals for optimal averaging are between 10 and 15 minutes. Figures 15 and 16 show the behaviour of the signals of a 1500-7/d kiln with tyre feed without averaging and with various averaging times.

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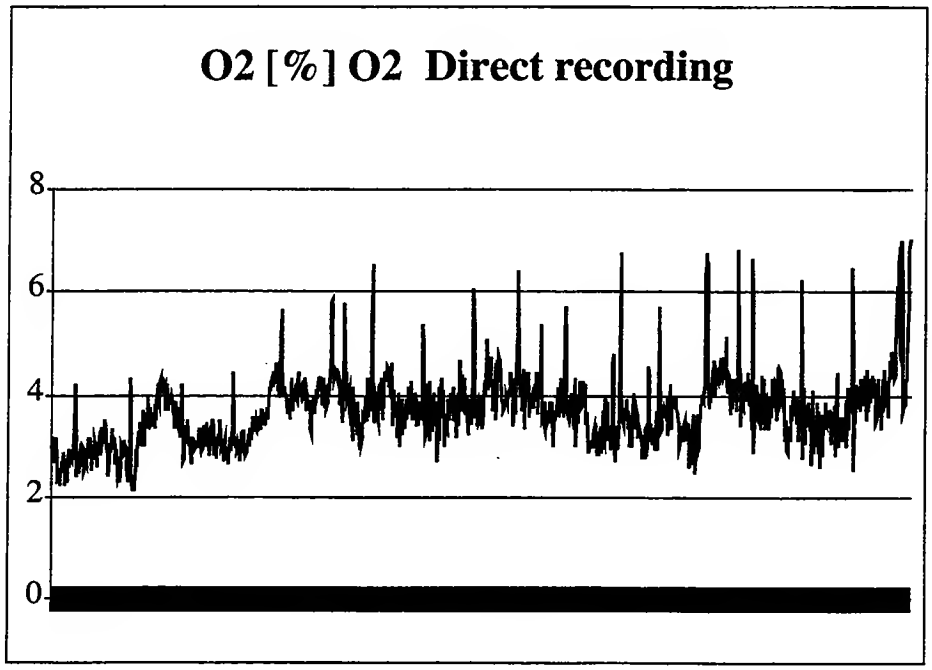
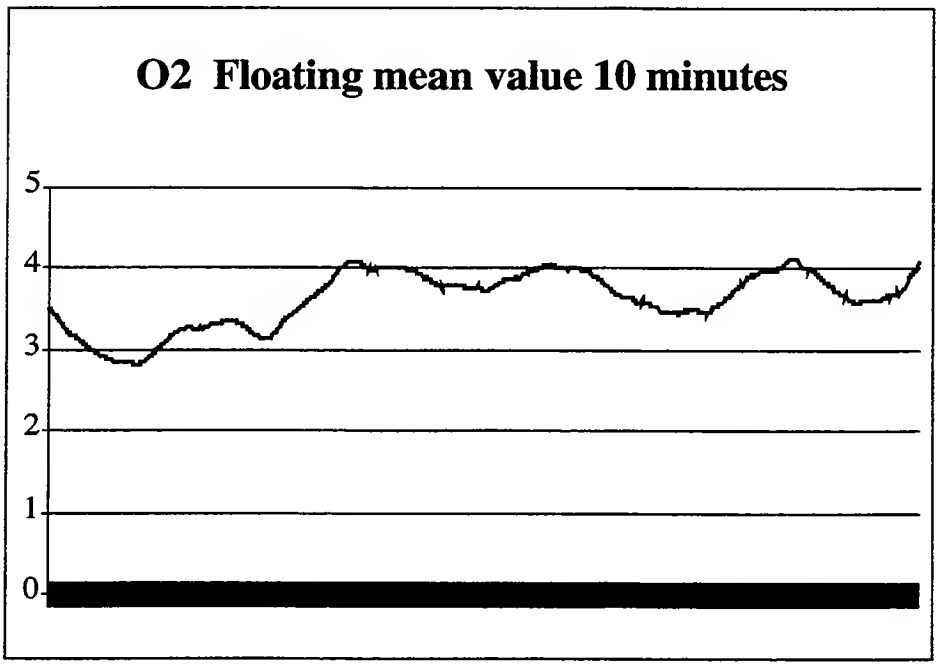
Figure 15:


O2 [%] O2 Direct recording
8
6
4
2
0


O2 Floating mean value 10 minutes
5
4
3
2
1
0

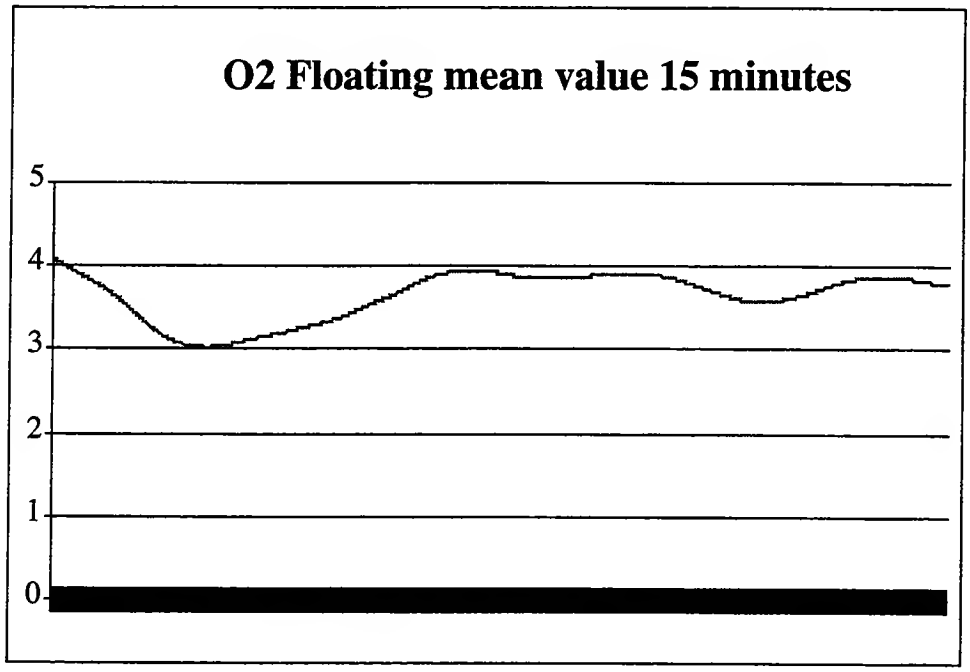
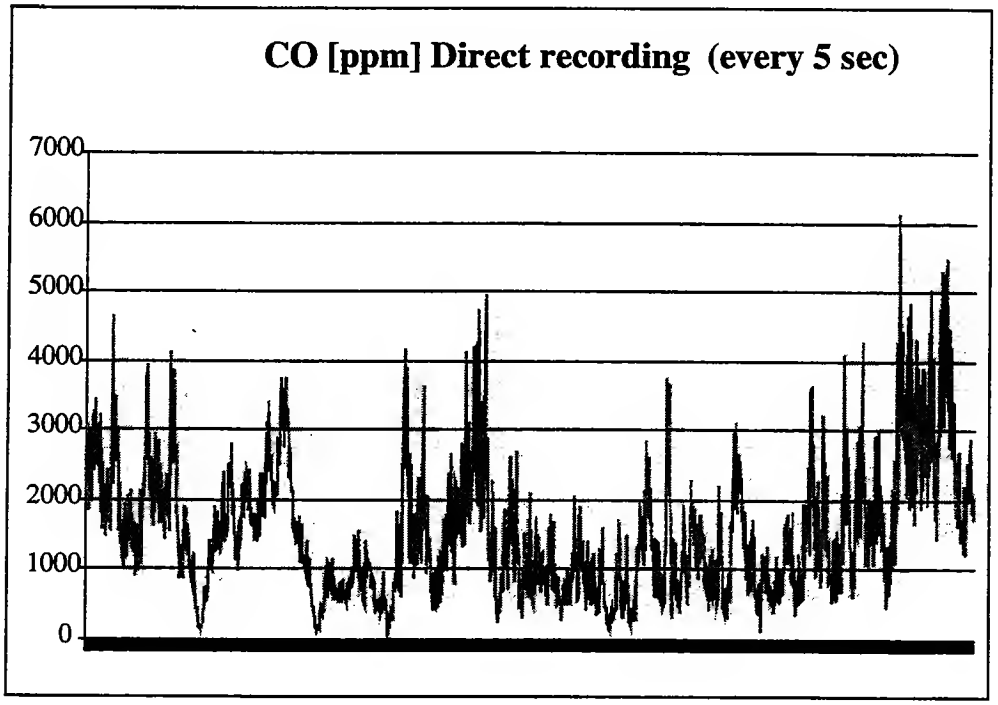


Figure 16:



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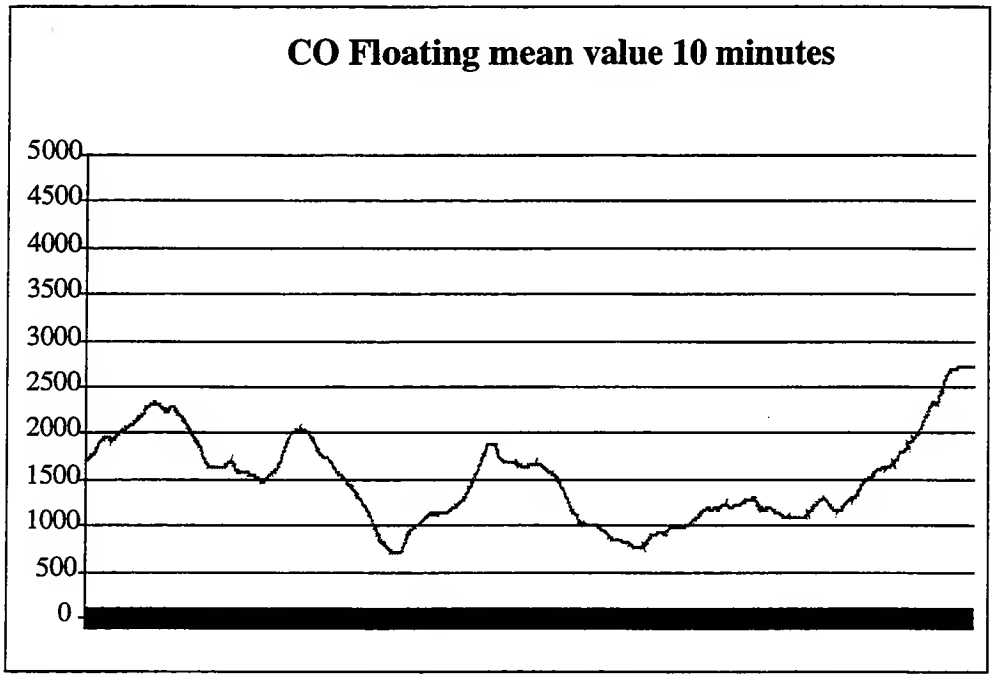

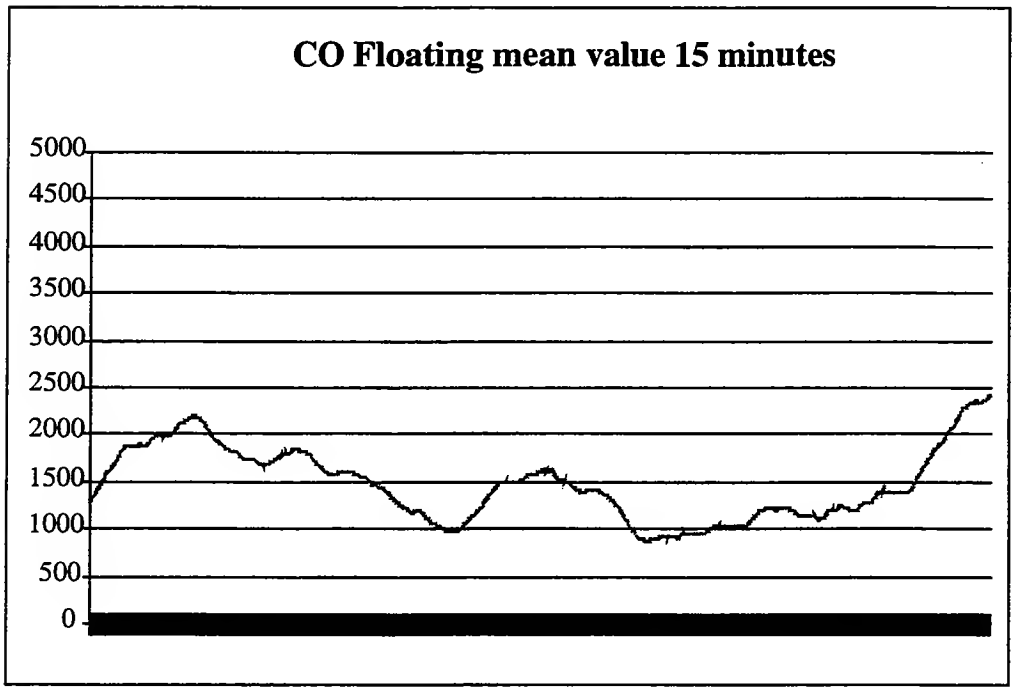

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9.3 Data logging

The interpretation of measured signals on the basis of trend curves is too rudimentary for sophisticated tasks. Visual comparisons of such curves easily lead to misinterpretations. In connection with optimisations or in the search for faults in the process, the signals from gas analysis must be processed with other measured quantities from the system in the PC. A precondition for this is that the data can be digitally acquired, logged and input into the PC. In plants with modern process control systems, this precondition is usually met. Where this is not the case or as part of temporary measurements, independent data logging systems should be used for data recording purposes.

9.4 Emission computers

The form of documentation of the measured emissions is prescribed in more and more countries. In Europe, there are plants whose emission computers are connected via a data line to the authorities concerned. The emission computers are normally designed for documentation of the following values:

- Half-hourly average values
- Hourly average values
- Daily average values
- Exceeded limiting values

Emission computers primarily meet the officially prescribed evaluation and documentation of emission values. As the emission data also contains valuable process-relevant information, they are also useful for the operator. Depending on the regulations of the country concerned, the valuation of emission values can be taken over from internal process control systems.

10. MAINTENANCE AND QUALITY ASSURANCE

Reliable continuous operation of continuous gas analysis is dependent not only on the efficient design of the measuring equipment, but also on systematic maintenance and repair. The maintenance work to be carried out can be subdivided into the following categories.

- Condition check
- Adjustment check, cleaning
- Replacement of wearing parts
- Function test

Table 4 shows the maintenance intervals of a typical gas analyser. Maintenance must be carried out daily (D), weekly (W), monthly (M), quarter yearly (QY), half yearly (HY) or yearly (Y). The time cycle of the individual working procedures depends on the system-specific conditions and essentially on the age of the measuring equipment. The work undertaken is noted in a logbook for practical purposes.

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Table 4: Maintenance Intervals

	Visual checks	Adjustments, checks, cleaning works	Replacement of worn parts	Function Test
Sample gas preparation:				
Flow measurement (floating body instrument)	D	W		
Suction pressure (Manometer)	D	M		
Humidity in the sample gas pipes	D			
Function check of the wetness monitor				
Heated sample gas pipe temperature		HY		HY
Sample gas cooler condensate pump	D		HY	
Sample gas pump check and replace membrane			Y	
Tightness check		QY		Y
Cabinet air conditioning	D			Y
Magnetic valves, function and tightness check		HY	Y	
Cabinet cleaning		QY		Y
Probes:				
Tightness check		QY		
Cooling	D			
Filter change			HY	
Cleaning			HY	
Analyzers:				QY
Indication plausibility	D			
Adjustement		W		
Filter change			QY	
Total overhaul		3Y		

The operating conditions and type of construction of the equipment used varies considerably. Instructions for maintenance and repairs must be adapted to local conditions and defined according to individual experience.

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10.1 Visual checks

The term visual check includes visual inspection of the equipment, indicating instruments, signal states and checking heated parts for a hot or cold condition by trained and experienced personnel.

10.2 Adjustment checks, cleaning

This category includes adjustments, readjustments, etc., of conditions which can change for different reasons.

10.2.1 Tightness checks

Vacuum check

The hose connections at the probe are sealed and the sample gas pump is started. The maximum vacuum which the pump can produce must be noted when commissioning the measuring equipment. At maximum vacuum, the part of the pump is closed and the pump stopped. The sample gas preparation system up to the sample gas pump (suction area) is considered tight when the vacuum does not change significantly for a period of 5 minutes following temperature equalisation.

Nitrogen check

See Chapter 7.7

10.3 Replacement of wearing parts

When replacing parts, it must be ensured that no leaks occur. According to experience, weak points in this respect are filter housings and connecting points in the sample gas pipe. Suitable sealing material must be available at all times, so that any type of seal, union, etc., can be replaced if there is the slightest suspicion of a leak.

10.3.1 Sample gas pump

The sample gas pumps are usually diaphragm pumps, whose diaphragm and valve plates should be replaced at least once a year.

The ball bearings of the motor and eccentric should be replaced every two years in continuous operation. Standby sample gas pumps should be available.

10.3.2 Cleaning sintered metal and ceramic filters

Ceramic filters can be cleaned with diluted hydrochloric acid (1 part concentrated hydrochloric acid to 10 parts water). The filters must be placed in the acid for about 2 hours.

The parts must subsequently be flushed with water pressure from the inside to the outside, blown out with compressed air and dried in a drying cabinet at about 100°C.

10.3.3 Fine filters

The filter element should be replaced at regular intervals, e.g. every three months. Fine filters with condensate monitor are tested by moistening the filter element. The instrument must then respond.

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10.3.4 Gas analysers

The gas analysers are tested with test gases or by means of the built-in calibrating cuvettes and readjusted if necessary. For testing with gases, it must be ensured that the flow rate roughly corresponds with the operating values. When selecting the test gases, the specifications of the instrument manufacturer must be observed. The gas analysers should be adjusted at regular intervals, at least every two weeks. Adjustments must be noted in the maintenance report. If significant adjustments are required, or the adjustment range limits (greater than 80% or less than 20%) are reached, the instrument must be tested and inspected if necessary.

Wearing parts are, e.g. emitter, receiver, diaphragm motor.

For zero readjustment, nitrogen is normally used. With the exception of oxygen measurement, air can also be used as a zero gas if the analysers are not too sensitive.

For sensitivity adjustment, a test gas is used that normally consists of the measuring components and nitrogen. For compensation of the residual moisture, it is advantageous to feed the test gas upstream of the sample gas cooler. The test gas concentration for sensitivity adjustment should be 80 to 90% of the respective measuring range end value.

10.4 Function test

This term means that certain functions gas analysis are extensively tested, e.g. to ensure that the detector in the fine filter responds if moisture is present and the sample gas pump is stopped.

10.5 Test gases

As initially mentioned, new gas analysers no longer require any test gases. The internal calibrating and control options ensure that the accuracy is maintained over a long period (normally two years). After this time, the analysers should be tested by the manufacturer's service department and in particular the internal control devices (calibrating cuvettes) checked. Test gases are available in various accuracy classes according to particular directives, e.g. VDI 3490, sheet 2. Depending on the manufacturing and analysis accuracy, distinction is made between three classes. The test gases are delivered with analysis certificates that contain all essential data, such as:

- Measuring component and accompanying gas
- Test gas production method
- Measuring component concentration
- Relative error of this concentration
- Pressure of container filling
- Minimum application pressure
- Minimum and maximum storage temperature and maximum test gas storage time (limit date)

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The concentration specified on the analysis certificate should be written in large and easily readable letters on the test gas bottle.

The test gases are available in bottles of varying size. A 10 litre bottle (about 1100mm high, preferably of aluminium) is normally sufficient for a two-year supply. The bottle pressure is between 100 and 200 bar, depending on the gas composition. The main valve must be closed after each use. Test gas deliveries max take time in certain regions; this should be taken into account when the bottle pressure starts to get low.

10.5.1 Mixing test gases

Certain gas components can be combined as a mixture in a bottle. This possibility considerably facilitates maintenance (fewer bottle transports and savings on gas bottle hire charges). Caution! New multi-component gas analysers must only be tested with gas mixtures but not calibrated. Only individual components in nitrogen are permitted for calibration.

With one exception, all standard components can be mixed together. This exception applies to oxygen O_2.

With oxygen analysers, the measuring range should be such that air can be used for calibration. Nitrogen must always be used as a residual gas.

Example of a test gas mixture:

Component	Chemical symbol	Analyser measuring range	Concentration in test gas
Sulphur dioxide	SO_2	3000ppm	2800ppm
Carbon monoxide	CO	5%	0.45%
Nitrogen monoxide	NO	2000ppm	1900ppm
Carbon dioxide	CO_2	as accompanying gas	25%
Nitrogen	N2	as residual component	74.08%

The accuracy of the test gases reduces the more components are mixed. The number of mixing components is thus limited.

Further, general information on test gases is provided in the annexed instructions.

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10.6 Fault signals

Faults in the measuring instrument should be indicated by status signals, e.g.:

- Insufficient pressure in the suction pipe (blockage of the probe or gas path)
- Excessive pressure in the suction pipe (possible failure or leakage of a solenoid valve)
- Status signals of the analysers (failure of the analyser or electrical fault)
- Operating temperature of the sample gas cooler or failure of the sample gas cooler
- Operation of the condensate monitor (can indicate failure of the sample gas cooler or faulty water-cooled probe
- Adjustment/operation changeover valve should automatically inform the control room that the measuring equipment is being serviced and is therefore not ready for operation
- Minimum contact of flowmeter (insufficient sample gas)

Status signals for measuring equipment faults are checked for their plausibility by simulated operating deviations.

10.7 Automated maintenance equipment

Theoretically, numerous maintenance procedures can be automated, e.g. automatic zero and limit range adjustment, sample gas flow rate control, blowing out of the filter, etc. However, sample gases are media that can give rise to various difficulties.

They have a corrosive effect, contain dust, must not be adulterated, can be very moist, etc., to mention but a few of the unpleasant characteristics that make automation difficult.

Practice shows time and again that solenoid valves are the weak point within measuring equipment, so that only valves of the best quality must be used for evaluation. The number of necessary solenoid valves increases with an increasing degree of automation; due to their susceptibility to faults, the failure rate also increases, so that the savings expected from automation are accordingly lost.

The same applies when a group of analysers is switched to two or more measuring points with the intention of saving on expensive analysers. Apart from the operating difficulties, experience has shown time and again that the saved investment costs are quickly absorbed by the corresponding maintenance costs.

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11. MEASURING ERRORS

The accuracy of gas analysers is often overestimated. The reason for gas analyses being relatively inaccurate is not because of the analysers, as is often assumed. The following examples explain the most important influences responsible for this.

11.1 Sample gas sampling

Significant measuring errors can occur as a result of non-representative sampling. In rotary kiln systems, these occur particularly at the rotary kiln inlet and kiln inlet chamber measuring points. These measuring errors occur mainly at the "filter" measuring points in coal pulverising mills.

11.2 Sample gas preparation

11.2.1 Sorption and chemical reaction

In the probe and sample gas preparation system, sample gas components can react with each other. These reactions are reinforced by catalytic effects. Furthermore, reactions with dust or gas conducting components are possible.

11.2.2 CO reduction

At high temperatures (above 600°C), CO is reduced to a large degree depending on the residence time. This effect is the reason why only cooled probes may be used at temperatures above 500°C.

11.2.3 NO_2 formation in the probe and sample gas preparation system

At temperatures of about 100°C and a concentration of 900ppm NO, the theoretical formation rate of NO_2 is about 0.5ppm per second.

At lower temperatures and high gas residence times, this formation takes place more rapidly. The indication delay of the NO measuring equipment should therefore be as short as possible.

11.2.4 SO_2 reduction in the probe and sample gas pipe

Rust, dust deposits, non-ferrous metal parts (especially copper) in the gas path of the sample gas can reduce SO_2 up to 100%. For the sample gas pipes and all other gas conducting parts, Teflon, high-alloyed steel or steel with special coating (e.g. PTFE) must therefore be used.

11.2.5 SO_2 reduction of the filter dust of the sampling probe

SO_2 reacts with the $CaCO_3$ and CaO contained in the filter dust of the sampling probe with the formation of $CaSO_4$. With high dust contents in the process gas, measuring errors of up to 100% can occur. SO_2 can therefore only be measured fairly accurately process gas with a low dust content. This applies in particular to SO_2 measurements in the rotary kiln inlet. Tests have shown that SO_2 is initially indicated only a short time after cleaning the filter. Within several minutes, the indication falls to zero again, because the SO_2 is absorbed in the dust building up in the filter.

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11.2.6 Interaction with a liquid

Errors due to the solubility of the measuring component

The solubility of certain measuring components in water must be taken into account for wet sampling probes. If the injector water of the wet sampling probe is not fed into the circuit, CO_2, NO_2 and SO_2 can go into solution, depending on the probe design.

Condensation of the water vapour which is separated as water in the sample gas cooler by the extractive method, can also distort gas analysis. This is of particular significance for SO_2 emission measurement when low concentrations are present. The theoretical solubility of SO_2 in water is low and could be compensated by appropriate calibration. However, tests on cement kiln systems in a direct operating mode have shown that measuring errors for SO_2 are far above the theoretical distribution equilibrium when SO_2 reacts with other gas components such as NH_3. An SO_2 measuring device can be tested for these errors by the sample gas being transferred from the sampling probe directly into concentrated sulphuric acid (H_2SO_4) as the drying medium and not via the sample gas cooler. If distinct differences in the measured value indication occur, a chemical reaction is most probably taking place in the sample gas cooler.

Apart from SO_2, NO_2 also dissolves in the condensate of the sample gas cooler. However, since the proportion of NO_2 in the total NO concentration can be disregarded with proper sampling, the resulting measuring error can normally be disregarded.

11.3 Volumetric errors due to the solubility of accompanying components

If accompanying components, e.g. CO_2 go into solution, the concentration of the other measuring components increases. The partial absorption of CO_2 in an injector probe, increases, e.g. the O_2 concentration.

11.4 Gas analysers

The accuracy of the gas analysers, provided they are in a perfect condition, is not an issue. If a gas analyser is viewed as a whole, the analyser is the most accurate link in the measuring chain, with a few exceptions. The analyser manufacturer's specifications relating to cross-sensitivities, disturbing components as well as temperature and pressure influences must be observed.

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12. SPECIAL FUNCTIONS OF GAS ANALYSIS

12.1 CO monitoring for protection of electrostatic filter

If, for any reason, combustible gases should develop (ring fractures, faults in fuel dosage, etc.), in the kiln system, there is a risk of a potentially explosive mixture forming in the electrostatic filter. High-voltage discharges inside the filter cause sparks, which can lead to explosions in a gas atmosphere of appropriate composition.

As a safety measure, the CO component of the gas discharging from the kiln system is measured. Once a certain level is reached, the high-voltage should be switched off before the gases reach the electrostatic filter. Measurement can initially take place in precalcining plants following complete combustion in the auxiliary firing equipment, i.e. in the uppermost cyclone stage of the heat exchanger at the earliest. The residence time of the gases from this point to the electrostatic filter is relatively short (several seconds) and varies considerably from plant to plant (with or without cooling tower, combined/direct operation, etc.).

Filter monitors, whose effect alone depends on an adjustable maximum value, reassure responsible personnel, however, they are not optimal solutions. On the one hand, unnecessary filter shutdowns are initiated, on the other hand, it is not ensured that they would operate promptly in an emergency. A tried and tested possibility of optimising this process is offered by sequential filter shutdown.

12.1.1 Basic concept of filter shutdown

Compared with the usual method, filter shutdown is optimised in two essential steps. The first step consists of shutting down the filter chambers step by step. The second step consists of taking into account a "reducing" or "increasing" CO trend.

The residence time of the gases in the electrostatic filter itself is used in order to further delay shutdown. The filter chambers are shutdown at approximately five second intervals, depending on the size and gas flow rate. However, as soon as CO measurement indicates a reducing trend, the shutdown cycle is stopped.

12.1.2 Optimisation of CO measurement

The following measures must be taken in order to make CO measurement as efficient as possible:

- Short distance between sampling point and analyser
- Large sample gas volume (up to 300l/h setting;short delay time)
- Small volumes in gas path, particularly in sample gas cooler
- Select analyser with fast response time
- Use double measuring system for two-section WT systems
- Formation of logical signals "CO increasing" and "CO reducing"

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12.1.3 Specifications

- Selection of measuring point according to Figure 1
- Gas preparation system appropriately adapted according to Figure 12
- NDIR gas analyser, measuring range 0 to 5%, CO with 3 adjustable maximum limiting values
- Arrangement of a shutdown procedure

With regard to the CO analyser, it should be noted that the same must have a high measuring range for CO monitoring. It cannot be used simultaneously for continuous CO observation, as the normal CO content is below the detection limit of this analyser. If CO is to be continuously measured, a second analyser with a much lower measuring range of 5000ppm (0.5%) must be used.

12.1.4 Shutdown procedure

The procedure for shutdown of the individual filter chambers is shown on the basis of two examples in Tables 5 and 6. The specified numerical values of the shutdown sequences and the CO level are only guide values and must be adjusted to the respective plants.

For CO values below 10%, a risk of explosion is virtually excluded. However, the threshold values must be set far lower, as the measurement reacts with delays despite optimal preconditions (dead time + indication delay).

Table 5: Switching procedure, for a simple heat exchanger kiln with 3-changer filter

Time-point	CO - Analyzer switching threshold			Trend	Alarm-Stage	Filter chamber			Fuel	
Sec.	0,8%	1,5%	4,8%			1	2	3		
X	0	0	0	+-	0	1	1	1	1	
X	1	0	0	+-	1	1	1	1	1	
5	1	1	0	+	2	0	1	1	1	
10	1	1	0	-	1	1	1	1	1	
10	1	1	0	+	3	0	0	1	1	
10	1	1	1	+	5	0	0	0	0	
15	1	1	0	-	1	1	1	1	1*	
15	1	1	0	+	4	0	0	0	1	
15	1	1	1	+	5	0	0	0	0	

1 = "On"
0 = "Off"

* Check the purging time befor switching the filter on again

Table 6: Switch-off pocedure, for a two strings heat exchanger kiln with 3-chamber filter

Time-point	CO - Analyzer 1 switch. threshold			CO - Analyzer 1 switch. threshold			Trend		Alarm-Stage	Filter chamber			Fuel
Sec.	0,8%	1,5%	4,8%	0,8%	1,5%	4,8%	1	2		1	2	3	
X	0	0	0	0	0	0	+-	+-	0	1	1	1	1
X	1	0	0	0	0	0	+-	+-	1	1	1	1	1
X	1	0	0	1	0	0	+-	+-	1	1	1	1	1
X	1	1	0	1	0	0	+-	+-	1	1	1	1	1
5	1	1	0	1	1	0	+	+	2	0	1	1	1
5	1	1	0	1	1	0	-	-	1	1	1	1	1
5	1	1	1	1	1	1	+	-	2	0	1	1	1
5	1	1	1	1	1	0	+	+	3	0	0	1	1
10	1	1	0	1	1	0	-	-	1	1	1	1	1
10	1	1	0	1	1	0	+	+	3	0	0	1	1
10	1	1	1	1	1	1	+	+	5	0	0	0	0
10	1	1	0	1	1	0	-	-	1	1	1	1	1*
15	1	1	0	1	1	0	+	+	4	0	0	0	1
15	1	1	0	1	1	0	-	-	1	1	1	1	1
15	1	1	1	1	1	1	+	+	5	0	0	0	0

1 = "On"
0 = "Off"

* Check the purging time befor switching the filter on again

12.1.5 Special devices

Sick GM 950

Via an in-situ sensor (zirconium dioxide) CO is measured at a high response rate and processed to a shutdown command if required. As zirconium is fairly instable as a CO indicator, extractive supplied semiconductor sensors correct any deviations.

The device is new and still in the testing phase. The currently available test results have been positive up to now. The measuring method is relatively elaborate and the device gives the impression of being somewhat complex. During the test, no functional disturbance occurred.

12.1.6 Laser analysers

In-situ laser analysers offer promising prospects as already mentioned in chapter 8.6.3. It is conceivable that low-cost options will soon be available, as several measuring points can be served with the same analyser, e.g. filter monitoring (upstream and downstream of the filter) and various points in the coal pulverising mill.

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12.2 Monitoring of coal pulverising mills

In the safety concept of coal pulverising mills, the monitoring of critical gas components covers only a small part of the necessary equipment and measures. In addition, coal pulverising mills vary considerably in their design, so that the application and function of gas analysis is always determined as part of the entire plant.

12.2.1 Limiting values

In 1984, the following limiting values were listed by a working group of the Association of German Cement Works (VDZ):

O_2 content in moist exhaust gas downstream of the filter	14%
Temperature at mill inlet (coal moisture 12%)	400°C
Gas temperature downstream of mill upstream of filter	120°C
Gas temperature downstream of filter	120°C
Gas temperature upstream of filter minimum 30°C above dew point about	75°C
CO concentrate during filter shutdown	50ppm
Operation with hot gas generator	70ppm
Operation with kiln exhaust gase	not practical
Pressure upstream of mill	-1.5bar
Coal dust temperature	110°C
Air temperature of pneumatic transport	80°C

The limiting value for the oxygen concentration applies only to plants with inert gas operation. All other limiting values are the same for the safe operation of inert and non-inert gas operated plants. For coal pulverising mills heated with kiln exhaust gas, CO monitoring is not practical during operation, because the gases from kiln operation contain up to 10 times more CO than the limiting value of 70ppm permits.

CO shutdowns of the electrostatic filter for the kiln, must be coupled with the filter of the coal pulverising mill.

This information on the safety equipment of coal pulverising mills only contains the most important information and is intended to draw attention to the complexity and importance of such plants.

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13. CONVERSION FACTORS

Conversion of the gas components from volume to units of weight and vice versa, is often the cause of errors in adjustment or interpretation. Such errors occur most frequently in connection with NO and NO_2. The problem with these NO components is that the instruments measure NO, but the emission must be converted to NO_2.

Furthermore, emissions are considered in weight-related units (mg/m^3), but the instruments and the test gases are often specified in volume-related units (e.g. ppm or %)

Conversion from [ppm] to mg/m^3 Factor $f = \frac{M}{22.4}$

Whereby M = Mol mass (kg/kmol) of the gas components

Table 7

Components	Converted to	Mol mass [M]	Factor (f)	1/f
CO	CO	28.01	1.25	0.80
NO	NO	30.01	1.34	0.75
NO	NO_2	46.01	2.85	0.49
NH_3	NH_3	17.03	0.76	1.32

Example

Given:

- Test gas with 8.75ppm NO
- NO analyser, measuring range 0-2000mg/m^3NO

Required:

- Analyser indication at 875ppm
- How much NO_2 in mg/m^3 corresponds to the adjusted value 875ppmNO?

Solution:

- Analyser indication = 875 x 1.34 = 1172mg/m^3 NO
- Emission value NO_2= 875 x 2.05 = 1794mg/m^3 NO_2
- or from mg/m^3 NO to mg/m^3 NO_2 1172 x 1.53 = 1794mg/m^3 NO_2

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Chapter 5

High Level Control

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HIGH LEVEL CONTROL SYSTEMS

U. Haberstich
PT 98/14350/E (Revision of PT 94/4191/E)

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Summary

For more than 30 years, the cement industry has been exploring computer-based techniques to control and optimize the operation of cement kilns. The major reasons behind these endeavours are clinker uniformity, savings in energy consumption, increase in production, savings in refractory consumption and NOx reduction. Basically, cement kilns are difficult to control because of their non-linear, multivariable, behaviour and the poor quality of the available process signals. After several trials to describe the burning process with mathematical models a new approach in cement kiln control was investigated in 1980. Operator control strategies were studied, and a detailed record of the operator's behaviour was made while controlling the kiln. These "fuzzy" rules can imitate multivariable control actions and can combine information from variables to identify the kiln conditions. Within the "Holderbank" group, specific requirements of such a High Level Control (HLC) system was defined to compare the several suppliers. The system from ABB LINKman is the most powerful real-time expert system with very advanced features and a high user-friendliness. A standard implementation plan was made to reach a successful application of the system. The preparatory work in the plant must be carried out according to the recommendations made during the pre-project study. The identified process problems have to be solved. To achieve the "Best Operator Performance" the detailed and rigorous monitoring of the HLC performance is obviously essential at this stage. The optimization of the process with respect to product quality and related process factors, on a long-term basis, is the key to the ultimate level of savings arising from the HLC application. It is this optimization that can give the plant additional benefits over and above those arising from the average operator to „Best Operator Performance". The experience shows that substantial benefits like higher kiln output, lower heat consumption, longer refractory life, lower NOx emissions and lower standard deviation of the key variables can be expected from HLC systems, if they are properly implemented and used. The experience shows that the payback of 1 to 2 years is realistic, when considering only the investment costs in the HLC system.

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1. INTRODUCTION

For more than 30 years, the cement industry has been exploring computer-based techniques to control and optimize the operation of cement kilns. The major reasons behind these endavors are clinker uniformity, savings in energy consumption, increase in production, savings in refractory consumption, NOx reduction (Fig. 1 + 2, Annexes 1 + 2). The still high energy consumption of the cement manufacturing process, the stringent requirements on cement quality and the environmental aspects which are leading the governments to apply severe legislation regarding the emissions.

The classical process automation approach, consisting in defining a mathematical model of the process, led to only a very few successful kiln control applications. The improvement in the performance of hardware equipment, combined with the advent of artificial intelligence, is leading to a major step toward kiln control and optimization.

2. EVOLUTION OF HLC SYSTEMS

Basically cement kilns are difficult to control because of their non-linear, multivariable, behaviour and the poor quality of the available process signals. The control is then usually limited to a few secondary measurement loops, whereas the control of the primary parameters and the operating conditions are the responsibility of the kiln operators.

The first applied techniques were based on empirical or mathematical models. Although successful in simulating the kiln operation, these techniques were generally based on too many assumptions and used very complex theoretical models. For this reason they were not applicable and could not be extended to a broad selection of kilns. Other approaches such as the "hill climbing" techniques (Young et al, 1971) or statistical identification combined with optimal controller design by the state space method (Otomo et al, 1972) was also used but did not achieve any significant success.

Since the mid-seventies, a new approach based on the analysis of the human decision making in cement kiln control has been investigated (Umbers and King, 1980). Operator control strategies were studied, and a detailed record of the operator's behaviour was made while controlling the kiln. Basically this approach rests upon the concept of fuzzy logic introduced by Prof. L. Zadeh in 1965. The basic operator control rules were already prescribed by Peray and Waddell (1972). These "fuzzy" rules can imitate multivariable control actions and can combine information from variables, they work by identifying the kiln conditions and prescribing suitable corrective actions.

FL Smidth supplied the first commercially available kiln control system based on fuzzy logic in 1980. The concept of High Level Control was introduced at that time and is used to refer to systems, which provide not only supervisory control but also optimising control. Since then, many systems have been developed and are marketed. Some are using the concept of fuzzy logic and are called rule-based systems or are based on expert system shells. Others are more conventional and apply PID control or adaptive-predictive controllers. The penetration of HLC in the cement industry has been very intensive over the last decade, about 300 applications have been reported in kiln control applications.

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3. SPECIFIC REQUIREMENTS OF A HLC SYSTEM

The following basic requirements were specified:

1) The provision of a high degree of user-friendliness; this aspect is extremely important, since kiln control strategy needs to be adapted when process conditions change. The maintenance of the application control strategy must be easy for the plant engineers to carry out.
2) The concept of "autopilot" as used in the previous version of the supplier's HLC system must be included to make it possible for the operator to switch the system on-line or off-line at any time without disturbing the process.
3) The use of a toolkit based on the G2 expert system shell which provides advanced features such as real-time facility, graphical interface and object-oriented programming; the toolkit, which is a software layer between G2 and the HLC applications, is the support for developing and implementing control strategies without programming skills.
4) The possibility for having multiple applications on the same system, typically one kiln, one cooler, mills and the kiln simulator.
5) The provision of a system incorporating tools and facilities allowing for consistent process optimization.
6) Very helpful for the introduction is the inclusion of a standard interface between the HLC system and the tailor-made kiln simulator in order to provide a training platform for the operators; this platform would allow for the simulation of kiln upsets and disturbances. The plant engineer would thus be able to develop and test new strategies before real implementation in the actual plant application. Figure 3 (Annex 3) shows a typical configuration.
7) The standardisation of the control strategies, in order to shorten commissioning time and make exchange of experience easier between different users. It is important to have a standardised way of both configuring and maintaining the control strategies.

Within the "HOLDERBANK" Group **LINKman** from ABB LINKman Systems Ltd, London, is applied. LINKman Graphic, the new version of this HLC system, is based on G2, a very powerful real-time expert system shell from Gensym Corp. LINKman Graphic is particularly user-friendly and offers very advanced features [4].

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4. PROCESS OPTIMIZATION WITH HLC

As mentioned above, process optimization is the major target to be achieved. There are typically three phases associated with our concept of successful application. See also [3]. These phases are outlined below.

Phase 1: Plant Preparation

The preparatory work in the plant must be carried out according to the recommendations made during the pre-project study.

Phase 2: Achievement of „Best Operator Performance"

Very often bottlenecks are detected, process problems are identified and experience is gained in the period immediately following HLC implementation. Detailed and rigorous monitoring of the HLC performance is obviously essential at this stage. The identified problems then have to be addressed during the secondary commissioning. Attempts must be made to achieve the highest possible run time of the HLC system, typically 90 % or more, using an adequate and consistent control strategy. At this stage, the HLC is expected to operate with the same performance as the best operator with respect to production output, product quality, heat consumption, etc. The associated benefits depend on the size of the plant and the performance previously achieved in manual operation. This phase, based on the best operator know-how, can be considered as the foundation of the whole HLC project.

Phase 3: Process Optimization

The optimization of the process with respect to product quality and related process factors, on a long-term basis, is the key to the ultimate level of savings arising from the HLC application. It is this optimization that can give the plant additional benefits over and above those arising from the average operator to „Best Operator Performance". Process optimization mainly involves the plant technical staff. It requires an evaluation of the process performance, an estimation of the potential savings, which can be achieved, and an assessment of the control strategy performance. Since plant conditions change in respect to raw material quality, availability of alternative fuels, product quality requirements, etc., process optimization must be considered as a permanent task.

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5. PRINCIPLES OF OPERATION

Basically, in the case of a conventional preheater kiln, HLC manages the control of the following parameters:

- the kiln feed rate
- the kiln speed
- the IDF speed (or damper position)
- the fuel to the kiln burner

In case of a precalciner installation, the precalciner fuel and the position of the tertiary air damper (AS system) have also to be controlled automatically.

The HLC system needs to access the process relevant data, such as preheater temperatures and pressures, the gas composition, the kiln amperage, the burning zone temperature, etc.

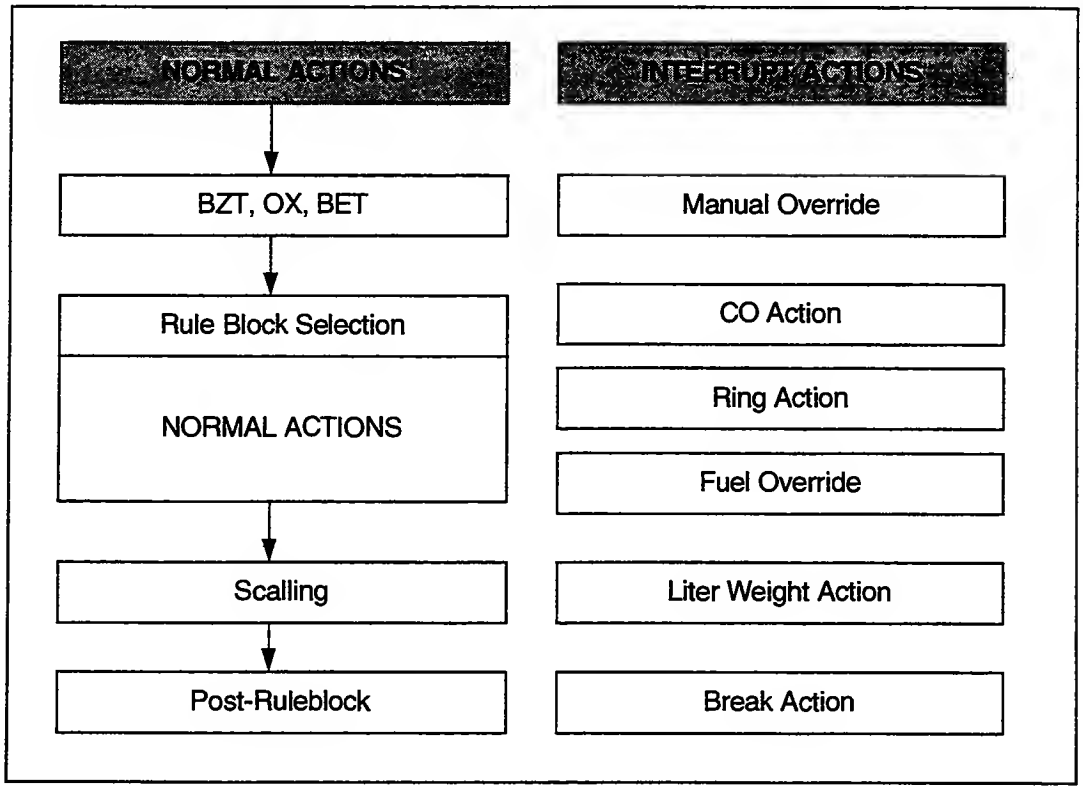
The principles of kiln control operation depend on the HLC system used. The principles used in LINKman are hereafter presented.

LINKman

LINKman works in two basic modes:

"Normal Actions" and "Interrupt Actions".

The approach used in LINKman is sequential. In a first phase, a specific logic checks whether the kiln is in normal operation or in upset situation.



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Normal Actions

If the kiln is in normal condition, then the parameters BZT, OX and BET are calculated.

- BZT (Burning Zone Temperature) represents the clinker burning degree, calculated out of NOx, Kiln Amps, and zone temperature.
- OX, the draught index, based on the gas composition at kiln inlet.
- BET (Bet End Temperature): the degree of preparation of the material when entering the kiln.

LINKman selects then one of 4 ruleblocks to determine the required setpoint changes, depending on:

- the deviation between the actual feed rate and the desired target,
- the spare draught capacity,
- The process conditions.

Rule Block	Actions on
General	Feed, Fuel, Fan
Top-feed	Fuel, Fan
Top-damp	Feed, Fuel
Stable	Fuel

The changes calculated using the ruleblocks are scaled to physical changes. However these changes are merely based on the present kiln conditions. Specific actions are then carried out, if required, to take into account the previous actions and the process dynamics, they are included in the "Post-Rule Block Processing" module.

Interrupt Actions

If a kiln-upset condition is detected, a dedicated control action is applied. As these actions have a higher priority than the normal actions, they are called interrupt actions.

Examples:

Break-Action	if the kiln is in unstable conditions for a longer period
Ring-Action	if a ring fall is detected
Hot-Action	if the kiln gets very hot
CO-Action	if a high amount of CO is detected

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6. BENEFITS AND KEYS OF SUCCESS

The experience shows that substantial benefits can be expected from HLC systems if they are properly implemented and used. The following table shows typical ranges of quantifiable benefits:

Item	Typical Range
Kiln output	+ [0 - 5 %]
Heat Consumption	- [0 - 5 %]
Refractory Life	+ [0 - 30 %]
Long-term clinker strength	+ [0 - 5 %]
Electrical energy for clinker grinding	- [0 - 10 %]
NOx emissions	- [0 - 30 %]
Number of kiln stops	- [0 - 30 %]
Standard deviation of key variables	- [0 - 50 %]

In addition, the following qualitative benefits have to be mentioned:

- The working conditions of the operators are greatly enhanced. As the computer handles the routine tasks, the operator can concentrate on more important matters.
- Process analysis and optimization are made more easily since opportunities for testing new control strategies and new ideas are available, assuming that the HLC is user-friendly enough.
- The use of HLC imposes to keep the instrumentation in a good operating state.

The keys of success with HLC systems are:

- An adequate and reliable instrumentation (sensors, actuators, PID controllers, etc.)
- A stable and uniform raw mix chemistry
- An optimized combustion
- A highly motivated personnel
- A follow-up of the performance of the system

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7. HOW TO JUSTIFY AN INVESTMENT IN HLC

In order to justify an investment in HLC, it is important to estimate the potential cumulative savings over the lifetime of the system. Figure 4 (Annex 4) gives an illustration of these savings over a period of ten years.

In this figure, curves 1, 1a and 1b indicate the cumulative savings with HLC when achieving „Best Operator Performance". If the HLC has a robust strategy and is permanently adapted to the evolution of the burning process, then the cumulative savings will increase consistently according to curve 1.

However, if the system is not properly maintained or not adjusted to changing process conditions, sooner or later it will stop functioning and the savings will obviously stop. These situations can take place very shortly after installation (curve 1a) or later on (curve 1b).

Curve 2 illustrates the cumulative savings, which can be realized in cases where the HLC is used as a tool for permanent process optimization. These extra savings are on top of the savings indicated on curve 1. The benefits resulting from this optimization can be achieved after reaching an HLC run time of more than 90 %. The savings through optimization can increase even more if the original HLC application is extended to other plant areas such as raw milling, stack gas flow control or integration of an efficient raw mix control strategy.

To summarize, simply by achieving „Best Operator Performance", the payback is generally less than two years (considering only the investment in HLC). Furthermore, if the HLC is used efficiently as a permanent optimization tool, then the cumulative savings will increase even more.

The experience shows that a payback of 1 to 2 years is realistic, when considering only the investment costs in the HLC system.

8. CONCLUSION

Over the last decade, HLC systems have penetrated the cement industry. About 300 applications have been reported which represents roughly 15 % of the cement manufacturing installations.

There is no doubt that the proper implementation and use of a HLC system, although requiring relatively low investment costs, provide significant enhancements in terms of productivity.

However it is of the utmost importance to select the adequate system, that means a system which has proven to be efficient and performant in the long term.

It must be remembered that the success of any HLC system depends upon:

- the quality of the instrumentation
- the raw meal preparation
- the quality of the combustion system
- the motivation of the works' personnel and the acceptance of the system by the operators

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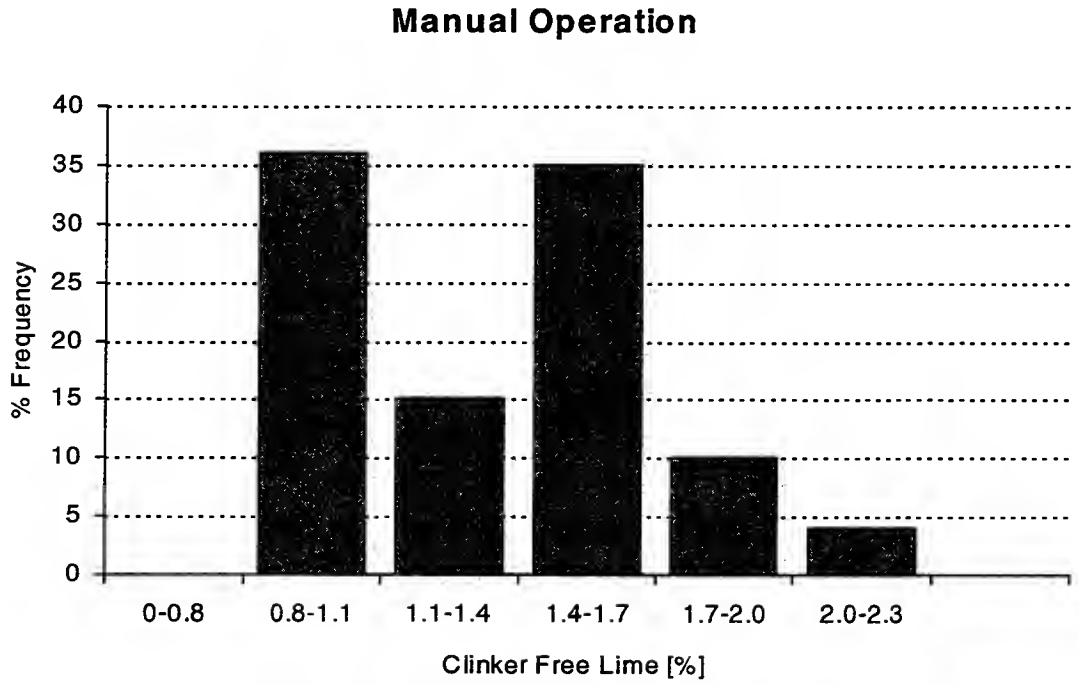
10. ANNEX

Figure 1: Example of Improved Clinker Uniformity



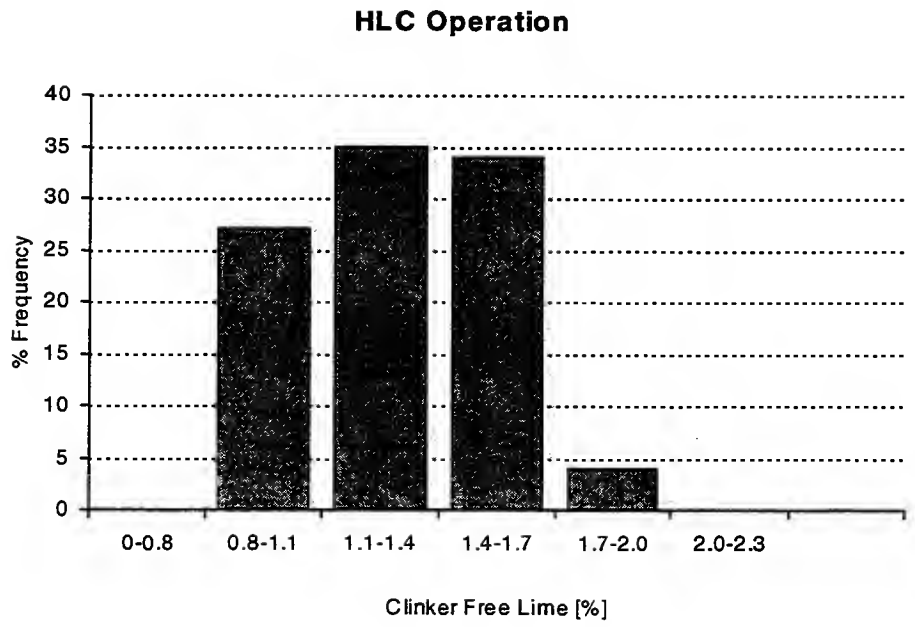


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Figure 2: Example of Improved Heat Consumption

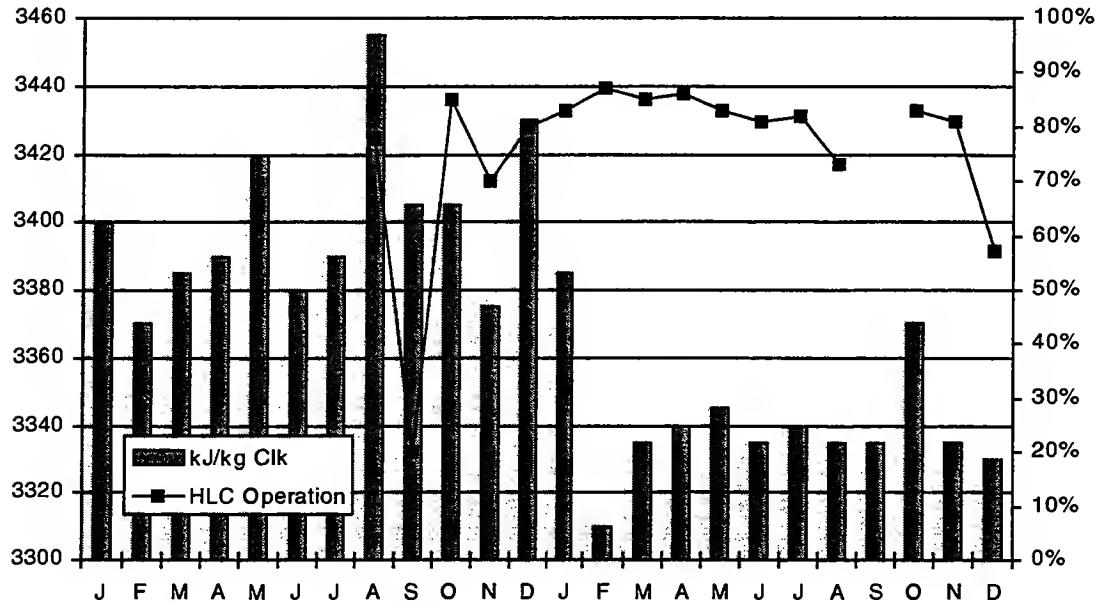

Figure 3: Operator Training

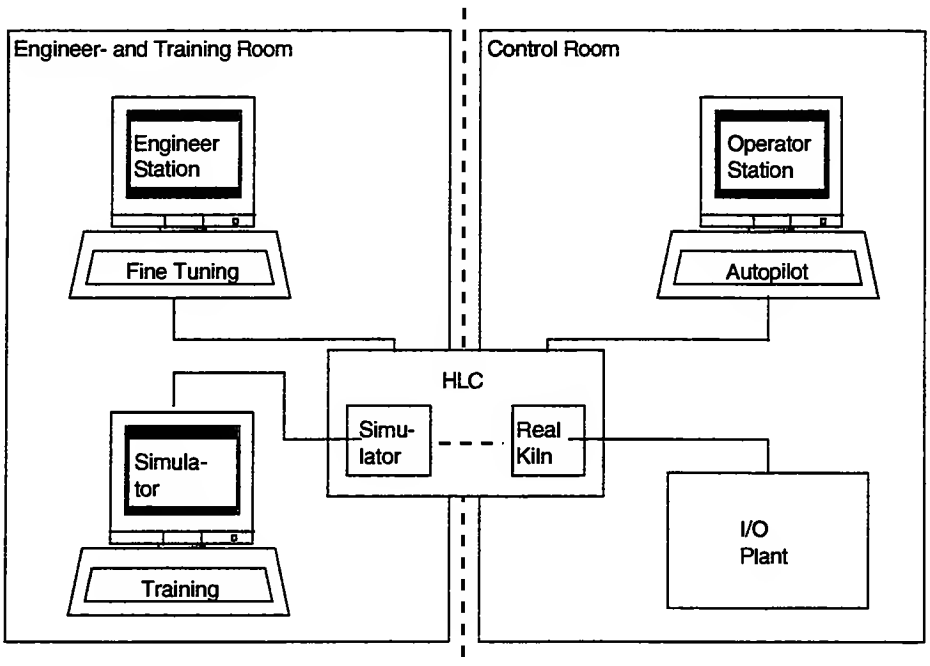

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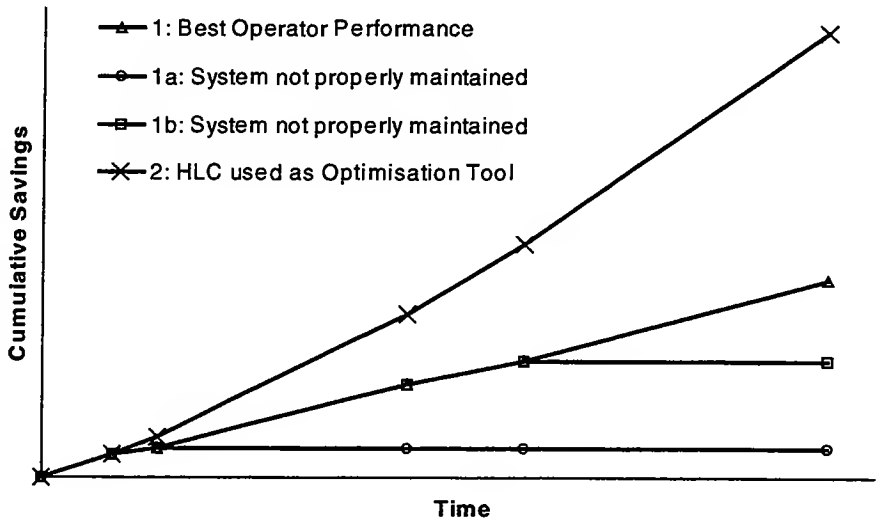
Figure 4: Lifetime Benefits of Process Optimization with HLC



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Field Preparation for High Level Control - Process Preview

U. Haberstich
PT 99/14495/E

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Summary and conclusion:

The introduction of a High-Level Control System requires a proper preparation of the plant to ensure an efficient and successful implementation and commissioning. For that purpose a so-called preproject from HMC is proposed in order to disclose eventual difficulties or even disability of the plant to implement such a system. A team of the Process Technology Department (HMC/PT) and Holderbank Engineering Switzerland (HES) will help the plant to achieve the best preparation.

HES will check the possibilities to connect the LINKman system to the installed plant control system and the sensitivity of measuring and control devices.

HMC/PT will check the quality aspects and the kiln behavior in order to disclose eventual process problems.

The main items of the LINKman preproject are:

- Kick-off meeting with explanation of the LINKman system and the procedure to implement it.
- Collection of all available plant data (Flow-sheets, technical descriptions of the installation).
- Collection of all available process data (Log-sheets, quality data, flow-sheets and trends).
- Combustion check.
- Description of the used strategy and rules for manual kiln and cooler control (which parameters were used) and a description of the most common kiln and cooler problems (dusty clinker, coating falls, etc.).
- Control of the adjustments of the most important control loops.
- Instrumentation check (type, position, sensitivity).
- Definition of the preparatory work to be done before the LINKman implementation.
- Conclusion meeting with the definition of the implementation schedule (ordering and commissioning). Definition of the project leader and of the responsible for the LINKman Hardware and Software (Strategy).

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1. INTRODUCTION

The introduction of a kiln High-Level Control System like LINKman requires several preparatory works by the plant to ensure an efficient and successful implementation. The capability of the kiln to be driven by an automatic system has to be ensured by a pre-project. As in every project, the organization has to be determined and responsibles have to be nominated.

The LINKman pre-project has to cover mainly the following capability checks of the kiln system:

- Disclosing eventual process problems
- Defining an adequate kiln and cooler control strategy
- Checking the instrumentation
- Checking the sensitivity of the measuring and control devices (HES)
- Defining the connection of the LINKman system to the installed plant control system (HES).

2. DETAILED SCOPE OF WORK

2.1 Definition of responsibilities and meetings

A kick-off meeting has to be organized at the beginning of the pre-project. Experience has shown that a more detailed explication of the LINKman system avoids confusions. The implementation schedule (ordering and commissioning) has to be defined as well as the project leader and the responsibles for the LINKman Hardware and Software (Strategy).

The responsible for the Hardware issues is usually an electrical engineer with knowledge of the plant control system and the automatic control loops.

The responsible of the Software (strategy) needs detailed knowledge about the burning process and kiln control. Therefore, a process engineer is strongly recommended. He will be the future "LINKman – Champion" and doing all further modifications of the strategy after the commissioning.

Before the implementation also the exact position of the Hardware has to be defined (location of the operator- and engineer-station in the control room and location of the LINKman cabinet).

After finishing the study, all requirements will be discussed during the conclusion meeting at the end of the visit. The preparatory work has to be terminated until the first commissioning of ABB. If stronger process problems were detected, it may be recommended to postpone the order of the system.

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2.2 Data collection

To prepare the system before delivering, ABB needs sufficient data about the plant. Therefore, a collection of all available plant data is required. This contains mainly:

Plant descriptions:

- Flow-sheet of the kiln system
- Flow-sheet of the plant control system (PLC-System)
- Kiln manufacturing data (see also Appendix 1)
- Layout of the control room (to place the Hardware and pre-configure the cabling)
- List of available control loops
- Instrumentation list.

Process data

- (During 1 week of representative production)
- Quality data of Raw meal, Hot meal and Clinker
- Operator log-sheets
- Statistical distribution of the freelime.

For 24 hours

- 1-day-trend of all important process values of that week (burning zone temperature, NOx, Amps, Calciner temperature, Preheater exit temperature, etc.)

Additional information

- Printout of all available kiln process screens (only 1 momentary printout)
- Short description of the used strategy and rules for manual kiln and cooler control (which parameters were used). Is there a correlation between the clinker quality and some kiln control parameters (NOx, kiln amps, burning zone temperature)?

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2.3 Combustion check

Process problems strongly influence the kiln behavior. Incomplete combustion leads to enhanced Alkali/Sulfur cycles within the kiln system. If the molar Alkali/Sulfur ratio of the total input of Alkalis and Sulfur is within the desirable range (0.8 to 1.5), minor encrustation and Sulfur ring formation take place.

To avoid a bad or wrong implemented strategy and bad availability of the LINKman system, a combustion check is required.

A combustion check contains:

- Description and analysis of all fuel at the main firing
- Gas analysis data at kiln inlet (O_2, CO, NOx)
- Assessment of sintering zone, coating and the burner position
- Alkali/Sulfur balance. For this, sufficient samples of the raw meal, hot meal and clinker have to be taken and analyzed.
- Burner check (Primary air amount, fuel oil pressure, exit velocities, pressure fluctuations, etc.)
- Description and analysis of all fuels at the secondary firing
- Gas analysis data at preheater exit (O_2, CO, NOx)

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2.4 Definition of the strategy

LINKman offers standard strategies for almost every kiln type. One of those strategies will be pre-configured before delivery and modified on site during the commissioning.

To ensure a proper predefinition of the kiln and cooler control strategy, a description of the current used strategy and the rules for manual kiln and cooler control is required.

Kiln Control:

Define the most important parameters estimating the burning zone temperature, the kiln inlet temperature and the oxygen level.

A description of the most common kiln problems (dusty clinker, coating falls, etc.) will help defining required special actions.

Cooler Control:

Define the most important parameters to ensure proper clinker cooling and high efficiency.

A description of the most common cooler problems (kiln rushes, red rivers, hot plates, etc.) will help defining required special actions.

2.5 Control loops

For a proper working High Level Control System, the in the PLC installed control loops have to work properly and smoothly. Therefore, the most important loops (kiln hood pressure control, under grate pressure control, etc.) have to be revised and adjusted, if required. A further possibility is the complete removing of a loop from the PLC in order to add it into the LINKman strategy or to switch it off during LINKman control.

2.6 Instrumentation check

The instrumentation is the most important point for an automatic kiln control. The signals have to be convenient and reliable. Therefore, the type and position of the most important sensors have to be revised and corrected if required. Especially the gas analyzers need periodic calibrations and maintenance inspections.

For a list with required process signals and its position see Report HES 98/6347/E.

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3. APPENDIX 1

3.1 KILN MANUFACTURING DATA

Plant

Kiln	No. :	.....
	Supplier:	.....
	Nominal Capacity:	..... (t/d)
	Max. Capacity:	..... (t/d)
	Type of fuels :	.....
		.....
		.....
		.....
		.....
		.....
	Heat Consumption:	..... (kJ/kg)
	Length:	..... (m)
	Diameter:	..... (m)
	Slope:	..... (%)
Preheater:	Number of strings:	.....
	Number of stages:	.....
Precalciner:	Type:	.....
	Type of fuels:	.....
		.....
		.....
		.....
		.....
	% Fuel PC / Kiln:	.....
Burner	Type:	.....
	% Primary air:	.....
Cooler	Type:	.....
	Number of grates:	.....
	Number of fans:	.....
	Width x length:	.....

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Field Preparation for High Level Control

F. Fehr
HES 98/6347/E

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1. INPUT SIGNALS FOR HIGH LEVEL CONTROL

The Input Signals of the High Level Control system (HLC) are the measured variables of the controlled systems (kiln, cooler, mill). In the following a selection of the normally used input signals is indicated.

Selection of Input Signals

Wet kiln

- Sintering zone temperature (Pyrometer)
- Secondary air temperature (if available)
- Amps of kiln drive (Torque)
- O2/CO/NOx at kiln inlet
- Temperature at chain zone
- Backend temperature
- Backend pressure
- Kiln hood pressure
- Any other significant process variable used by the kiln operators

Preheater kiln

- Sintering zone temperature (Pyrometer)
- Secondary air temperature (if available)
- Amps of kiln drive (Torque)
- NOx at kiln inlet or at preheater exit
- O_2/CO at kiln inlet
- Kiln inlet temperature
- Kiln inlet pressure
- Kiln hood pressure
- Preheater exit temperature or second stage from top
- Preheater exit pressure
- O_2/CO at preheater exit
- Any other significant process variable used by the kiln operators

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Precalciner kiln

- Basically the same signals as for preheater kilns, but additionally:
- NOx at kiln inlet
- Temperature exit lowest cyclone stage
- Tertiary air temperature

Lepol kiln

- Sintering zone temperature (Pyrometer)
- Secondary air temperature (if available)
- Amps of kiln drive (Torque)
- O_2/CO/NOx at kiln inlet
- Temperature in hot chamber of Lepol grate (Pyrometer or Thermometer)
- CO/O_2 after intermediate fan
- Temperature intermediate fan
- Pressure in hot chamber above grate
- Pressure in hot chamber underneath grate
- Kiln hood pressure
- Any other significant process variable used by the kiln operators

Clinker cooler

- Air rates of the individual fresh air fans
- Pressures of chambers 1, 2, 3
- Exhaust air temperature
- Temperature of cooler plates
- Middle air temperature (if any)

Cement mill

- KW of mill motor
- KW of bucket elevator
- Rate of separator returns (t/h)
- Noise level by electronic ear
- Temperature/pressure at mill inlet and outlet
- Production rate, e.g. belt weigher, pressure of pneumatic transport, etc.

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1.1 Checklist: Assignment of Input Signals

INPUT SIGNALS	Min.	Max.	Target

Name of Signal, Value

Sensor:
Location:
Significance:
Stability:
Comment:

Name of Signal, Value

Sensor:
Location:
Significance:
Stability:
Comment:

Name of Signal, Value

Sensor:
Location:
Significance:
Stability:
Comment:

Name of Signal, Value

Sensor:
Location:
Significance:
Stability:
Comment:

Name of Signal, Value

Sensor:
Location:
Significance:
Stability:
Comment:

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1.2 Example of a Checklist

INPUT SIGNALS		Min.	Max.	Target
1)	Secondary air tempe rature	650°C	1040°F	815°C
		1200°F	1900°F	1500°F
	Sensor:	o.k.		
	Location:	o.k.		
	Significance:	Indication is sensitive. Tendency is o.k.		
	Stability:	o.k.		
	Comment:	Measuring equipment is adequate. Useful signal for LINKman II. Upper/lower limits are exceeded, therefore the limits have to be adjusted.		
2)	Cooler exhaust air temperature	95°C	230°C	150-175°C
		200°F	450°F	300-350°F
	Sensor:	o.k.		
	Location:	o.k.		
	Significance:	not looked at		
	Stability	not stable due to unstable cooler operation.		
	Comment:	Measuring equipment is adequate.		
3)	Clinker temperature	-	-	
	Sensor:	o.k.		
	Location:	Existing location gives no representative signal.		
	Significance:	not looked at		
	Stability:	not looked at		
	Comment:	As a better location is not available, this temperature should not be considered for automatic control.		
4)	Grate speeds of cooler	-	-	
	Sensor:	-	-	
	Location:	-		
	Significance:	-		
	Stability:	-		
	Comment:	unproblematic signal.		

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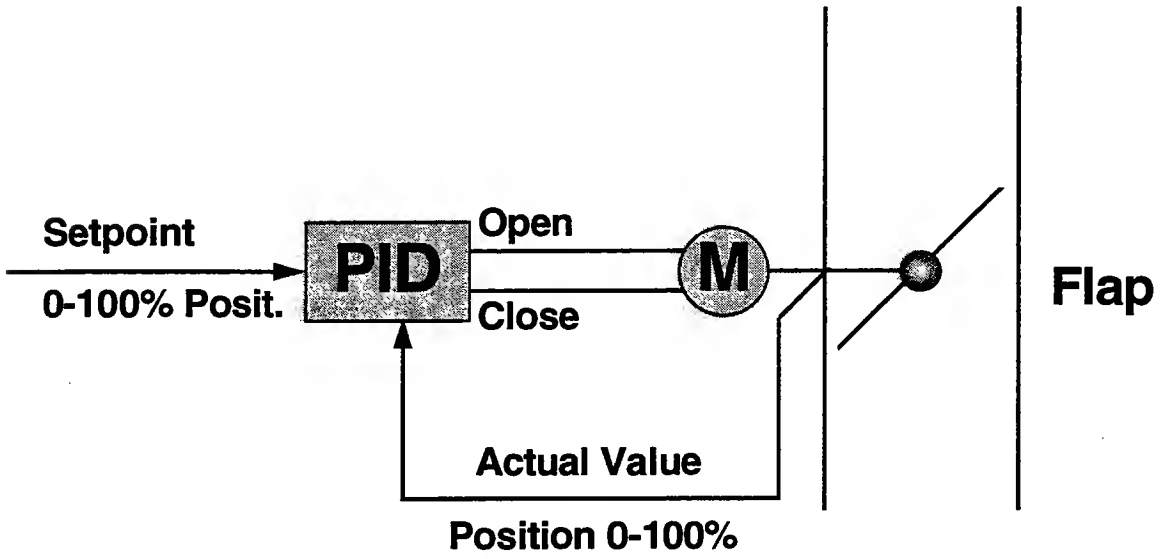
2. OUTPUT SIGNALS FOR HIGH LEVEL CONTROL

2.1 Primary Control Loops, Actuators

Basically all High Level Control output signals go as setpoints to primary control loops which then drive actuators.

Example:

Figure 1:



Example of a Primary Loop

The behaviour of the variable controlled by the primary loop is influenced by the following factors:

- repeatability of the sensor (specifications)
- absolute accuracy of sensor (calibration)
- type of actuator: continuous, step-wise
- tuning of PID controller
- deadband to protect actuator
- disturbances from outside (e.g. flushing material)

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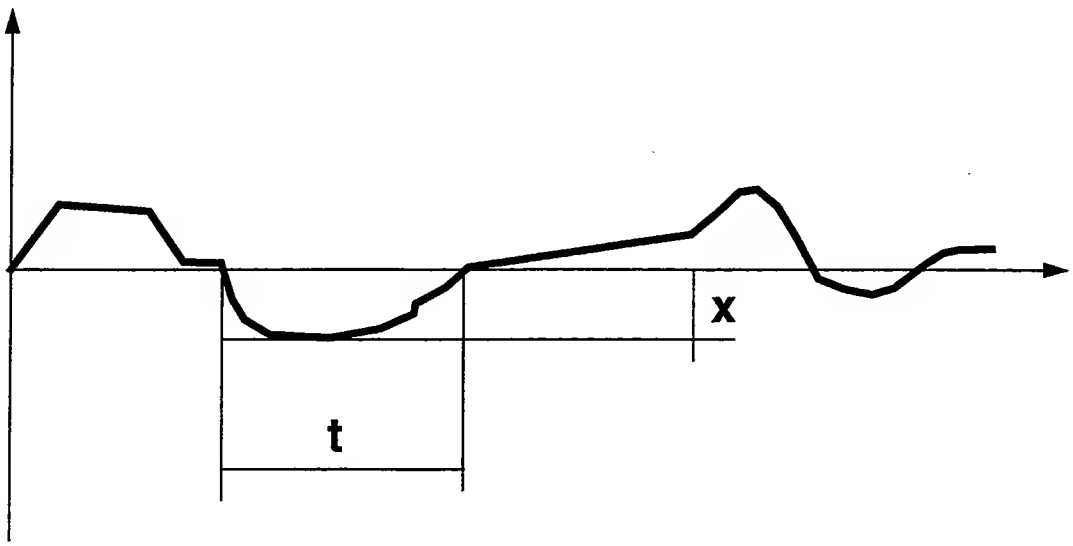
For HLC the primary loops have to fulfil the following criterias:

1) Tolerable deviation from setpoint

Figure 2:



Tolerable Deviation from Setpoint

Δ x: max. tolerable deviation during a time > Δ t

Δ x and Δ t depend upon what is controlled by the primary loop (e.g. coal, slurry to a wet kiln)

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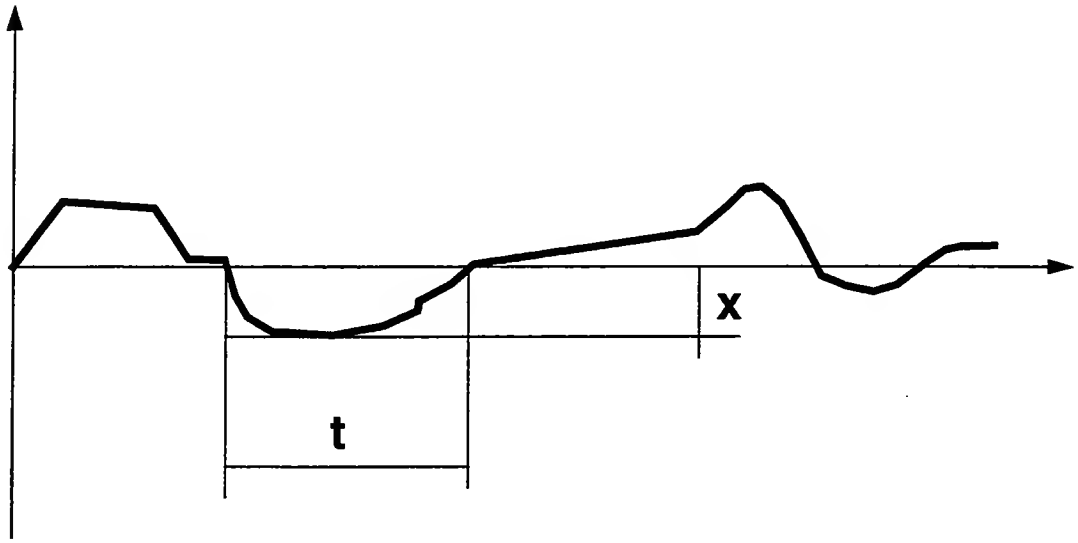
2) Sensitivity of setpoint change

What is the minimum applicable setpoint change that causes a reaction of the controlled variable?

Figure 2:



Tolerable Deviation from Setpoint

The criterias which have to be fulfilled are given in the following list for every type of primary loop. Most of the loops are not critical so that no criterias for the tolerable deviation are given:

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Table 1

	Tolerable Deviation from Setpoint		Sensitivity of Setpoint Change
	x[% of Span]	t [min]	SPmin [% of Span]
Cooler			
Volume rate fresh air fans			
- speed	-	-	0.5%
- damper position	-	-	1.0%
Kiln hood pressure			
- speed	-	-	0.5%
- damper position	-	-	1.0%
Grate speed ratio	-	-	1.0%
Under grate pressure	-	-	1.0%
Kiln			
Kiln fan			
- fan speed	-	-	0.5%
- damper position	-	-	1.0%
Kiln drive	-	-	0.5%
Fuel rates	3%	0.5	0.5%
Feed: wet	3%	10.0	0.5%
preheater	3%	2.0	0.5%
Dust insufflat. rate	10%	0.5	1.0%
Tertiary air dampers	-	-	1.0%
Intermed. fan (Lepol)	-	-	0.5%
Lepol grate speed	-	-	1.0%
Water to granulator	-	-	1.0%
Mill			
Mill feed	3%	0.5	1.0%
Separator speed	-	-	0.5%
Cooling air fan	-	-	
- fan speed	-	-	0.5%
- damper position	-	-	1.0%
Water injection	-	-	1.0%

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2.2 Example of a check list 1

PID Loop: **Kiln drive**

	HAC Code	0%	to	100%	Unit
Setpoint:	.....	.....	to	.....	[ ]
Actual Value:	.....	.....	to	.....	[ ]
Manipulated Variable: (actuator)	.....	.....	to	.....	[ ]

Visual check of actuator:

Tuned:

Parameters: K_P = ... T_I = ... [ ] T_D = ... [ ]

Max. deviation: Delta x = ... [% of span]
during: Delta t = ... [min]

Sensitivity: .... [% of span]
SPmin =

Remarks:

PID Loop: **Precalciner fuel rate**

	HAC Code	0%	to	100%	Unit
Setpoint:	.....	.....	to	.....	[ ]
Actual Value:	.....	.....	to	.....	[ ]
Manipulated Variable: (actuator)	.....	.....	to	.....	[ ]

Visual check of actuator:

Tuned:

PID Parameters: Kp = ... T_I = ... [ ] T_D = ... [ ]

Max. deviation: Delta x = ... [% of span]
during: Delta t = ... [min]

Sensitivity: .... [% of span]
SPmin =

Remarks:

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2.3 Example of a check list 2

PID Loop: **Main burner fuel rate**

	HAC Code	0%	to	100%	Unit
Setpoint:	.....	.....	to	.....	[ ]
Actual Value:	.....	.....	to	.....	[ ]
Manipulated Variable: (actuator)	.....	.....	to	.....	[ ]

Visual check of actuator:

Tuned:

Parameters:	Kp = ...	T_I = ...	[ ]	T_D = ... [ ]
Max. deviation:	Delta x =	....	[% of span]	
during:	Delta t =	....	[min]	
Sensitivity: SPmin =		....	[% of span]	

Remarks:

PID Loop: **Kiln feed**

	HAC Code	0%	to	100%	Unit
Setpoint:	.....	.....	to	.....	[ ]
Actual Value:	.....	.....	to	.....	[ ]
Manipulated Variable: (actuator)	.....	.....	to	.....	[ ]

Visual check of actuator:

Tuned:

PID Parameters:	Kp = ...	T_I = ...	[ ]	T_D = ... [ ]
Max. deviation:	Delta x =	....	[% of span]	
during:	Delta t =	....	[min]	
Sensitivity: SPmin =		....	[% of span]	

Remarks:

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3. INTERFACE TO PROCESS CONTROL SYSTEM

For any process control application, the High Level Control System needs to read data from the process (inputs from sensors) and to write data to the process (outputs to primary control loops). Moreover, for kiln/mill control, the High Level Control System requires also operator inputs such as:

- operating targets (NOx, O_2, etc.)
- operator setpoints (kiln feed, fuel feed, etc.)
- laboratory data (clinker factor, fuel heat value, % free lime etc.)

These inputs can be entered either from the LINKman II or from the Process Control System (PCS) The data communication between the PCS and the HLC is mostly based on a RS 232/422 serial link with the PCS brand specific communication protocol. The data set for exchange has to be pre-processed and stored in a specific memory location to be available for the data communication software.

For a kiln application for example, process data are accessed every 10 seconds and outputs are sent every 5 minutes, or with higher frequency if required, to update setpoints. The kiln strategy, for example runs every minute.

To connect the LINKman II to the PCS, two possible configurations are proposed:

- If the PCS has a bus connecting the whole plant (plant loop), then the LINKman II is interfaced by using a serial link to the interface box of the loop. An example is presented on the next page where the LINKman II is interfaced through a CIU (Computer Interface Unit) to a Bailey Network 90.
- If the PCS is structured according to the plant departments, the LINKman II has access to the individual departments through individual serial lines. The second picture shows a typical interface in a application where an Allen Bradley Programmable Logic Controller is used in connection with the HOLDERBANK ODH system as Men Machine Interface (MMI).

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Figure 4:

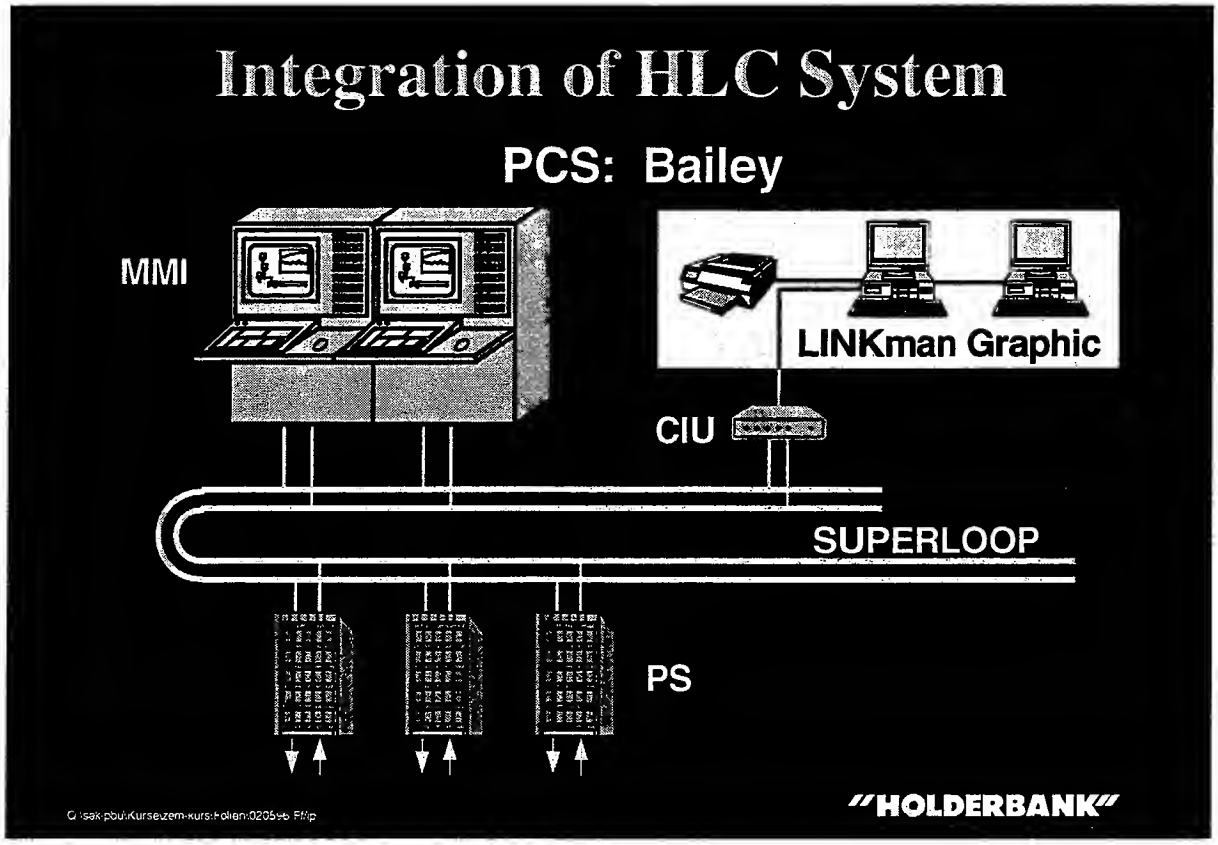
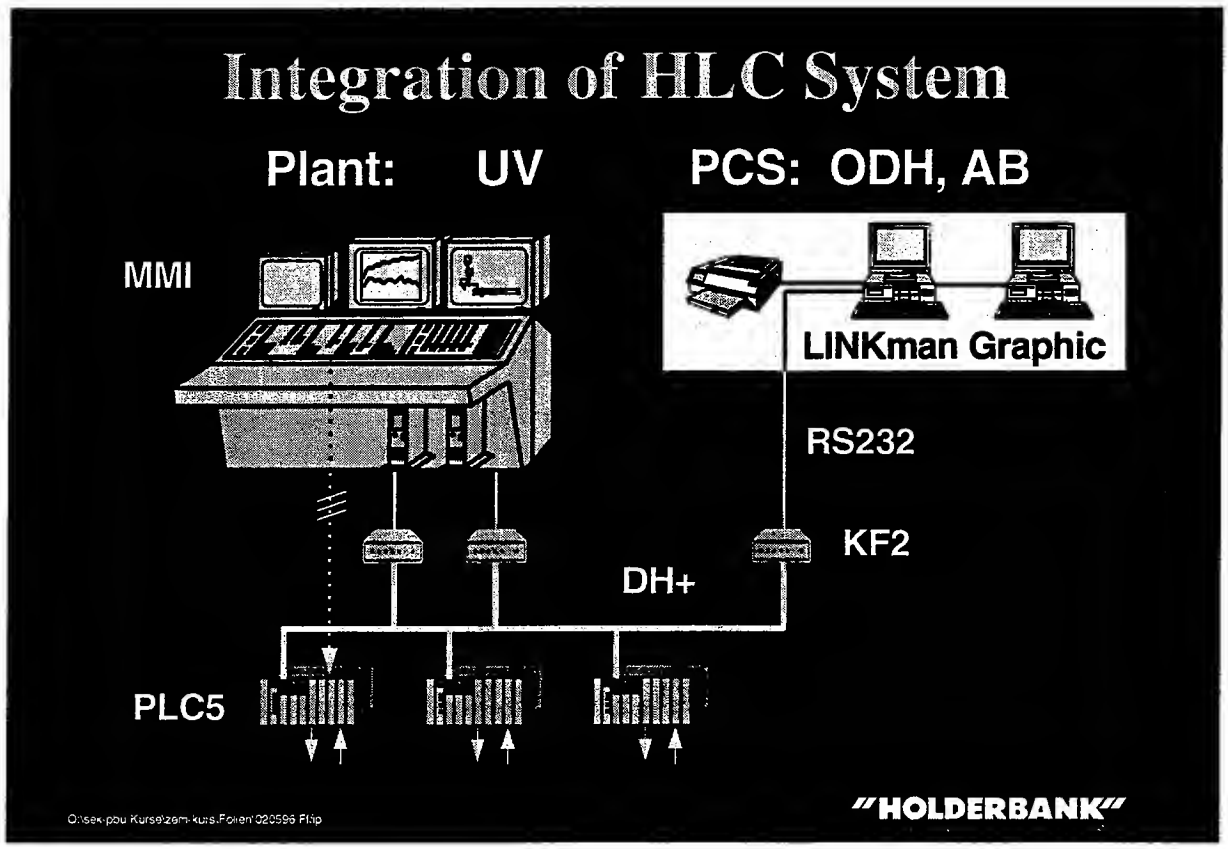

Integration of HLC System
PCS: Bailey
MMI
LINKman Graphic
CIU
SUPERLOOP
PS
"HOLDERBANK"

Figure 5:



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3.1 Data Exchange

The PCS does not only act as data acquisition system for the LINKman II, it serves also as operator interface (MMI, setpoint change, manual mode control), it hosts the basic loop controller, is responsible for automatic/manual switching including fail safe procedure if the LINKman II or communication breaks and does the alarm handling for the whole process (also HLC alarms). Because of this, the communication data set consists not only of analog inputs and outputs, but also of digital signals:

- Digital control bits from the PCS (digital inputs)
- Digital alarm and status bit from the HLC (digital outputs)

DIGITAL INPUTS

The PCS system has to provide different status bits from the process to indicate specific conditions which the LINKman II needs to work for proper operation.

Example:

- LINKman II on/off
- failure of instrumentation
- group ready/running
- direct/indirect operation of raw mill
- select status (type of cement etc.)

DIGITAL OUTPUTS

In a similar way, the LINKman II has to send status bits to the PCS to inform the operator about its status.

Example:

- LINKman II available
- on/off line
- normal condition
- upset condition
- alarms
- etc.

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3.2 Security

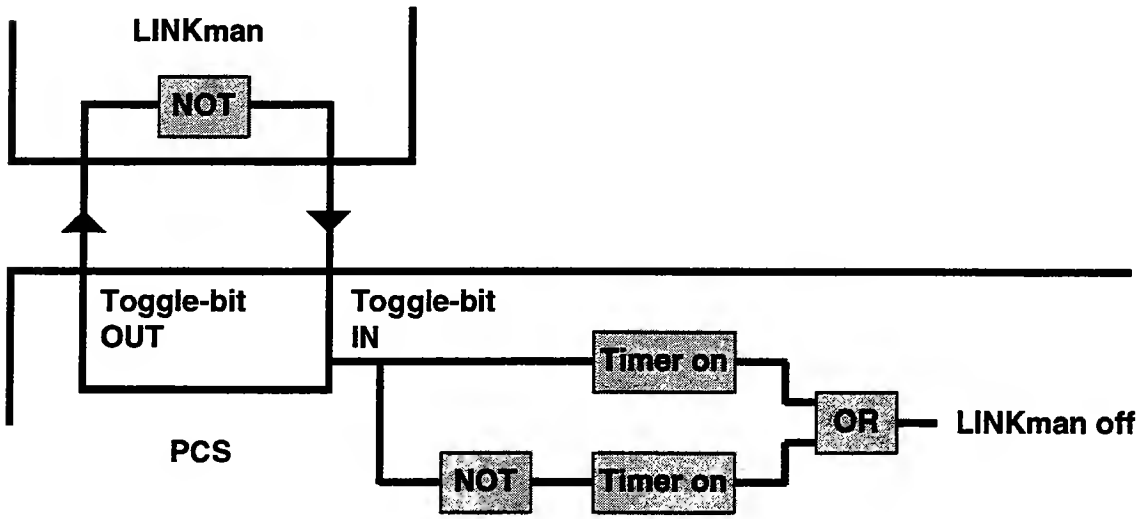
The communication and proper execution of the program have to be guarantied. Therefore, a watch-dog function on the PCS has to be realised:

Example:

Communication/HLC watch-dog

To check the communication as well as the LINKman II operating status (HLC on or off), a watch-dog toggle-bit has to be programmed.

Figure 6:



If the LINKman and the communication are on, the toggle-bit oscillates

The two on-delay timers control the on- and the off-time of the toggle-bit. If the time exceeds the pre-set delay time of the timers (typical 30 s), the toggle-bit has not been inverted and this means, that the LINKman II or the communication is off.

The PCS is responsible to switch the HLC on- or off-line (on operator's request) and to monitor the HLC's on/off-state feedback and to switch the setpoint signals accordingly (setpoint from operator MMI or from HLC). If the HLC goes off-line and the watch-dog detects a problem, the setpoint will be set to the operator control and an alarm has to be evoked.

Note: The setpoint switching has to be made bumpless. (see PID subject)

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4. EXAMPLES OF MOST IMPORTANT INPUTS IN RESPECT OF THE LOCATION AND CALIBRATION OF THE TRANSMITTER

4.1 General

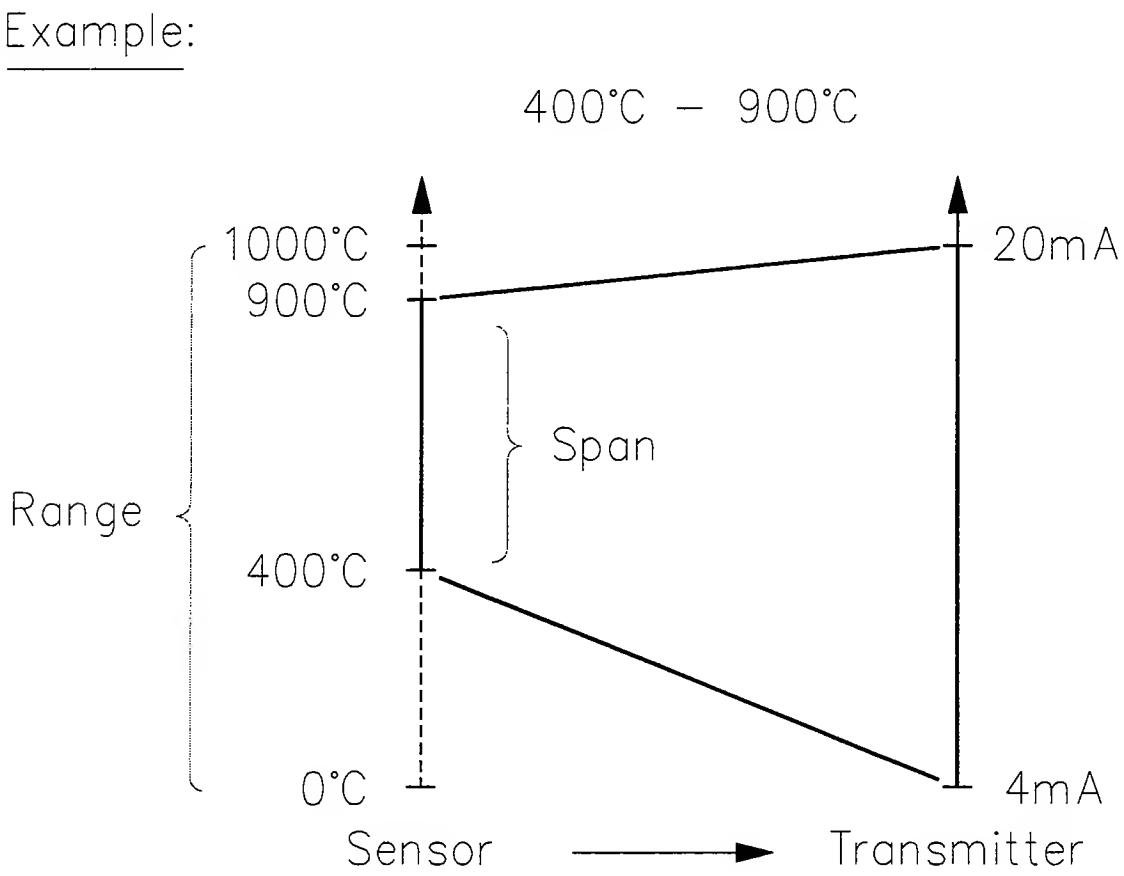
4.1.1 Definition:

Range = Max. - Min. of the sensor

Example: 0°C - 1000°C

Span = Used range for the electrical signal
f or monitoring the process

Figure 7:



4.1.2 Location of the Sensor

The location of the sensor is the most important thing in the instrumentation. There are a few locations which are really false and a lot which are good. For maintenance of the sensors it is quite frequently necessary (especially in the kiln area), that the location of the sensor should be above all easy accessible.

4.1.3 Calibration of the Transmitter

The calibration includes first of all the checking of the transmitter. There are different methods how to check:

1) Reference measurement: The sensor will be tested on a well-defined media (boiling oil, known weight, etc.)
2) Comparison measurement: Measure the process value by an other measuring device (hand-held thermometer/multimeter, pressure U-tube, etc.)
3) Find out the actual physical measurement by measuring the electrical signal coming out of the sensor (only if the characteristics are well-known,should not be used on thermocouple due to the cold reference junction).
4) Simulating the process with an alternative source by entering a calibrated signal on the primary side.

While applying method 1 or 2 as described above, the process value on the highest level (screen of the supervisor or high level control system) - if already installed - has to be monitored. If o.k., the procedure of course is quickly and successfully finished. If not, the output of the transmitter has to be measured by an Amp-Meter (4...20mA) and the transmitter has to be calibrated according its manual. In very seldom cases, the problem is in the scaling of the PLC or the supervisory system.

Note: Two hints on calibration

* Do a Plausibility Check. This means that the signal should be checked of its plausibility. Is it on its expected value? If not, there is something wrong. Do not accept everything you measure. Repeat the measurement if it is not plausible or try to make the calibration by an other way.
* Accuracy. If the sensor is not accurate enough or the transmitter has a too large range, try to calibrate it within the operating range.

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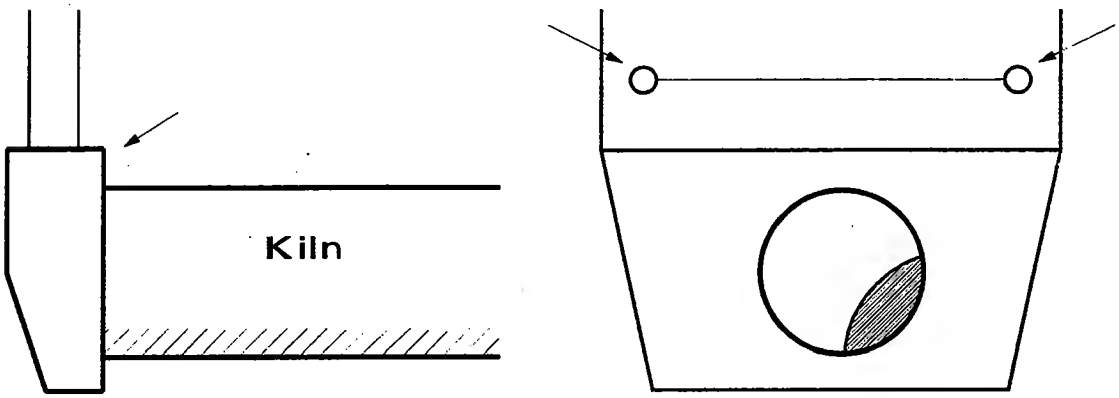
4.2 Examples

4.2.1 Temperature

EXAMPLE 1: KILN INLET TEMPERATURE

LOCATION:

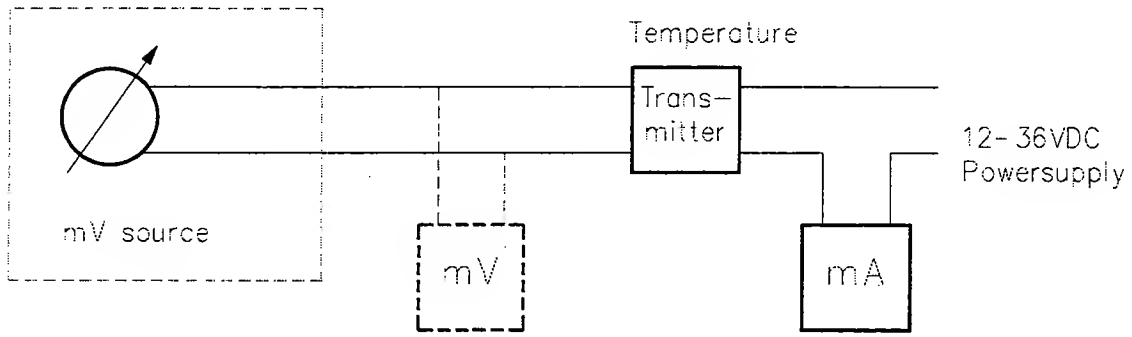
Figure 8:



- Easy accessible for maintenance
- Cleaned by pulling out (daily)
- Tip of sensor should be in the air-stream but not affected too much by incrustation
- Sensor should not be damaged by falling material
- The final position has to be evaluated

CALIBRATION:

Figure 9:



- Inject a voltage in mV without the sensor into the transmitter, while measuring in the 4...20mA line. Take readings at the display for 0 and 100%.

SPAN:	minimum:	0%	=	1000°C	=	4.0mA
	maximum:	100%	=	1250°C	=	20.0mA
	unit:	1%	=	2,5°C	=	0.16mA
	typical value:	80%	=	1200°C	=	16.8mA (16:100x80+4)

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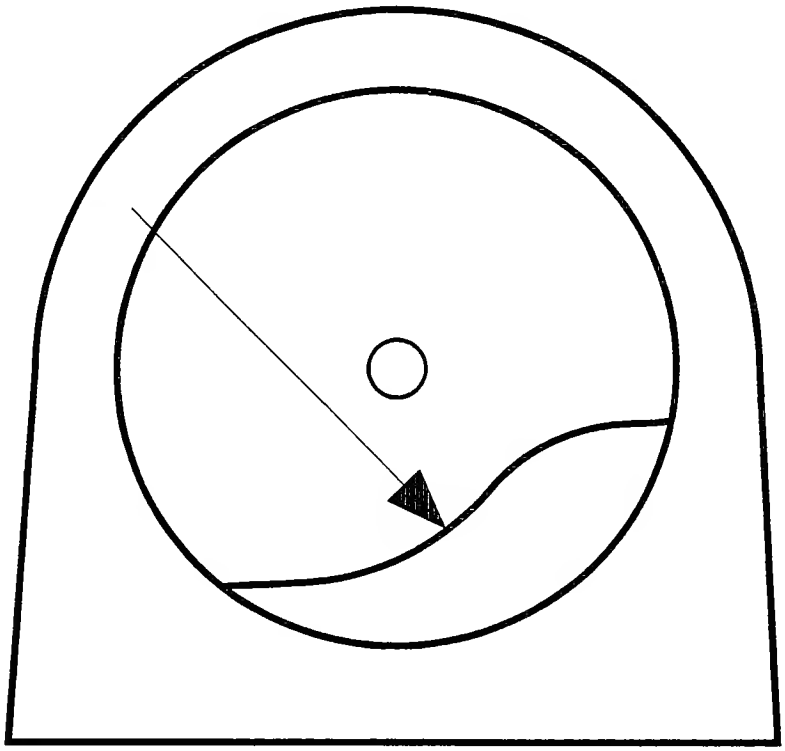
EXAMPLE 2: PYROMETER (at kiln hood for clinker temperature)

LOCATION:

Figure 10:



- Aim spot 30 cm (1 foot)
- Do not point into flame (radiation)
- Aim the pyrometer below the flame into the clinker just before the clinker flows out of the kiln
- Dust on the lens of the pyrometer or between lens and clinker affects the measurement, therefore, choose a short measuring distance

CALIBRATION:

- Calibration according to the manufacturer. (Does not have to be calibrated under normal circumstances)

SPAN:	minimum:	0%	=	1100°C	=	4.0mA
	maximum:	100%	=	1600°C	=	20.0mA
	unit:	1%	=	5°C	=	0.16mA
	typical value:	depending on the kiln				

Note: The absolute temperature is not so important, changes have to be monitored.

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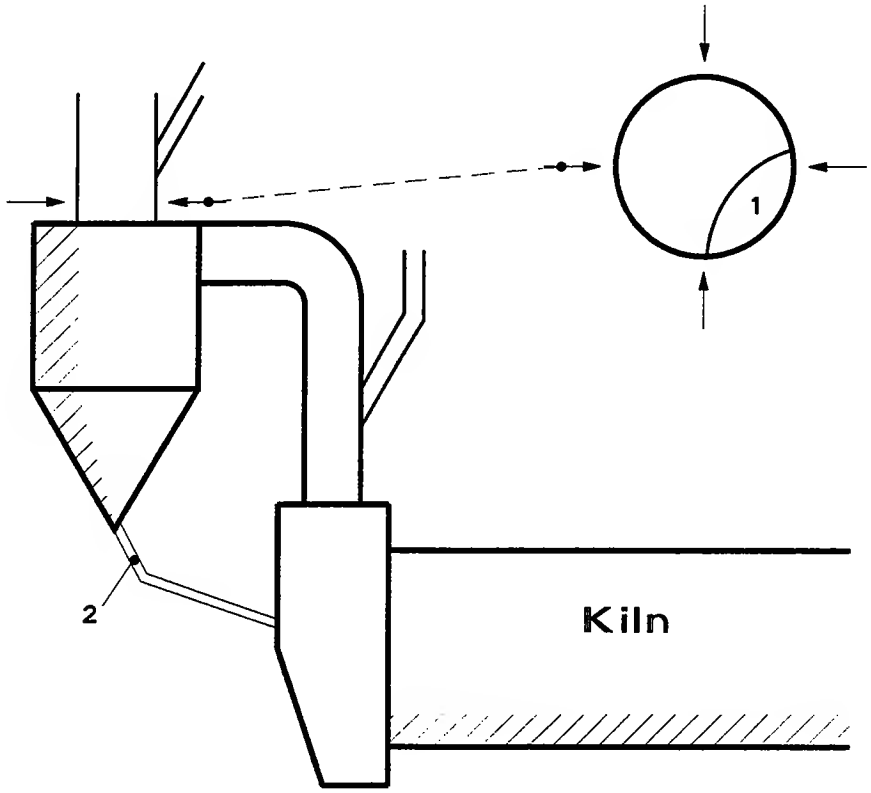
EXAMPLE 3: PRECALCINER LOWEST CYCLONES

LOCATION:

Figure 11:



1 Dead zone

2 Good position for material temperature

- Find the 2 right locations out of 4
- Parallel probes recommended

CALIBRATION:

SPAN:	minimum:	0%	=	600°C	=	4.0mA
	maximum:	100%	=	1000°C	=	20.0mA
	unit:	1%	=	4°C	=	0.16mA
	typical value:	72%	=	890°C	=	15.5mA (16:100x72+4)

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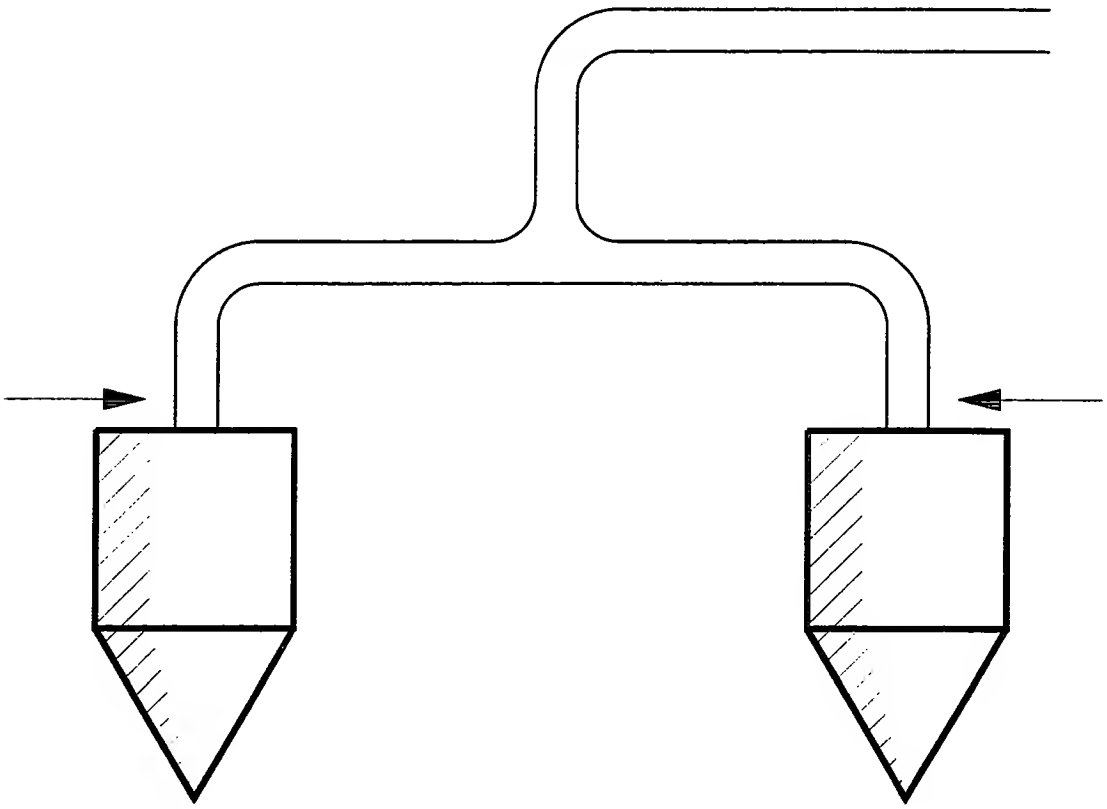
EXAMPLE 4: PREHEATER EXIT TEMPERATURE

LOCATION:

Figure 12:



- Good for minimizing of exhaust gas temperature
- Equilibrium of tower streams
- If the temperature of the second stage is more indicative, it is advantageous to use this value

CALIBRATION:

SPAN:	minimum:	0%	=	0°C	=	4.0mA
	maximum:	100%	=	500°C	=	20.0mA
	unit:	1%	=	5°C	=	0.16mA
	typical value:	70% =	350°C =	5.2mA (16:100x70+4)		

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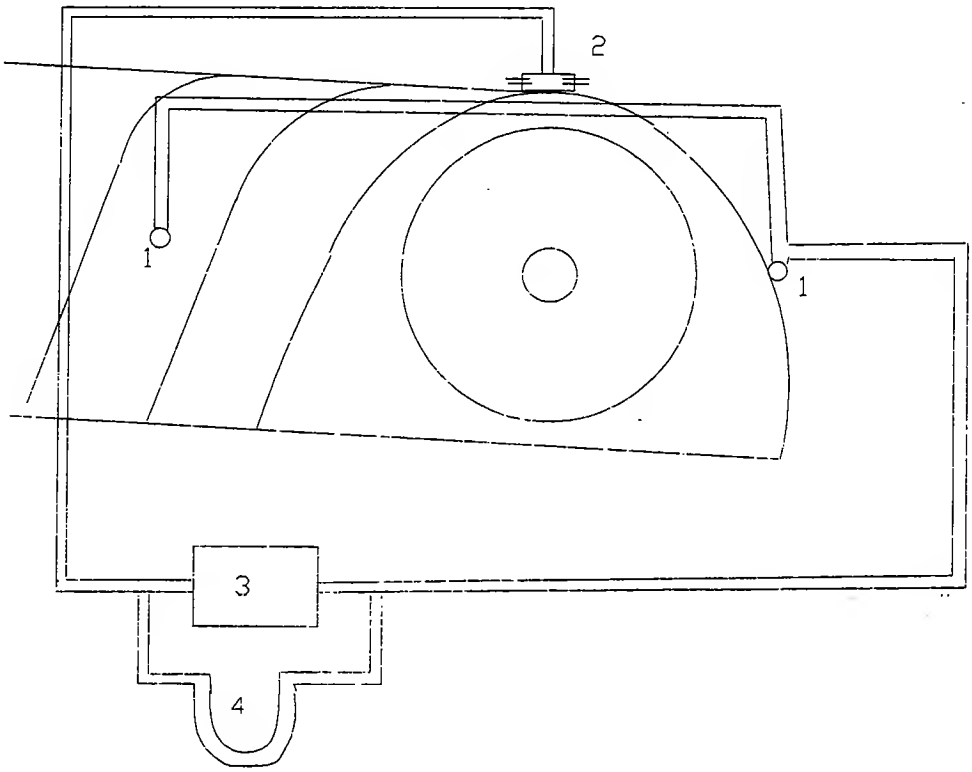
4.2.2 Pressure

EXAMPLE 1: KILN HOOD

LOCATION:

Figure 13:



1	Measuring points (hood pressure)
2	ambient pressure (heated up air)
3	transmitter
4	U- Tube for calibration purpose

CALIBRATION:

- Measure the pressure by an U-Tube (differential measurement)

grate cooler:	SPAN:	minimum:	0%	=	0.1mbar	=	4.0mA
		maximum:	100%	=	-0.3mbar	=	20.0mA
		unit:	1%	=	0.004mbar	=	0.16mA
	typical value: 37.5% = -0,05mbar = 10.0mA (16:100x37.5+4)						
planetary cooler:	SPAN:	minimum:	0%	=	-2mbar	=	4.0mA
		maximum:	100%	=	-3mbar	=	20.0mA

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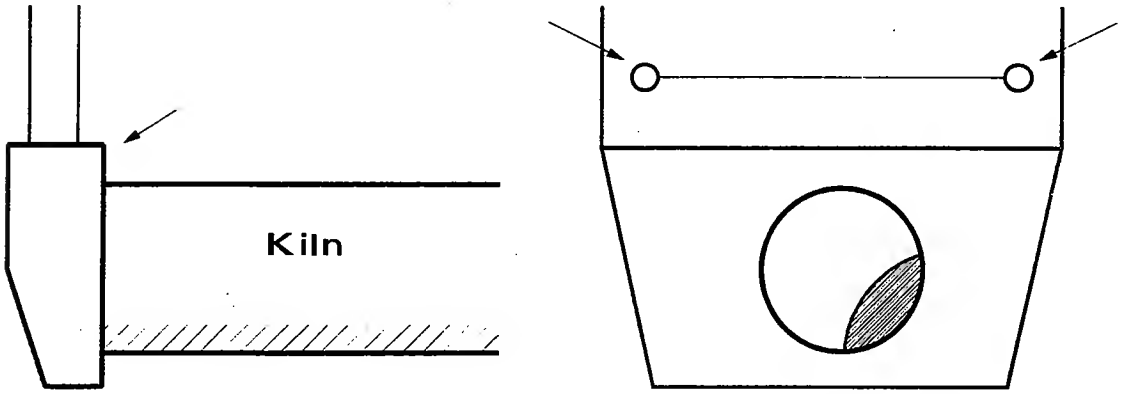
EXAMPLE 2: KILN INLET PRESSURE

LOCATION:

Figure 14:



- Daily cleaning required
- Indicates ring formation, limits max. production

CALIBRATION:

grate cooler:	SPAN:	minimum:		0%	=	0mbar	=	4.0mA
		maximum:		100%	=	10.0mbar	=	20.0mA
		unit:		1%	=	0.1mbar	=	0.16mA
		typical value:			=	2-4mbar		
planetary cooler:	SPAN:	minimum:	0%		=	0mbar	=	4.0mA
		maximum:	100%		=	10mbar	=	20.0mA
		typical value:			=	3-7mbar		

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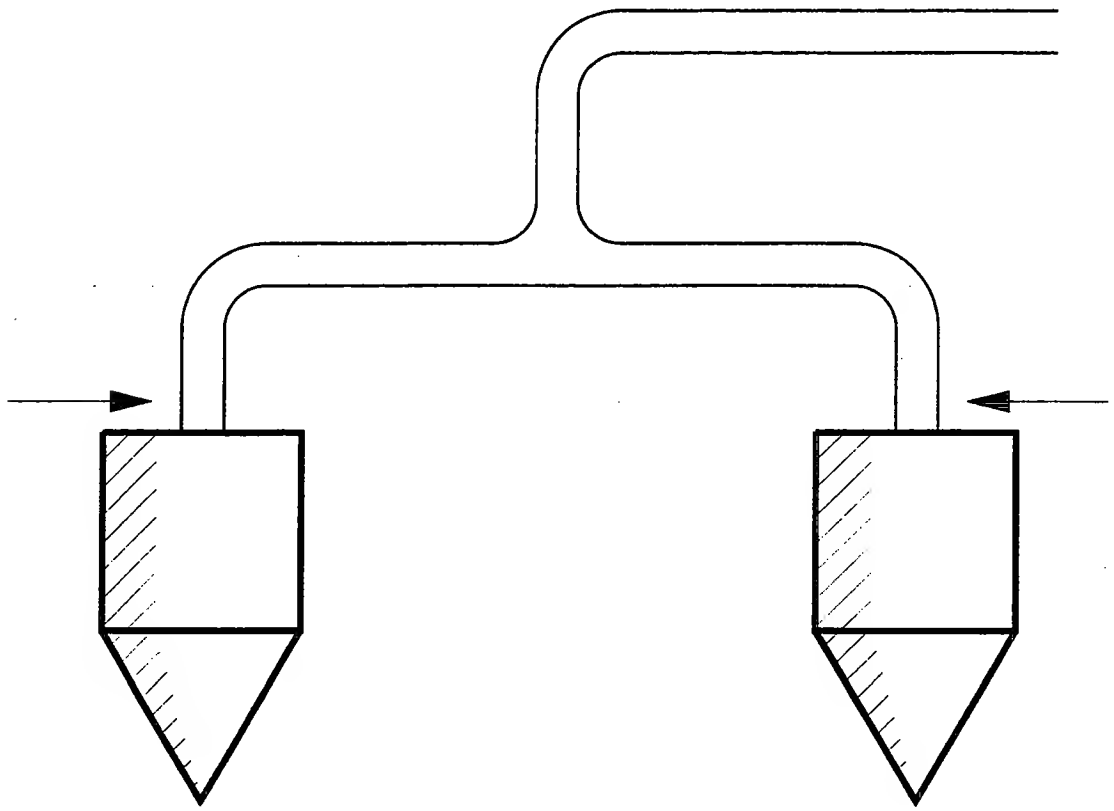
EXAMPLE 3: PREHEATER EXIST PRESSURE

LOCATION:

Figure 15:



- Please note, that this pressure may be different from the ID-fan inlet pressure because of the resistance in the tubes.

Typical value: 5mbar

i.e. 5mbar pressure difference from preheater top to ID-fan (at bottom, difference in height : 100 m)

CALIBRATION:

grate cooler:	SPAN:	minimum:	0%	=	0mbar	=	4.0mA
		maximum:	100%	=	10.0mbar	=	20.0mA
		unit:	1%	=	1mbar	=	0.16mA
		typical value:		=	40-60mbar		

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4.2.3 Speed

EXAMPLE: DC-DRIVE KILN

LOCATION:

- DC-Drive Panel. Usually the transmitter is in the panel as well.

CALIBRATION:

- By a hand-tachometer (analogous or digital) on the kiln motor or motor coupling.

SPAN:	minimum:	0%	=	0rpm	=	4.0mA
	maximum:	100%	=	1500rpm	=	20.0mA
	unit:	1%	=	15rpm	=	0.16mA
	typical value:	85%	=	1275rpm	=	17.6mA (16:100x85+4)

4.2.4 Current

EXAMPLE: KILN DRIVE CURRENT

LOCATION:

- DC-Drive Panel. Usually the transmitter is in the panel as well.

CALIBRATION:

- By the Amp-meter on the DC-drive panel while measuring in the 4...20mA line.

SPAN:	minimum:	0%	=	0A	=	4.0mA
	maximum:	100%	=	1000A	=	20.0mA
	unit:	1%	=	10A	=	0.16mA
	typical value:	30%	=	300A	=	4.8mA (16:100x30+4)

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IMPORTANT:

- The kiln amps are used to indicate the torque (and with the torque the:
  - coating falling
  - hot or cold clinker)
    If the field current is not kept constant by a thyristor controlled unit, the kiln drive amps have to be multiplied by the field current.
- The value has to be filtered in case of a planetary cooler, but in a way that ring breaks still can be registered
- Span must be 100% nominal motor current

4.2.5 Power/Energy

EXAMPLE: POWER OF MILL DRIVE

LOCATION:

- Medium-voltage Switchgear.

CALIBRATION:

- Calibration is usually not necessary. For checking the value, it can be calculated: voltage (phase to phase) x current x 1.732 (square root of 3) x power factor (cosø, see motor data, typical 0.97).

RANGE:	minimum:	0%	=	0kW	=	4.0mA
	maximum:	100%	=	2000kW	=	20.0mA
	unit:	1%	=	20kW	=	0.16mA
	typical value:	98%	=	1960kW	=	19.68mA (16:100x98+4)

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Chapter 6

Practical Work

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3. Gas Flow Measurement - Practical Work 393

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Control Technique - Practical Work

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SUMMARY

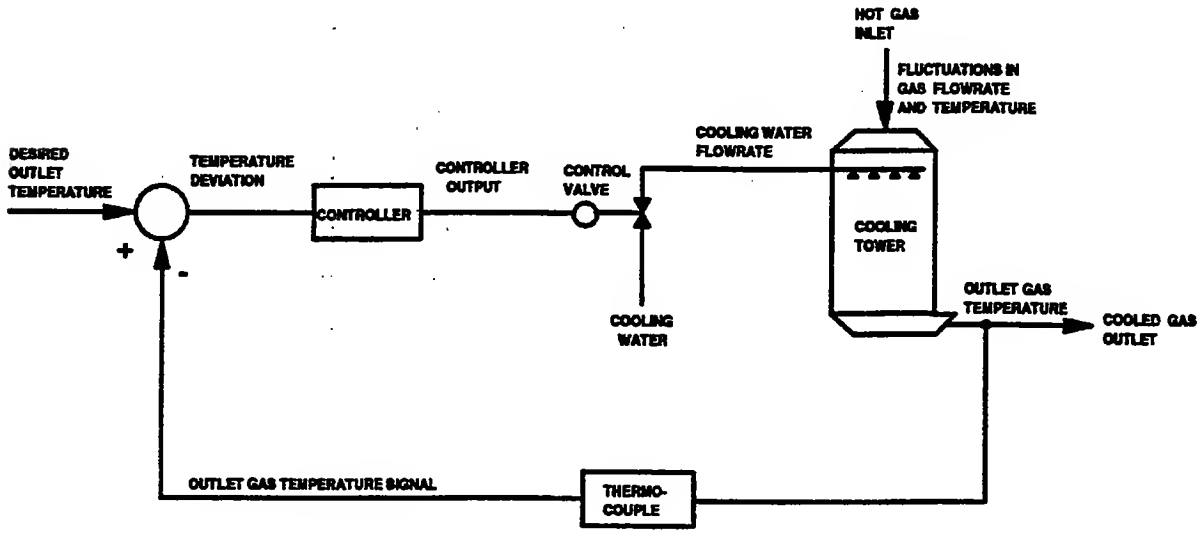
Lack of manpower, quality requirements and complicated processes have increased the importance of control to industry.

This lecture gives an introduction into the basic control theory and its use. Because of the variety of today's control techniques, applications are purposely omitted.

1. THE CONTROL LOOP

The functioning and the components of a control loop are explained by the example of cooling tower control loop.

Figure 1 Cooling tower control loop



1) **Process: The Cooling Tower**
The process consist of those machines and equipment which are controlled to produce a desired output.

2) **Process Inputs and Outputs**
For example:

Process inputs:	flowrate hot gas
	temperature hot gas
	flowrate cooling water
Process outputs:	flowrate cooled gas
	temperature cooled gas

3) **Controlled Output (c): Outlet Gas Temperature**
The purpose of the cooling tower control is to achieve a constant outlet gas temperature. Variations of the other process outputs are of no importance.

4) **Disturbances: Variations in Flowrate and Temperature of the hot Gas**
Variations of the process inputs cause disturbances to the process and therefore also to the process outputs, especially to the controlled output.

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5) **Manipulated Variable (m): Cooling Water Flowrate**
The cooling of the gas stream is achieved by spraying water into it. The degree of the cooling depends on the amount of water sprayed. Therefore, the outlet gas temperature can be controlled by manipulating the cooling water flowrate.

6) **Set point (s): Desired Outlet Gas Temperature**
The level at which the controlled output shall be kept has to be defined. This reference input or setpoint can either come from an operator or from an other control system.

7) **Feedback Elements (h): Thermocouple**
The process output you want to control has to be measured because you cannot control what you don't know. The feedback elements are always measuring devices.

8) **Actual Value (x): Temperature Signal**
The signal which is delivered by the feedback elements

9) **Error (e): Deviation between measured Gas Temperature and the desired Temperature**
The actual value is compared with the setpoint (desired value). The controller will react according to the deviation of these two signals.
Depending on the process it has to be selected between forward and reverse action:
Forward Action:
If the actual value is below the setpoint an increase of the controller output is required and vice versa.
Example: If the temperature in a kiln is too low the fuel has to be increased to heat up the kiln.
Reverse Action
If the actual value is below the setpoint a decrease of the controller output is required and vice versa.
Example: If the temperature after a cooling tower is too low, the flowrate of the cooling water has to be reduced to increase the temperature.
The switching between the two actions can simply be done by calculating the error either by subtracting the actual value (x) form the setpoint (s) (forward action) or by subtracting the setpoint (s) from the actual value (reverse action).

10) **Control Elements (g1)**
According to the error, the control elements act upon the manipulated variable. Two main elements:

- Controller: The controller is a device which generates the appropriate control signal according to the error (e) (see also the chapter "Controller").
- Actuator: The actuator transforms the controller output (y) into a change of the manipulated variable. In our example: A control value which acts upon the cooling water flow.

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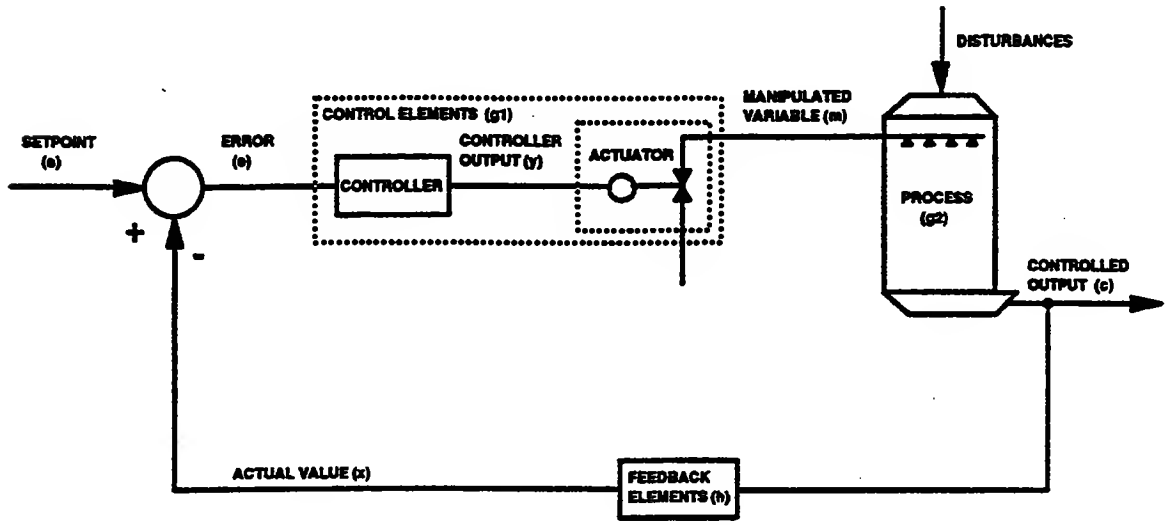
11) The commonly used expressions introduced above are summarized in the figure below:

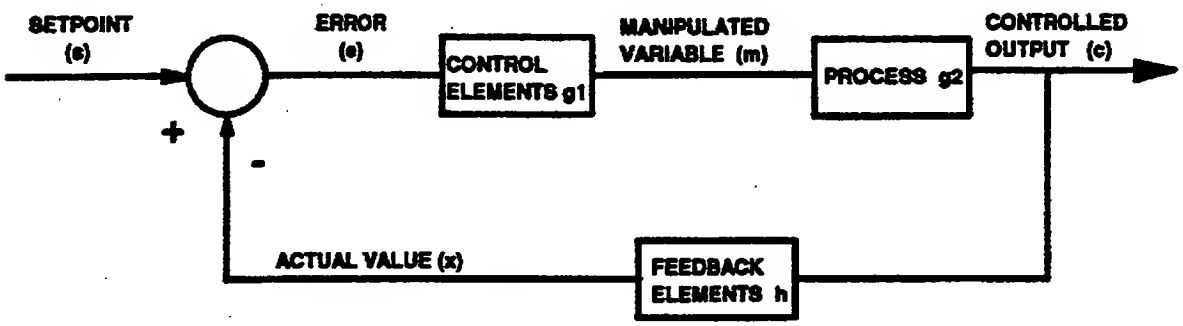
Figure 2 Cooling tower control loop



12)Block Diagram

Every closed loop control system can be represented with the same diagram.

Figure 3 Block disgram of control loop



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2. DYNAMIC BEHAVIOR OF THE PROCESS

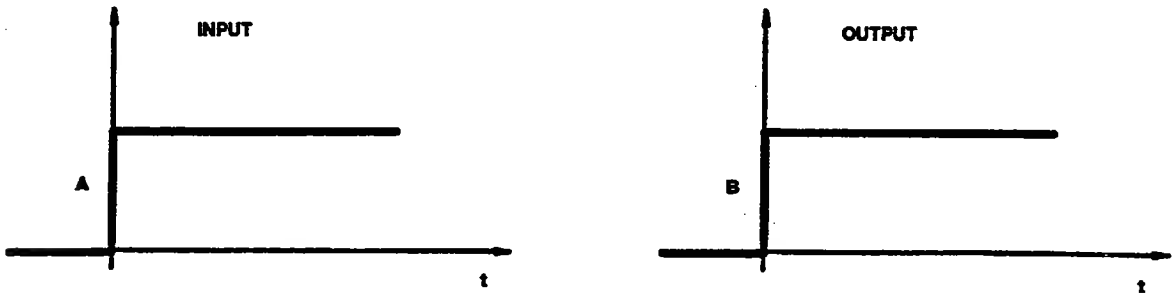
Depending on the nature of the process the output response to an input change or a disturbance varies. The most important tool to characterize the dynamic behavior of a process is the "transient responses": the reaction of the process to a step-change of the input.

2.1 Proportional Factor of the Process

The proportional factor describes how big the change of the output is caused by a one-unit change of the process input:

Proportional factors of the process = B / A

Figure 4 Proportional factor of the process

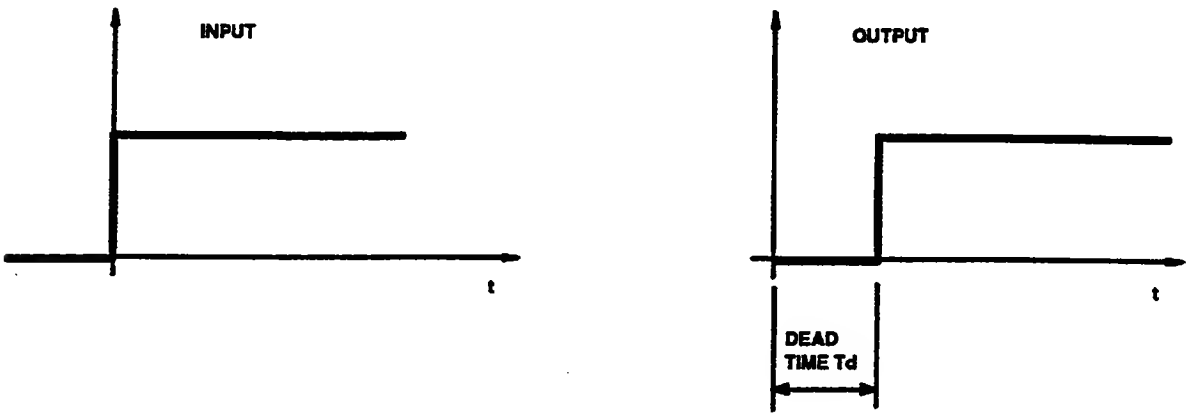


2.2 Dead Time Element

Most processes require a finite response time, often with a magnitude of minutes or hours. The dead time is defined as the difference in time between an input change and the resulting output change. The dead time slows down controller response because the controller cannot act until an error in output is detected.

An example for a typical dead time element is a belt conveyor. The dead time is given by the length and the speed of the belt.

Figure 5 Dead time



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The cooling tower as an example contains multiple dead time elements which can be added together:

- reaction time of the control value
- time the change in the water flow rate needs to reach the spraying nozzles
- time the gases need to pass the height of the cooling tower
- reaction time of the thermo-couple

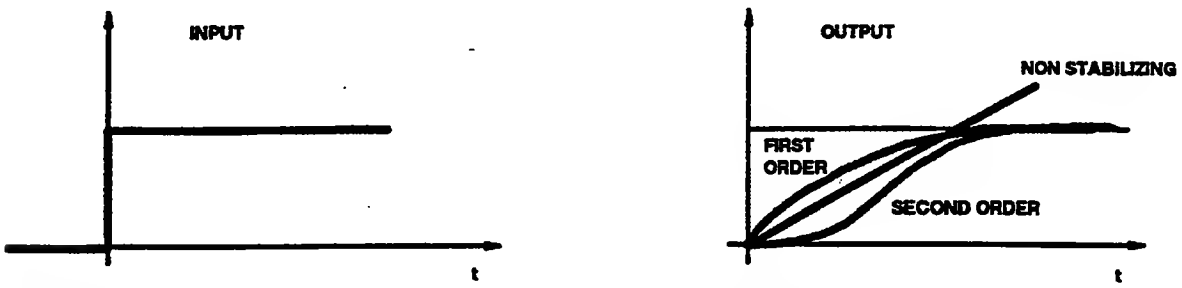
2.3 Delay Element

The reaction of a process to an input change is depending on the physical and chemical reactions controlling this process: heat transfer, mass transfer, chemical reactions. As all these reactions do not just happen within no time but follow certain time laws, the whole process shows certain time delays. These time delays can be classified in different types.

Example:

First order element, second order element. The most frequent is the second order delay.

Figure 6 Delay time

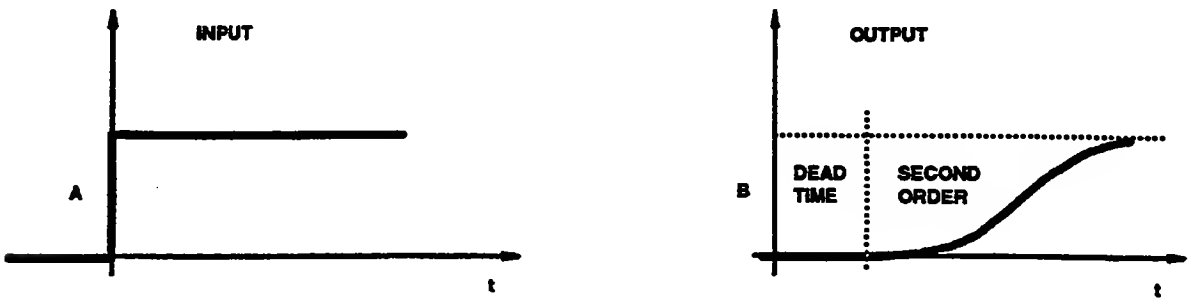


The delay in the cooling tower system is the time between the first noticeable change in the outlet gas temperature and the moment when the new stable temperature is reached.

2.4 Combinations

In most of the cases the process is a combination of a dead time and a delay element (second order).

Figure 7 Dead time plus delay time



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3. CONTROLLERS

As mentioned before the controller is a device which generates according to the deviation between desired and actual value an appropriate control signal to bring this deviation to zero.

3.1 Different Types of Controllers

1) On-Off Controller
   The controller output can only have two positions;: fully open and fully closed. The controlled variable is controlled within a band. When the variable falls below the lower limit of the band, the controller output changes to **"open".** The variable starts to rise again. When the upper limit of the band is reached, the controller output changes back to **"closed".** The variable starts to fall again.
   Example: Temperature control of a refrigerator.

2) Continuous controller (PID-controller)
   The output can have any position between open and closed.

3) Adaptive controller
   The controller adapts automatically its parameters to changing process characteristics. It has to be differentiated between controllers which adapt themselves continuously and controllers that adapt only when this is desired.

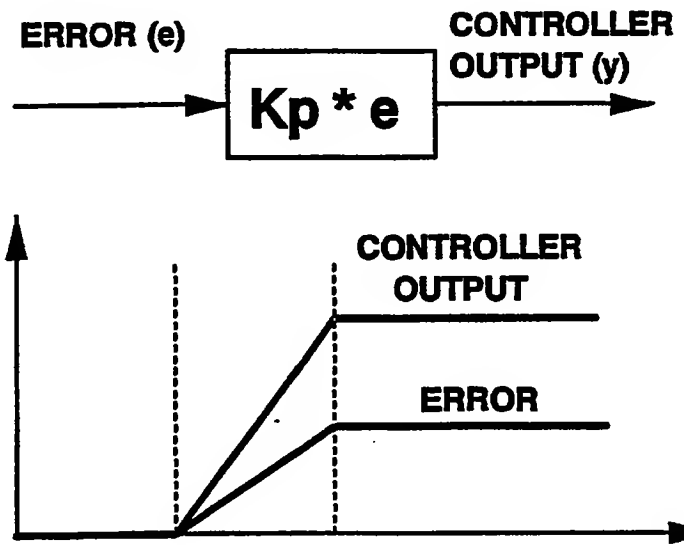
4) Predictive Controller
   The predictive controller uses a mathematical model to predict the behavior of the controlled process. With this information the controller can optimize the control actions.

3.2 PID - Controller

The most frequently used controller is the PID-Controller. The total control action is a combination of three basic control actions:

1) **P- Controller (proportional controller)**
   The output of the proportional controller is the error (e) multiplied by a constant factor. This factor is called the proportional constant or the proportional gain.

Figure 8 Proportional Controller



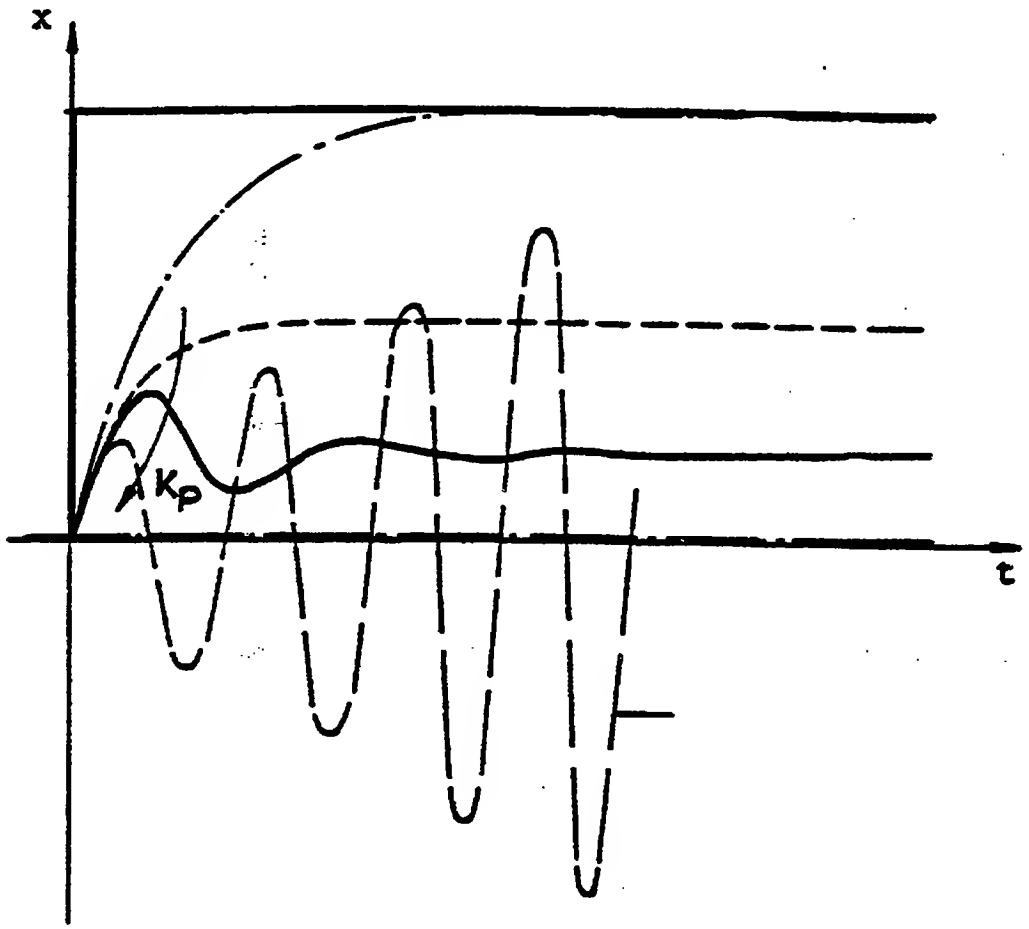
The control algorithm:

$$y = Kp \cdot e$$

Kp: Proportional constant

Characteristic Features:

- Increasing Kp causes stronger control actions and thus a faster return to the setpoint. But with increasing Kp the tendency of the system to cycle increases (see figure 9).
- Off-set: the P- controller is adjusted to work for one specific load of the controlled system (in the cooling tower example: one specific flowrate of the hot inlet gas). If the load changes, a permanent error.



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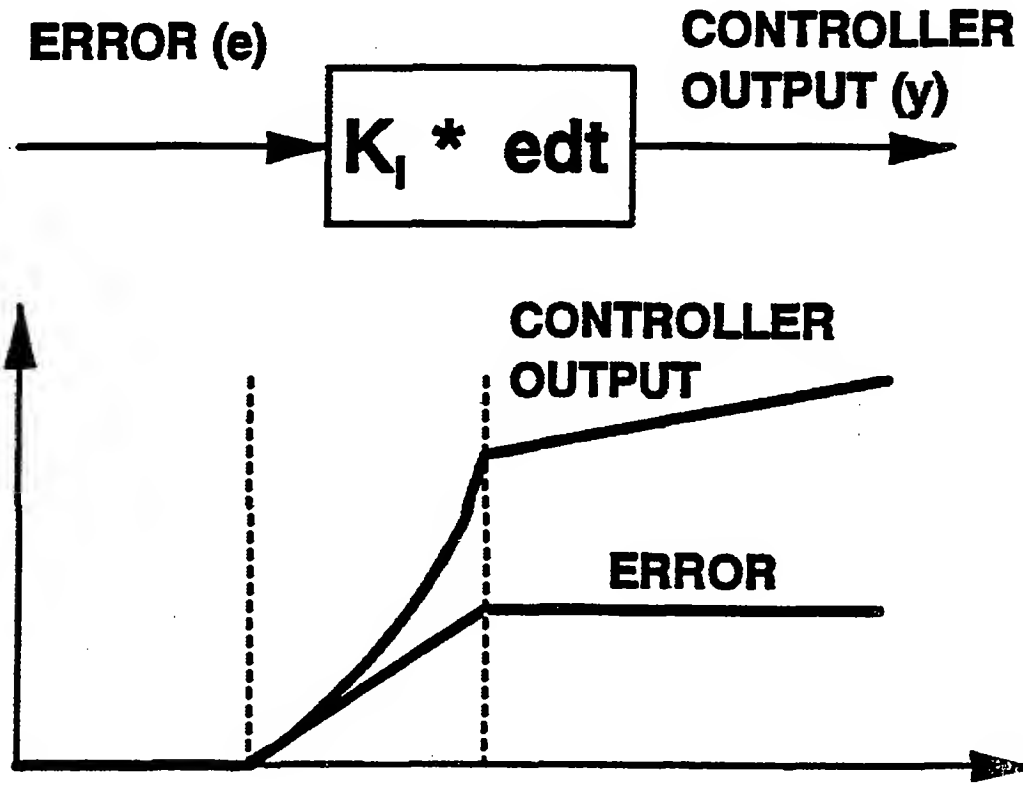
2) **I-Controller (integral or reset Controller)**
As long as a deviation between setpoint and actual value exists, the I-controller sums the deviation up with time and produces an output proportional to this sum.

Figure 10 Integral controller



The control algorithm:

$$y = KI \cdot \int edt$$

KI: integral constant or integral gain

KI can be expressed with the proportional constant

KI = Kp/TI

Kp: proportional constant

TI: integral action time - Time after which the output of the I-controller is equal the output of a P-controller (for a step).

Characteristic Features:

- As a permanent error leads to a continuously increasing control signal, an I-controller is very suitable to eliminate an off-set.
- With increasing KI increasing tendency to oscillate

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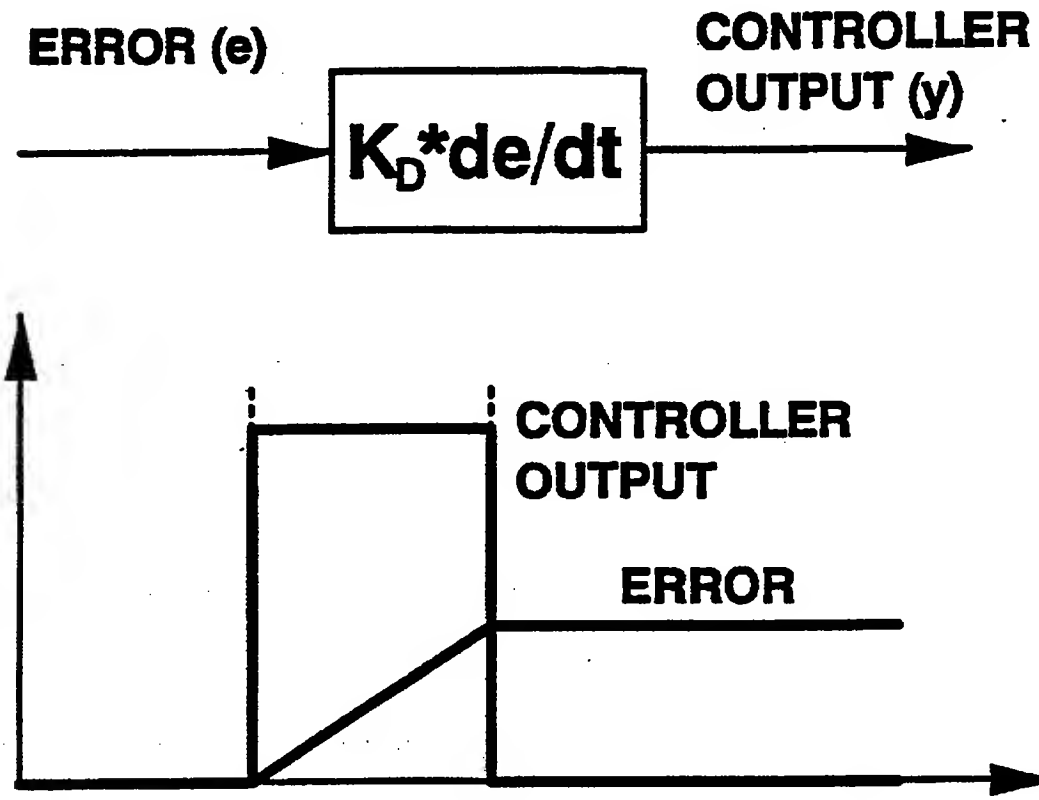
3) **D-Controller (differential, derivative or rate controller)**
The D-controller produces an output which is proportional to the rate of change of the deviation between setpoint and actual value. The D-controller weights the speed of a disturbance.

Figure 11: D-Controller



The control algorithm:

$$Y = KD \cdot de/dt$$

KD: derivative constant or derivative gain.
KD can be expressed with the proportional constant KD = Kp · TD

Kp: proportional constant

TD: derivative action time. Time after which the output of the D-controller is equal to the output of a P-controller (for a ramp).

Characteristic Features:

- D-controller reacts earlier because it detects slope errors which occur before real errors.
- A D-controller cannot be used alone. It is used in combination with P or PI controller actions.
- As the D-controller reacts to slopes, it can be unstable for highly oscillatory systems.

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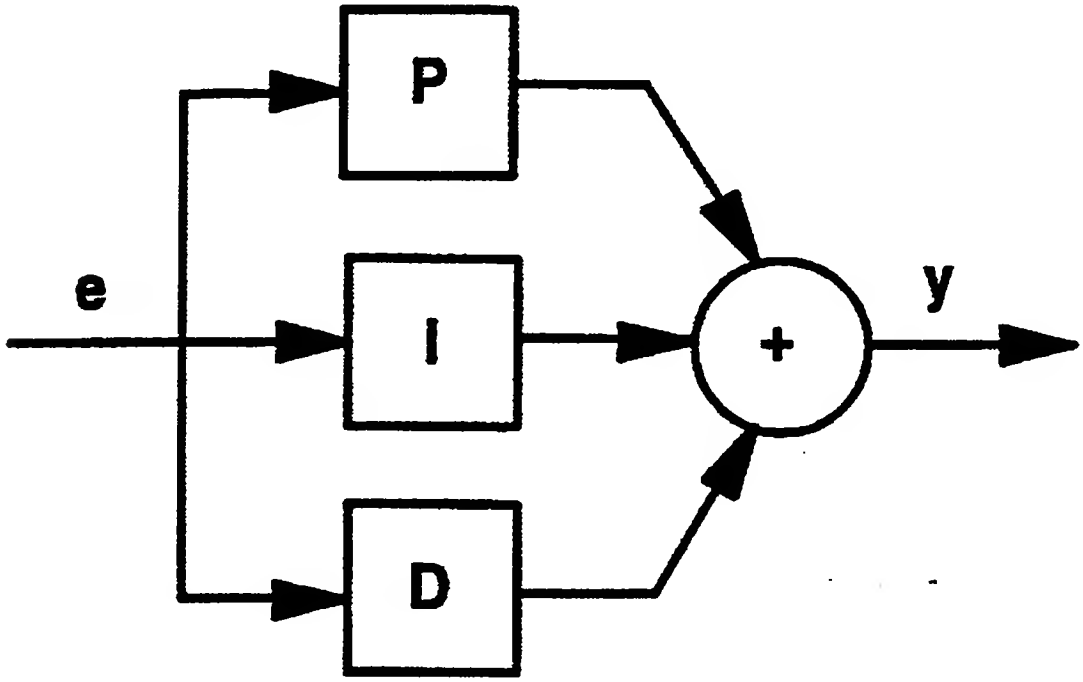
4) **PID - Controller**
The three basic control actions can be used alone (whit the exception of the D-controller) or as combinations like PI-Controller, PD-controller, PID-controller. In this case the total control action is the sum of the different basic control actions which means the controllers are arranged in parallel.
Example: PID - Controller (three mode controller)

Figure 12 PID Controller



The control algorithm when using the parameters Kp, KI, KD:

$$y = Kp \cdot e + KI \cdot \int edt + KD \cdot de/dt$$

The control algorithm when using the parameters Kp, TI, TD:

$$y = Kp \cdot e + (Kp / TI) \cdot \int edt + Kp \cdot TD \cdot de/dt$$

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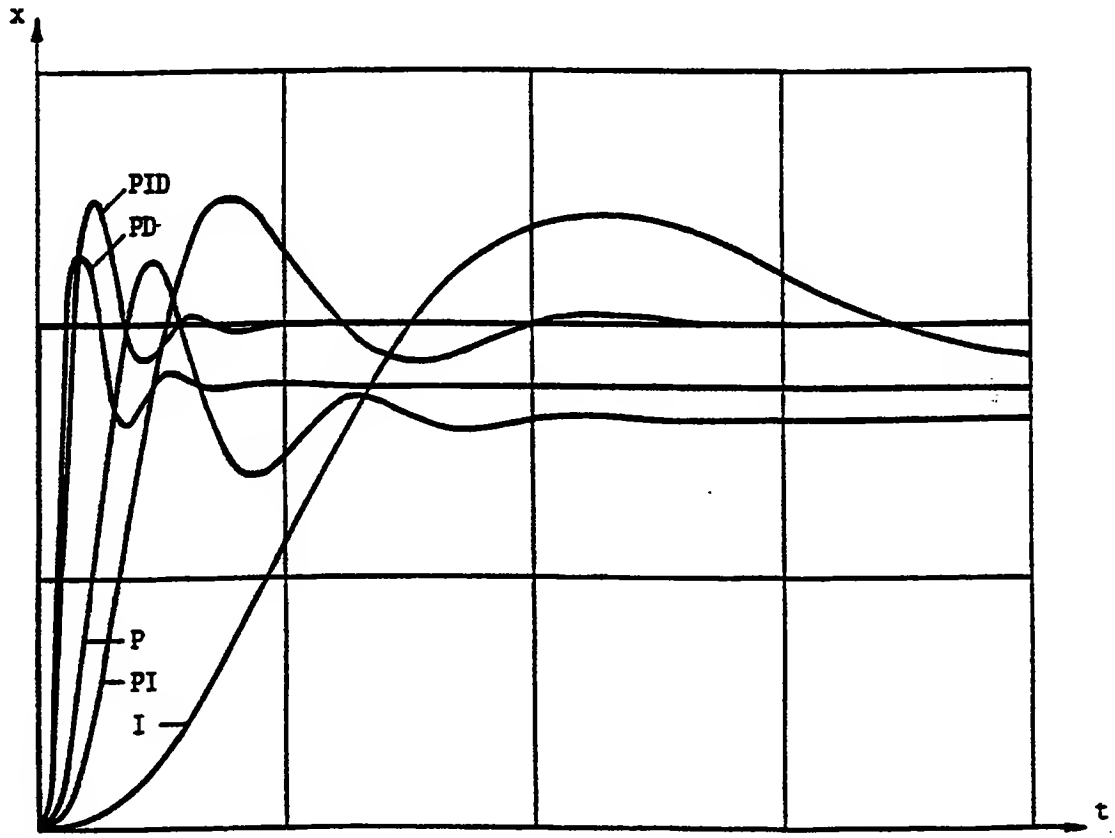
Figure 13 Comparison of the different controllers



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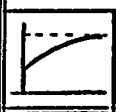
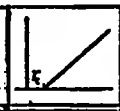
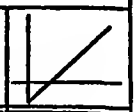
Which controller to choose depends on the plant behavior. Below you will find some guidelines on how to choose a controller for a given plant.

In a cement plant a lot of controllers are just PI - controllers because many of the processes which have to be controlled are relatively slow and thus a derivative control action (D - controller) is not necessary

Table 1 Guidelines how to choose a controller

Type of Plant / Controller Type	STATIC PLANTS					ASTATIC PLANTS			
									
P (PD)	can be used	well suited	suitable	only of limited use	only of limited use	well suited	well suited	suitable	can be used
I	well suited	well suited	only of limited use	only of limited use	well suited	can be used	cannot be used	cannot be used	cannot be used
PI	suitable, usually not required	suitable, usually not required	well suited	well suited	can be used (I-Controller is better	well suited	well suited, usually not required	well suited	well suited
PID	can be used, usually not required	can be used, usually not required	well suited	suitable (if T_t/T small)	cannot be used usually	usually unsuitable	suitable	usually unsuitable	usually unsuitable

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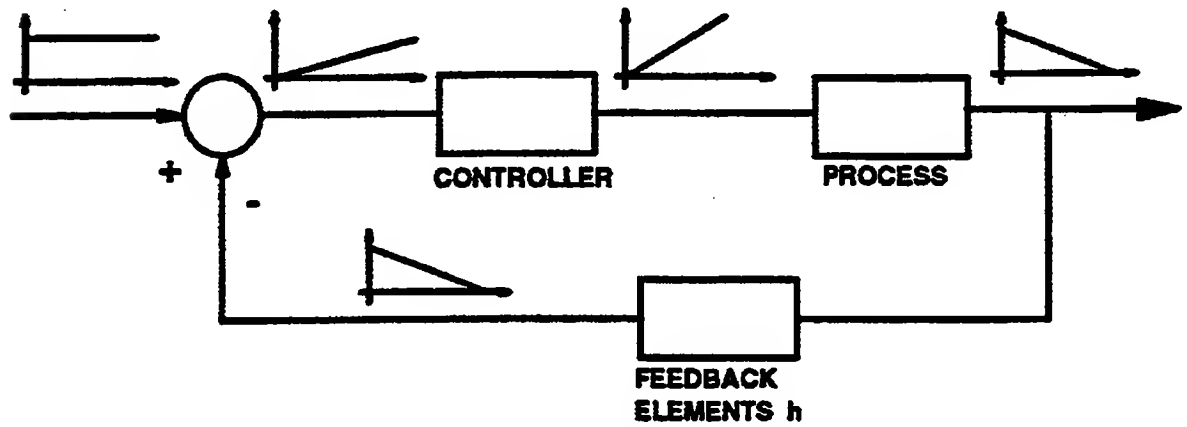
3.3 Analog and Digital Controller

A PID - Controller can be realized as an analog or as a digital controller.

a) **Analog Controller**
The control loop is closed all the time. The control operations are executed at every point of the time scale.
The control algorithms are realized on the hardware side with appropriate circuits. This means: the more sophisticated the control algorithms the bigger the hardware.

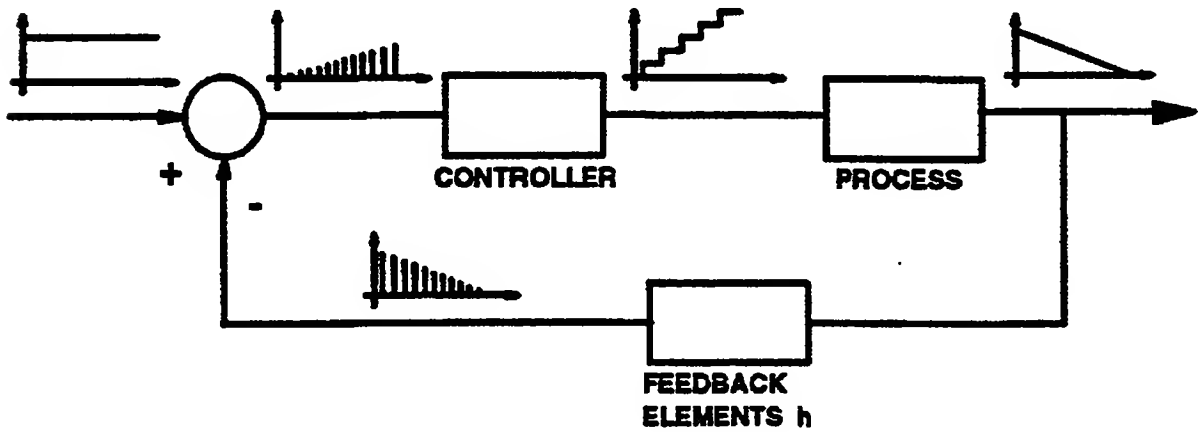
Figure 14 Analog Controller



b) Digital Controller

- The control loop is only closed at certain sampling points. The different arithmetical operations of the control algorithm are not executed simultaneously but one after the other.
- The control algorithms are realized on the software side.

Figure 15 Digital Controller



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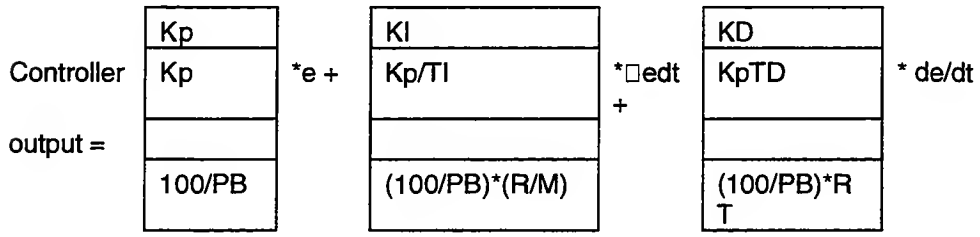
3.4 Terminology

Sometimes it can be very confusing that in the control technique a lot of different terminologies are used. Some of the terms used are listed below. We recommend strongly to stick to the terms used in the other chapters of this paper.



Kp	=	100 / PB	proportional gain or proportional constant
PB	=	Xp	proportional band
KI	=	Kp / TI	integral gain or integral constant
1 / TI	=	R / M	reset gain or reset rate
R / M			reset per minute
TI			integral action time
KD	=	KpTD = (100/PB) *RT	derivative gain or derivative constant
TD	=	RT	derivative action time or rate time or rate gain

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4. HOW TO TURN THE KNOBS - OPTIMAL SETTINGS OF THE CONTROLLER

4.1 Stabilization Point

Two possibilities to interrupt the process:

- changing the setpoint
- disturbing the process

According to settings of the PID parameters Kp, TI, TD the reaction of the control system to such an interaction varies:

Figure 16 Changing the setpoint

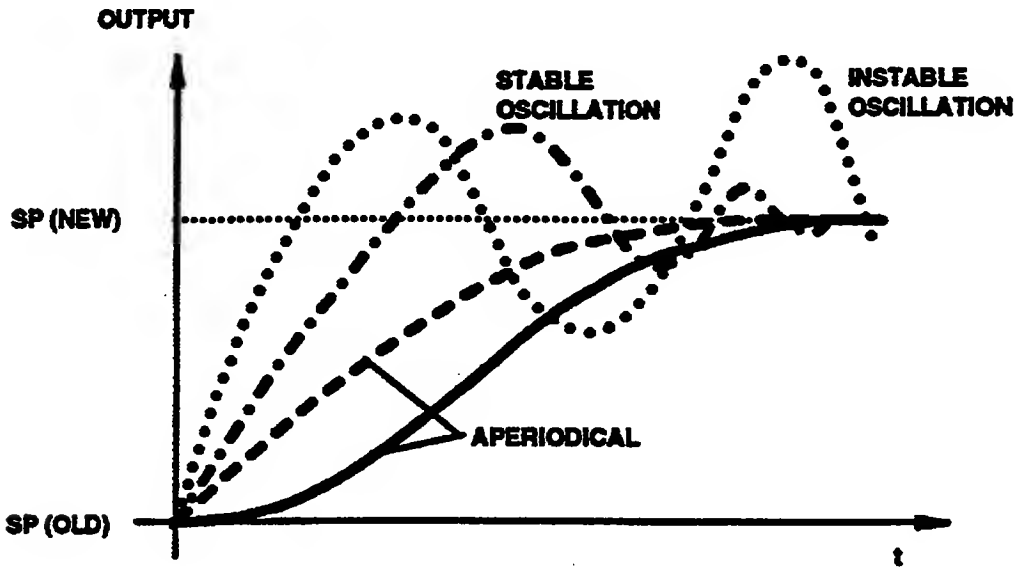
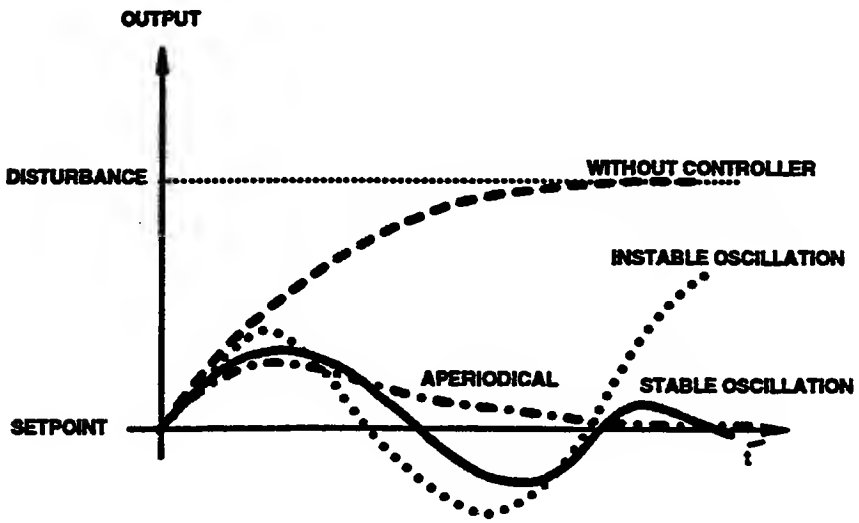


Figure 17 Disturbing



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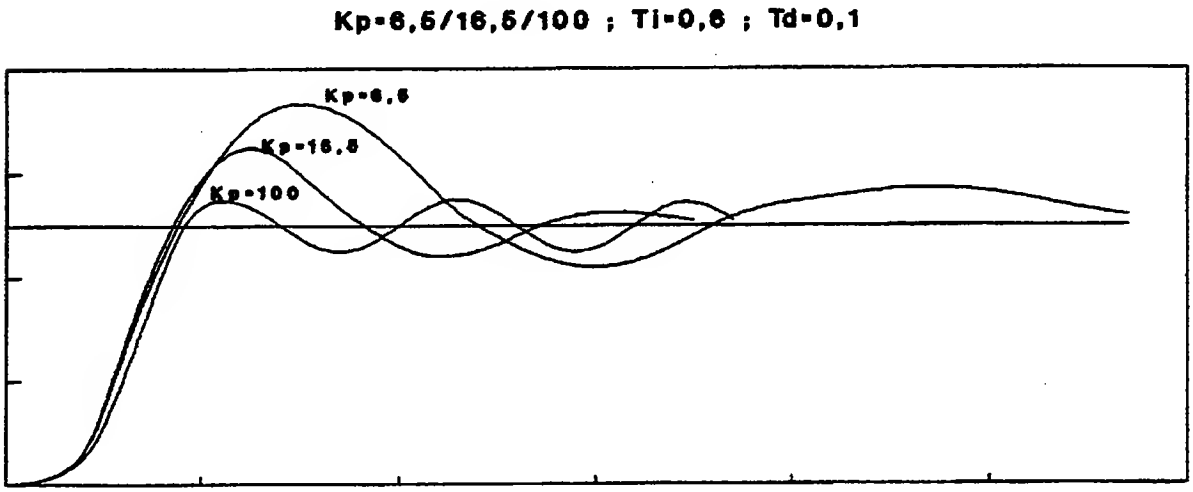
4.2 Adjustments of the PID Settings

the list and the figure below shall give you a better feeling about what happens if you change one of the PID parameters Kp, TI, TD.

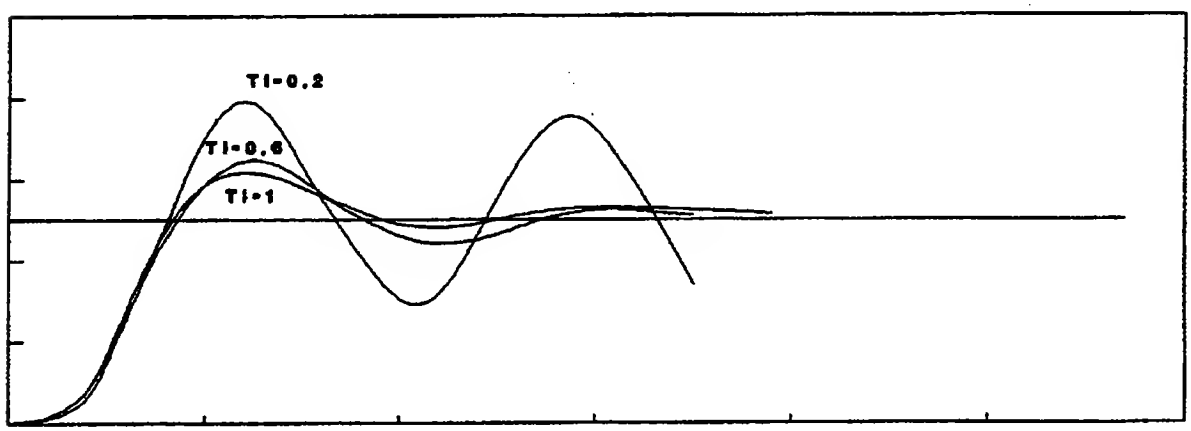
Adjustment		Reaction
Increasing P action	Increasing Kp	Speeding up control action: - smaller amplitude, - smaller period of oscillation Decreasing offset Increasing tendency to oscillate
Decreasing P action	Decreasing Kp	Slowing down control action: - bigger amplitude - bigger period of oscillation Increasing offset Decreasing tendency to oscillate
Increasing I action	Decreasing TI	Bigger amplitude Smaller period of oscillation Faster elimination of the offset Increasing tendency to oscillate
Decreasing I action	Increasing TI	Smaller amplitude Bigger period of oscillation Slower elimination of the off-set Increasing tendency to oscillate
Increasing D action	Increasing TD	Smaller amplitude Bigger Period First degreasing but beyond a certain point increasing tendency to oscillate
Decreasing D action	Decreasing TD	Bigger amplitude Smaller period First increasing, beyond a certain point decreasing tendency to oscillate

Adjustment of the PID Settings Kp



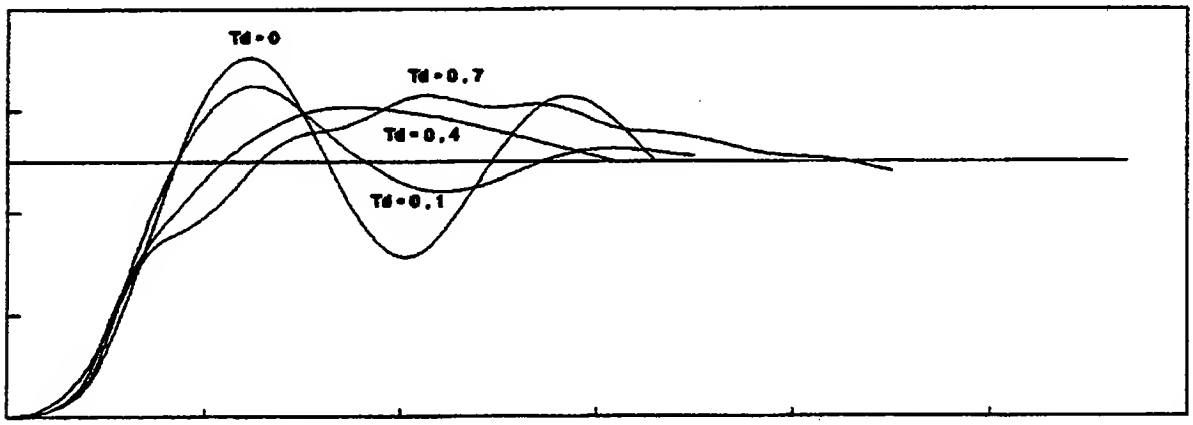
Adjustment of the PID Settings Ti

Kp=16,5 ; Ti=0,2/0,6/1 ; Td=0,1



Adjustment of the PID Settings Td

Kp=16,5 ; Ti=0,6 ; Td=0/0,1/0,4/0,7



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4.3 Optimal Settings of the Controller

Chien-Hrones Method

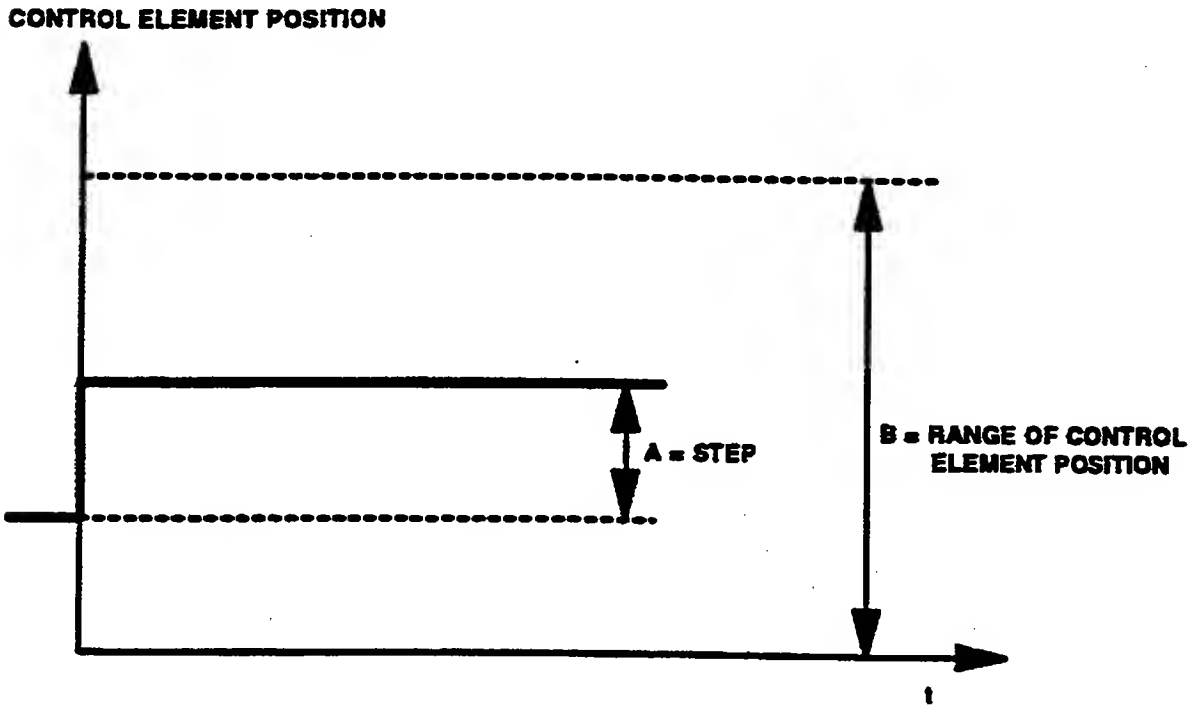
The optimal controller settings are determined from the reaction curve of the opened control loop to a step disturbance. This method is also called

- process reaction curve method
- lag reaction rate method

1) open the loop (switch controller to manual)

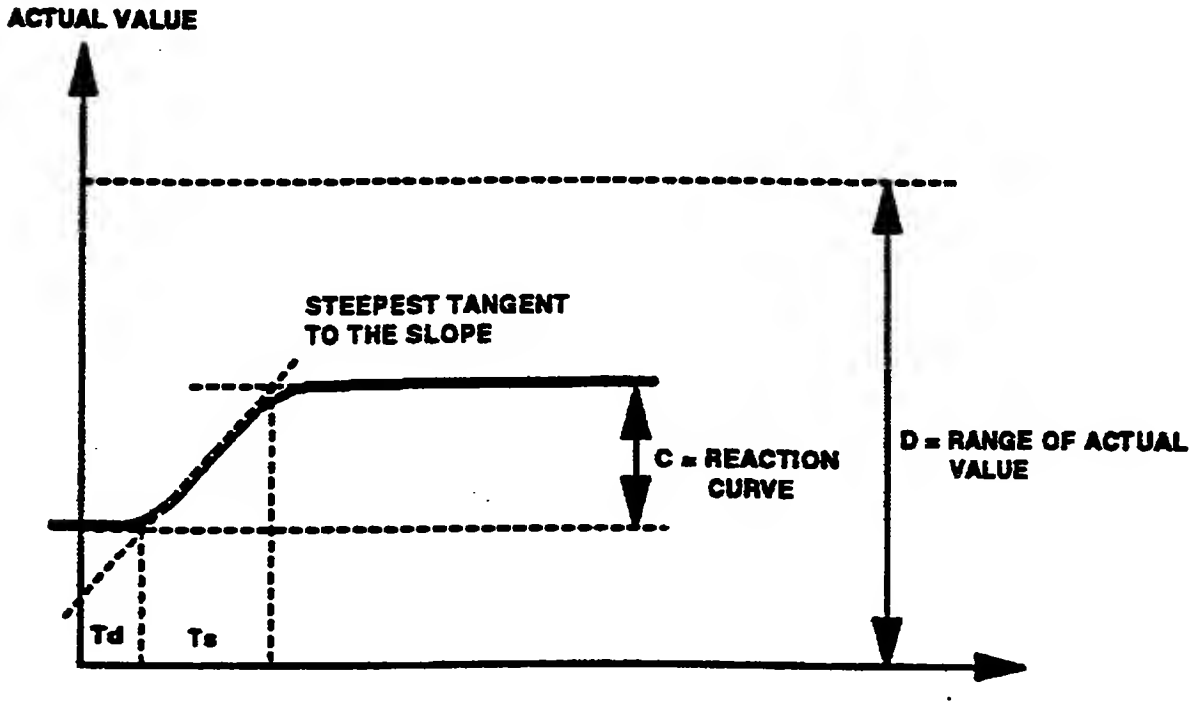
2) create a step on the controller output signal

Figure 18 Step in control element position



3) record the answer of the process to this step

Figure 19 Process reaction curve



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4) **determine the following expressions:**

Td: Dead time [μιν]
Ts: Time constant [μιν]
N: Slope = reaction rate in % of feedback signal range per minute [%μιν]

$$N=\frac{C/D*100}{T_S}$$

□P: % change in control element position [%]

$$\Delta p=\frac{A}{B}*100$$

5) Optimal settings (rough estimation)

$$Output=Kpe+\frac{Kp}{T_I}\int edt+Kp*T_D*\frac{de}{dt}$$

P-Controller	Kp	=	$\frac{\Delta p}{N*Td}$
PI-Controller	Kp	=	$0.92*\frac{\Delta p}{N*Td}$
	TI	=	3.3 * Td
PID-Controller	Kp	=	$1.2*\frac{\Delta p}{N*Td}$
	TI	=	2 * Td
	TD	=	0.5 * Td

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Example: Cooling tower control loop

A step is made in the position of the cooling water control value from 60 % to 50 %. The measured outlet gas temperature signal shows the following picture (the range of the temperature signal is from 100 °C to 300 ° C):

Figure 20

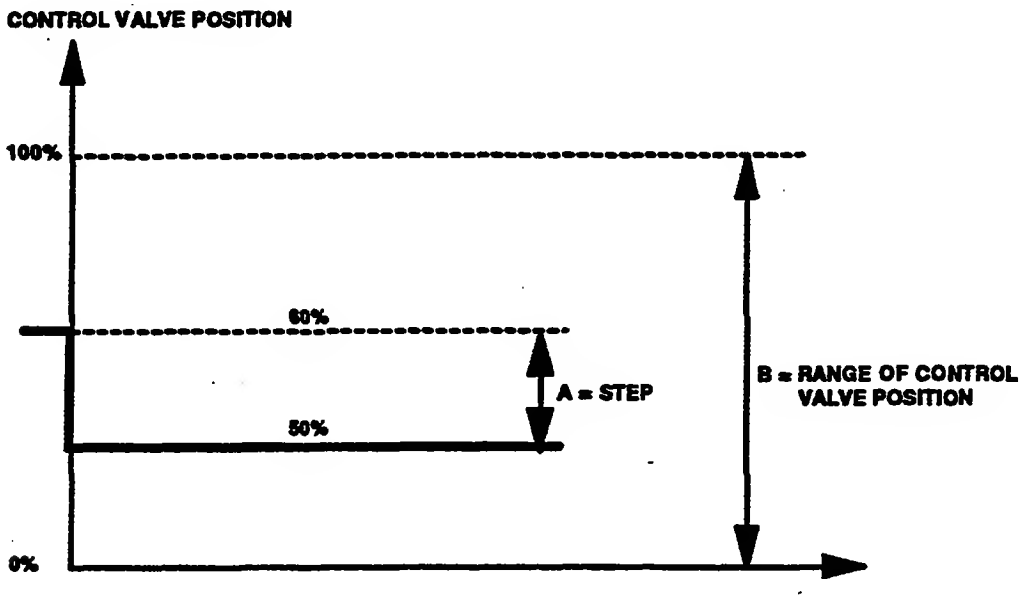
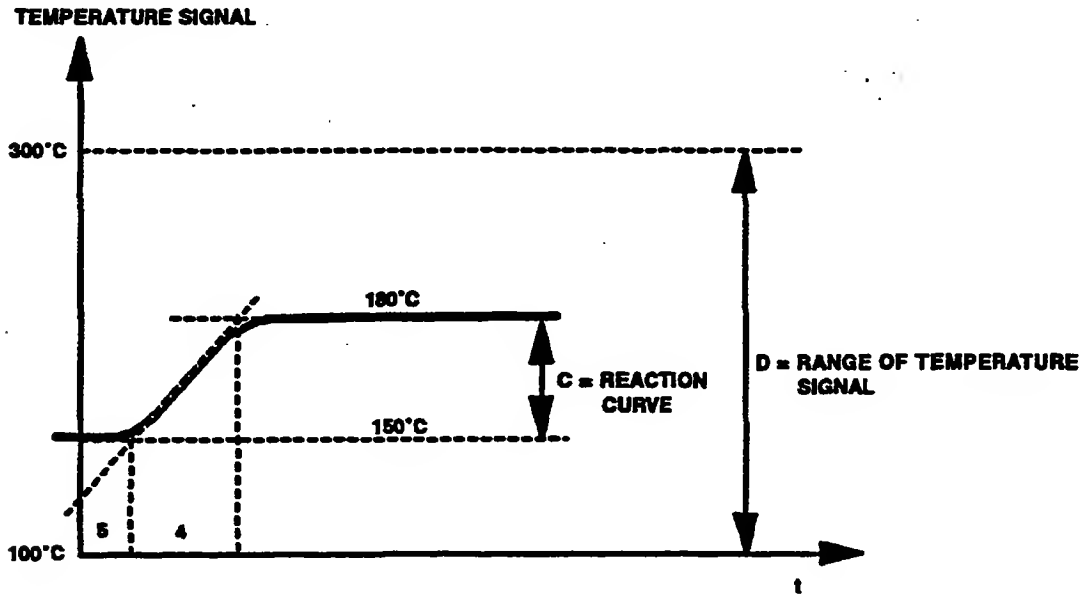


Figure 21



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Td = 5 min.
Ts = 4 min.

$$N=\frac{\frac{180-150}{300-100}*100}{4}=3.75\%/\text{min.}$$

$$\Delta p=\frac{50-60}{100}*100=10\%$$

Optimal Settings for PID-Controller

Kp	=	$\frac{1.2*\Delta p}{N*Td}$	=	$\frac{1.2*10}{3.75*5}$	=	0.64%/%
TI	=	2 * Td	=	2 * 5	=	10 min.
TD	=	0.5 * Td	=	0.5 * 5	=	2.5 min.

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PRACTICAL EXERCISE - Fuzzy-Control

F. Fehr.
98/6346/E

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2.2 Input/output Fuzzyfication 391
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3. Miscellaneous 392

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1. INTRODUCTION

This is an approach to teach the Fuzzy theories in a practical way, without a lot of paper and formulas. On behalf of a Siemens Demo Software, a Fuzzy controller will be shown and worked out empirically.

The goal of this exercise is to get a basic idea about

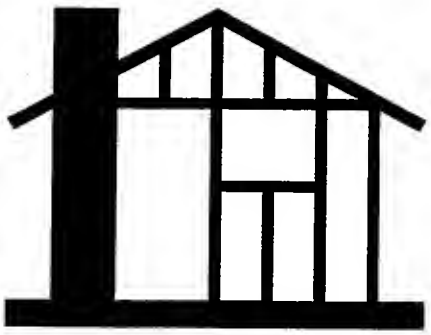
What is Fuzzy-Control

How does it work

There is a diskette included for every participant. Please take this diskette home as an example that you can recall anytime, whenever you are in a position to work with fuzzy-logic or explain to someone else what fuzzy is really about.

The task of this exercise is:

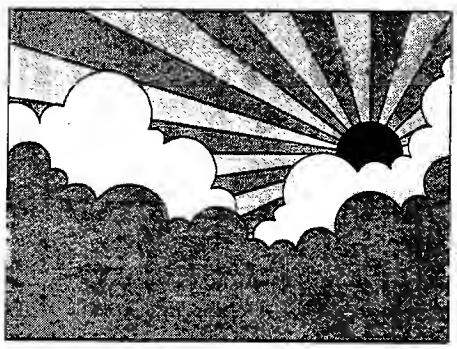
To set-up a temperature controller which uses a room and an outside temperature signal thus actuating a heater and an air-conditioner.



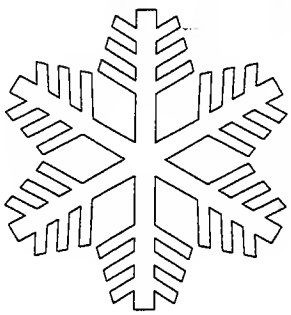
Room Temperature

Controller

Heater



Outside Temperature



Air Conditioner

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2. EXERCISE

2.1 Start / Set-up

- Start Fuzzy.exe
- Create new Project with the name "TEMPCON" ("File" "New...") with 2 Inputs and 2 Outputs.
- Assign names to inputs and outputs (see pictures in 1. Introduction)
- Define 3 temperature ranges (membership functions) for room temp (e.g. chilli, comfortable, warm).
- Define 3 temperature ranges for outside temp (e.g. cold, warm, hot).
- Define 4 ranges for the heater (e.g. off, low, high, on) and 3 ranges for the air conditioner (e.g. off, med, on).
- Adjust the temperature scale to reasonable values (e.g. room temp. from 10...30°C).

2.2 Input/output Fuzzyfication

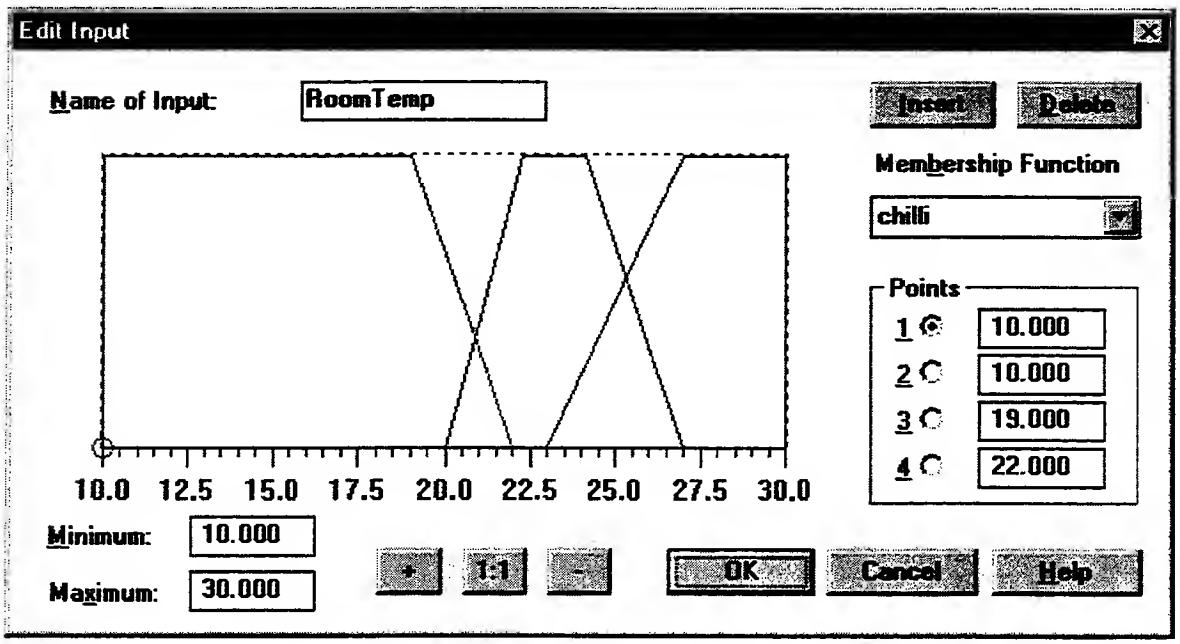
Perform the Fuzzyfication of the 2 inputs and 2 outputs by asking the following questions and set the curves accordingly:

... where is 0% cold...

... from where to where is 100% warm...

Out of 100 people, how many people would say that is chilli (first curve) at what temperature. Do the same for the other 2 curves. The number of people (in %) which is actually the degree of fulfilment (or the percentage of truth) is on the vertical axis, the temperature on the horizontal axis.

On the example in the next picture below, it is 50% chilli (50 out of 100 people would call it chilli) at 20.5°C and after 22°C it is not chilly anymore.



2.3 Implementing rules

Now fill in the rules in "If...then".

2.4 Testing

Use the menu "View" to simulate input values as constants or curves (sub menu "Curve Parameters...") and test the controller (sub menu "Curve Recorder").

2.5 Save

Save your project ("File" "Save" or "File" "Save As...").

For those who could not work through the exercise, there is an example of this exercise on the diskette, named exercise.txt. Rename it to *.fpl to be able to call it from the demo software.

3. MISCELLANEOUS

On the Cement Course CD-Rom, the directory \MISC\FUZZY contains all files and information being used for the practical work of fuzzy control logic described in the chapters above. Below a list of all files of the directory \MISC\FUZZY

Files	Description
FUZZY.EXE	program file for the Fuzzy demo
FUZZY.ICO	icon for the above file
README.TXT	supplementary supplier information about the fuzzy package
SHUTTERS.FPL	example "shutters controller"
PENDULUM.FPL	example "inverse pendulum"
EXERCISE.TXT	example "temperature control"
S7FUZ1EF.PPT	Power Point slides about the fuzzy package

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Gas Flow Measurement - Practical Work

H. Nyfenegger
PT 97/14231/E

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SUMMARY

This paper shows the basic equations for gas flow measurement. Furthermore, the principles of gas flow measurement by the Prandtl tube as well as the Orifices and Nozzles are explained.

Some of the presented principles and instruments will be used for the practical work carried out on a test equipment in the laboratory hall.

1. THEORETICAL BASIS

1.1 The Law of Bernoulli

(Daniel Bernoulli, 1700..1782 Swiss Scientist)

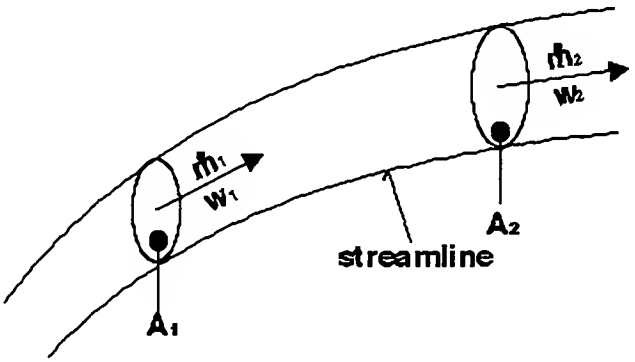
In aeronautics, a law or theorem state that in a flow of incompressible fluid the sum of the static pressure and the dynamic pressure along a streamline is constant if gravity and frictional effects are disregarded.

Resultant from this law is that if there is a velocity increase in a fluid flow, there must be a corresponding pressure decrease. Thus an airfoil, by increasing the velocity of the flow over its upper surface, drives lift from the decreased pressure.

As originally formulated, a statement of the energy conservation (per unit mass) for a non-viscous fluid in steady motion. The specific energy is composed of the kinetic energy $v^2/2$, where v is the speed of the fluid; the potential energy gz, where g is the acceleration of gravity and z is the height above an arbitrary reference level; and the work done by the pressure forces of a compressible fluid $\int v\,dp$, where p is the pressure, v is the specific volume, and the integration is always with respect to value of p and v on the same parcel. Thus, the relationship

$$v^2/2 + gz + \int v\,dp = \text{constant along a streamline}$$

is valid for a compressible fluid in steady motion, since the streamline is also in path. If the motion is irrational, the same constant holds for the entire fluid.



If the fluid is compressible

$\dot{m}_1=\dot{m}_2$

$\rho_1 \cdot \dot{V}_1=\rho_2 \cdot \dot{V}_2$

$\rho_1 \cdot A_1 \cdot w_1=\rho_2 \cdot A_2 \cdot w_2$

If the fluid is incompressible:

$\rho_1=\rho_2$

$\dot{V}_1=\dot{V}_2$

$A_1 \cdot w_1=A_2 \cdot w_2$

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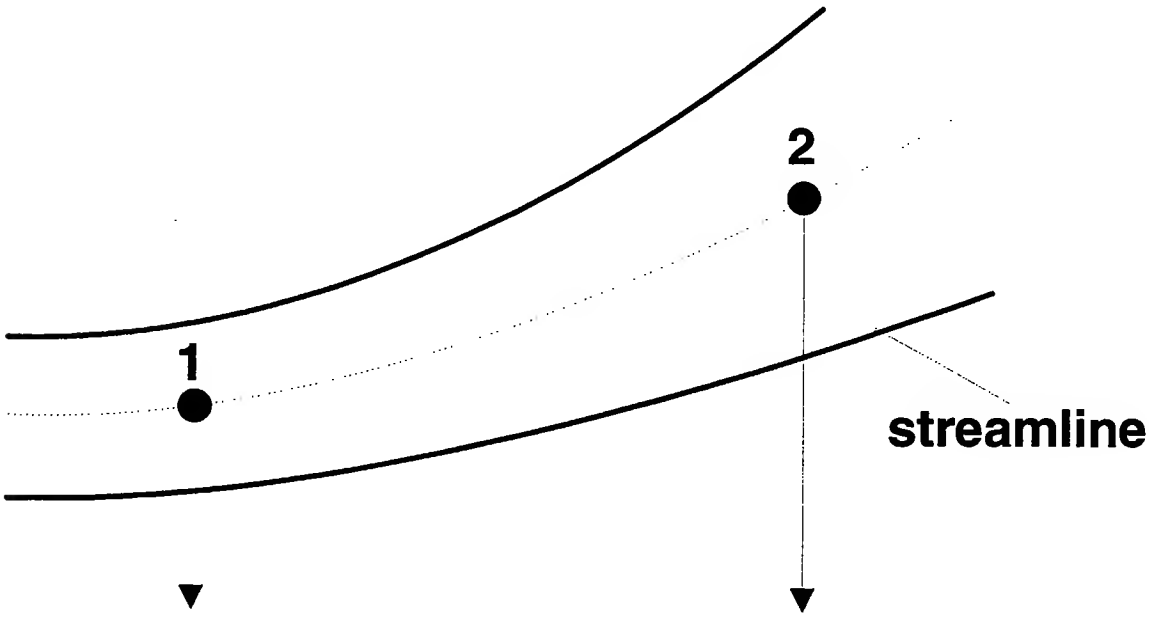

Sum of energies in point 1 = Sum of energies in point 2

$$\frac{P_1}{\rho_1} + \frac{w_1^2}{2} = \frac{P_2}{\rho_2} + \frac{w_2^2}{2}$$

If the fluid is incompressible, that means ρ is constant:

$$\rho_1=\rho_2=\rho$$

$$\frac{P_1}{\rho} + \frac{w_1^2}{2} = \frac{P_2}{\rho} + \frac{w_2^2}{2}$$

Symbols and units:

A	Cross section area	[m^2]
d	Diameter of the smallest aperture	[m]
D	Diameter of the tube	[m]
m	Cross section area ratio $m = d^2 / D^2$	[-]
$\dot{m}$	Mass flow rate	[kg/s]
p	Pressure	N / m^2]
$\dot{V}$	Volume flow rate	[m^3 / s]
w	Velocity	[m / s]
ρ	Density	[kg / m^3]

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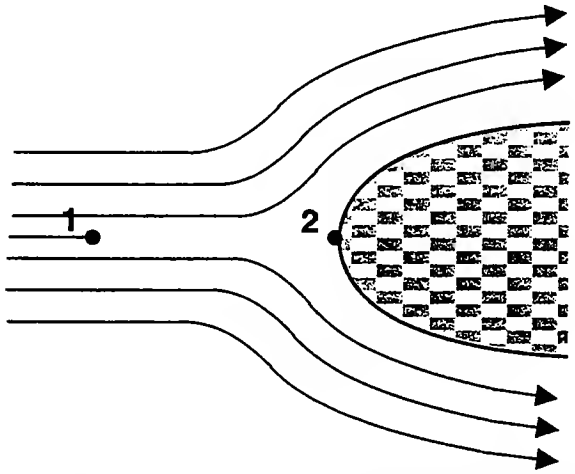
2. LOCAL VELOCITY AND PRESSURE MEASUREMENTS



The above figure shows a fluid flowing around an obstacle. One of the streamlines reaches the body at the point 2, called stagnation point. At this point the velocity of the fluid is zero.

The Bernoulli equation, for the undisturbed flow at point 1 to point 2,

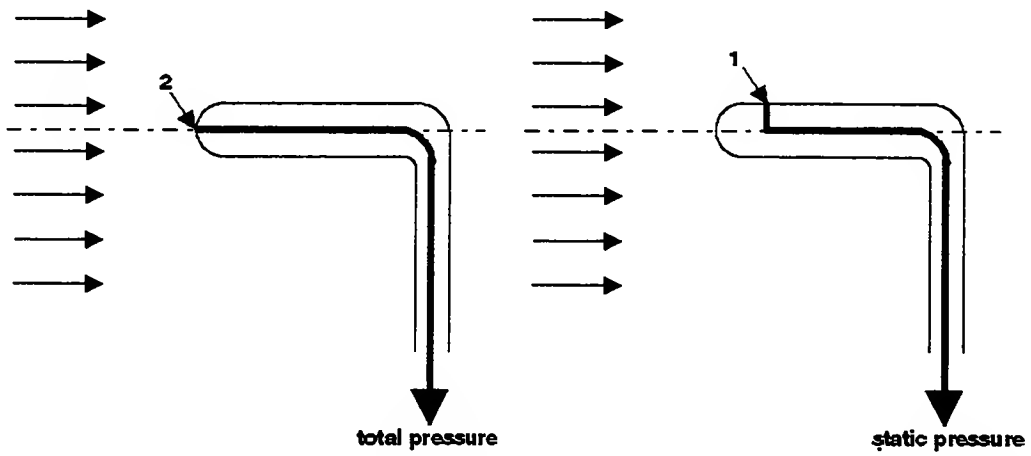
$$\frac{P_1}{\rho}+\frac{w_1^2}{2}=\frac{P_2}{\rho}+\frac{w_2^2}{2} \quad \text{or } (w_2=0) \quad P_2=P_1+\frac{\rho}{2}w_1^2$$

where P_2 is the total pressure
P_1 is the static pressure
$\frac{\rho}{2}w_1^2$ is the dynamic pressure

p_2, p_1 and $\frac{\rho}{2}w^2$ are pressures called total, static and dynamic pressure.

2.1 The Measurement of the Pressures

The total pressure of a fluid can be measured at a stagnation point. For this purpose the so-called Pitot tube is used.



At the point 2 a stagnation point is created where the **total pressure** can be measured.

A boring placed vertically to the flow (1) measures the pressure independent from the direction. This pressure is called **static pressure**.

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3. GAS FLOW MEASUREMENT BY THE PRANDTL TUBE

The dynamic pressure, defined as

$$\frac{\rho}{2} \bullet w_1^2$$

can be expressed as the pressure difference between the total and the static pressure. The relation is

$$p_2 - p_1 = \frac{\rho}{2} w_1^2$$

This difference is measured with the Prandtl Tube.

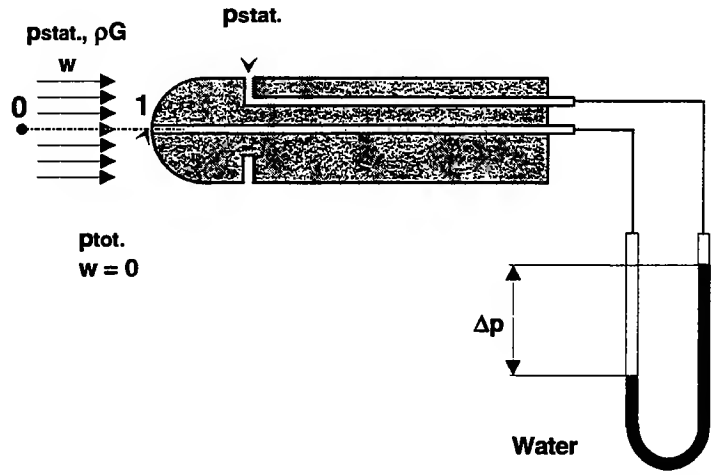


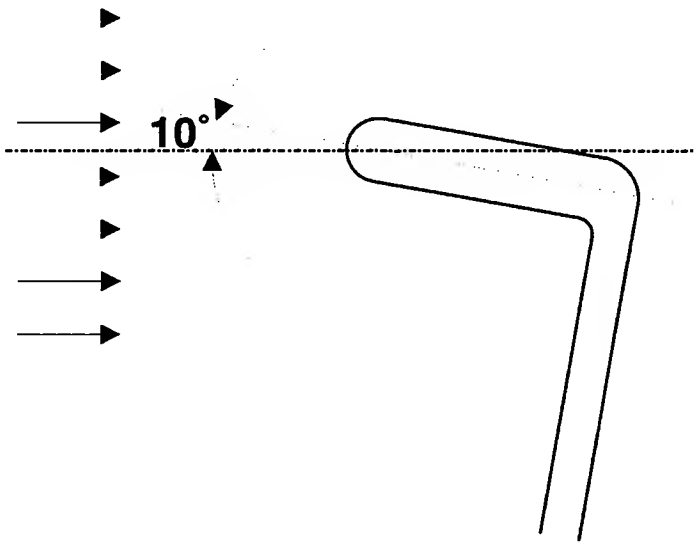
Fig.2

The inner part of this instrument corresponds exactly to a Pitot Tube and the total pressure is measured there. The outer part has the function of the static pressure measurement.

The difference between the two connections is the dynamic pressure.

The direction of the flow should correspond to the axis of the Prandtl Tube. Deviations until ± 10° however, do not essentially influence the measuring accuracy.

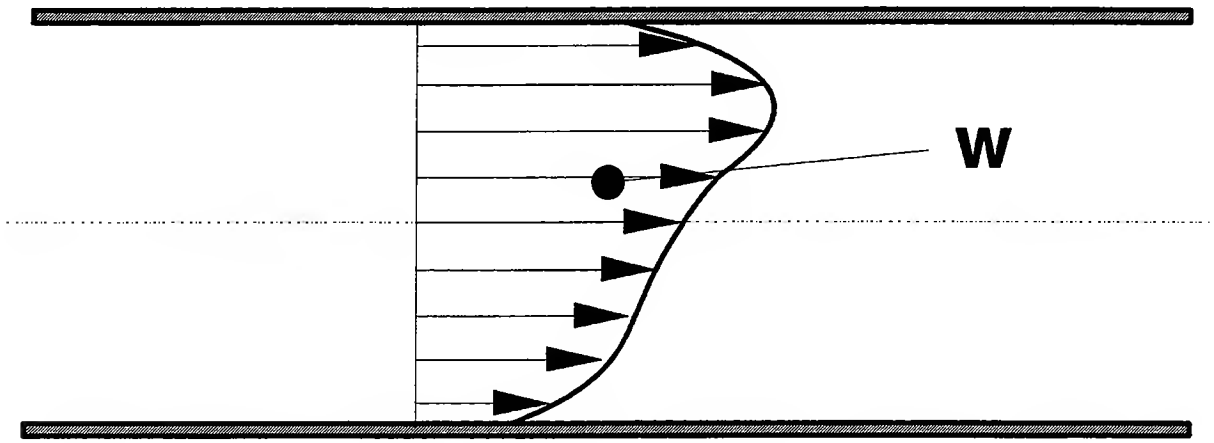
The error in the measurement of the pressure is below ± 1%.



With the dynamic pressure the fluid velocity can be calculated as follows:

$$w_1 = \sqrt{\frac{2 \cdot (P_2 - P_1)}{\rho}}$$

The Prandtl Tube measures a local velocity only. In order to get an integral velocity (e.g. the average value) the integration of several local velocities over the cross sectional area must be made.



$$w = \frac{1}{A} \iint_A w \cdot dA$$

The gas flow rate is finally calculated by multiplying the average gas velocity with the cross sectional area.

Specific information on the gas flow measurement by the Prandtl Tube is given in chapter 6 "Practical work in gas flow measurement".

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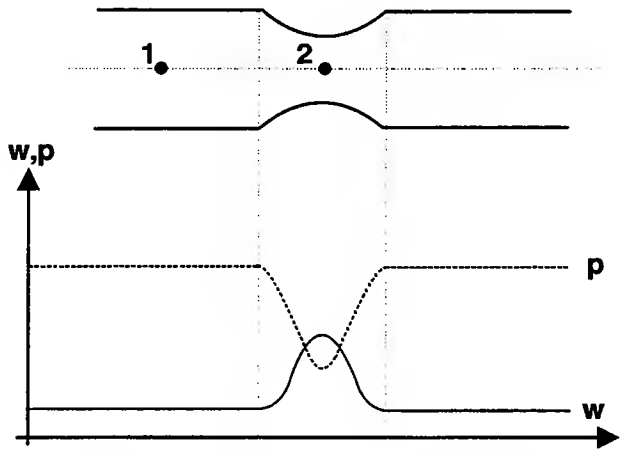
4. INTEGRAL GAS FLOW MEASUREMENT WITH NOZZLES AND ORIFICES

The idea of this method is to create a change of cross section of the whole fluid stream. By means of this change also the pressure and the velocity change and from the pressure difference the velocity can be calculated.

A precondition for this measurement method is a more or less equalized velocity profile. Disturbances like

- Flow separation behind bends, valves, etc.
- Pulsation behind a piston type compressor
- High dust loading
- etc.

falsify the measurement and make it useless in a lot of cases. As a rule of thumb it can be said that prior to such a flow measurement a straight tube length of more than 10 tube diameters is to be provided.



Between point 1 and 2 the continuity condition can be formulated as

$$A_1 \cdot w_1 = A_2 \cdot w_2$$

Between the same two points the Bernoulli equation (see 1.1) gives:

$$\frac{p_1}{\rho} + \frac{w_1^2}{2} = \frac{p_2}{\rho} + \frac{w_2^2}{2}$$

From the two relations the following equation is derived:

$$w_1 = \sqrt{\frac{2 \cdot (P_1 - P_2)}{\rho \cdot \left[\left(\frac{A_1}{A_2}\right)^2 - 1\right]}}$$

The velocity w_1 can be calculated from the pressure difference between the undisturbed flow (point 1) and the accelerated flow in the smallest cross-sectional area.

The pressure p_2 is lower than pressure p_1 and compared to the pressure p_1, is p_2 a negative pressure.

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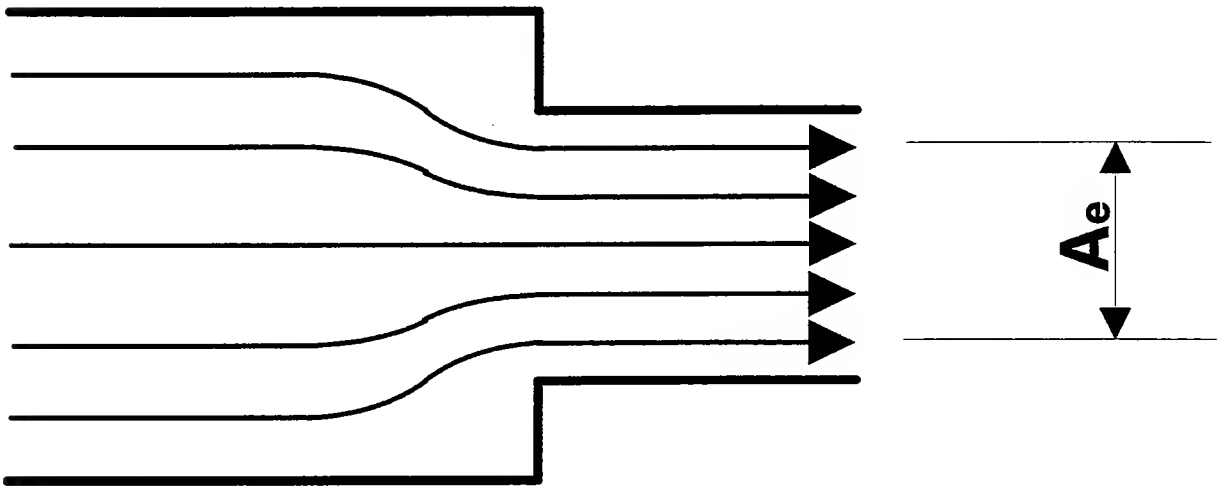
The relation is

$$(P_1 - P_2) = \frac{\rho}{2} W_1^2 \left[\left(\frac{A_1}{A_2} \right)^2 - 1 \right]$$

The pressure difference is a function of the ratio square of the cross sectional area. Great ratios produce therefore extreme underpressures! An example of such a great ratio is the reduction of the cross-sectional area during coating formation in preheaters.

The change of the cross-section causes, especially when it is done suddenly, a disturbance of the flow. The fluid cannot follow the wall and a contraction of the stream is produced.



The effective cross section A_e is now smaller than the real one and the calculated velocity must be corrected. The correction cannot be calculated, it must be measured. For this reason several types of integral measuring instruments are standardized by DIN 1952.

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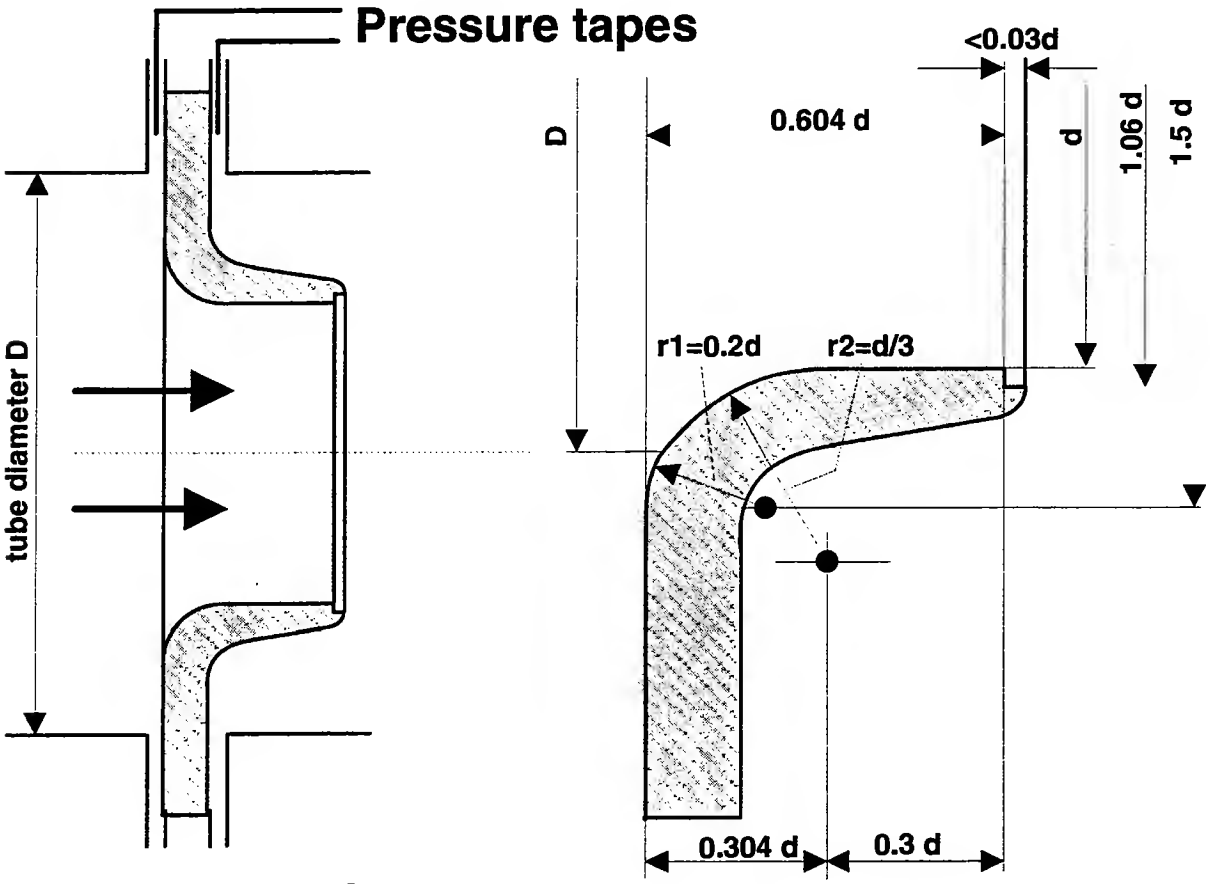
In the following the 3 basic standard instruments are sketched:

4.1 Standard Nozzles

Standard nozzles are used for tubes with diameters from 50 to 500 mm and a cross-sectional ratio (smallest area against tube area) of 0.1 to 0.64.



$$m = \left(\frac{d}{D}\right) = 0.444$$

The velocity in the tube and the volume flow, respectively, are calculated by the following relation:

$$w = \alpha \cdot \varepsilon \cdot m \cdot \sqrt{\frac{2 \cdot \Delta p}{\rho_1}}$$

$$\dot{V} = \alpha \cdot \varepsilon \cdot m \cdot \frac{\pi}{4} \cdot D^2 \cdot \sqrt{\frac{2 \cdot \Delta p}{\rho_1}}$$

The factor ε considers the change in the density of a compressible fluid. For incompressible fluids (e.g. water or air until a maximum velocity of 100 [m/s]) this factor has the value 1, otherwise it is tabulated in the DIN - Standards 1952.

The value α is the correction factor for the contraction of the fluid. It is a function of the Reynold's number and the ratio m of the cross-sectional areas. This factor is also tabulated in the DIN - Standards 1952. The table below shows an output of these standards.

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Correction factors for the contraction $\alpha = f(m^2, Re)$ for standard nozzles in tubes with smooth surfaces, valid for diameters D between 50 and 500 mm. Between the indicated values of m^2 (not m) can be interpolated linearly.

Re		2 · 10⁴	2,6 · 10⁴	3 · 10⁴	4 · 10⁴	6 · 10⁴	7 · 10⁴	10⁵	2 · 10⁵	10⁶ bis 2 · 10⁶
m	m²	α								
0,1000	0,01						0,9892	0,9898	0,9898	0,9898
0,1414	0,02						0,9917	0,9924	0,9927	0,9928
0,1732	0,03						0,9948	0,9954	0,9959	0,9960
0,2000	0,04	0,9798	0,9849	0,9883	0,9926	0,9961	0,9973	0,9984	0,9992	0,9994
0,2238	0,05	0,9822	0,9871	0,9906	0,9951	0,9977	1,0002	1,0018	1,0026	1,0027
0,2449	0,06	0,9849	0,9898	0,9930	0,9978	1,0008	1,0033	1,0047	1,0059	1,0061
0,2648	0,07	0,9878	0,9921	0,9956	1,0002	1,0033	1,0064	1,0080	1,0093	1,0096
0,2828	0,08	0,9907	0,9961	0,9984	1,0031	1,0063	1,0096	1,0113	1,0128	1,0130
0,3000	0,09	0,9939	0,9982	1,0014	1,0060	1,0093	1,0128	1,0147	1,0183	1,0166
0,3162	0,10	0,9973	1,0015	1,0048	1,0092	1,0128	1,0162	1,0182	1,0199	1,0202
0,3317	0,11	1,0009	1,0050	1,0080	1,0128	1,0189	1,0196	1,0217	1,0235	1,0238
0,3464	0,12	1,0048	1,0088	1,0118	1,0160	1,0194	1,0230	1,0253	1,0272	1,0276
0,3606	0,13	1,0088	1,0123	1,0153	1,0197	1,0230	1,0266	1,0290	1,0309	1,0312
0,3742	0,14	1,0129	1,0163	1,0192	1,0238	1,0287	1,0303	1,0328	1,0347	1,0380
0,3873	0,15	1,0173	1,0208	1,0234	1,0274	1,0308	1,0341	1,0388	1,0385	1,0388
0,4000	0,16	1,0219	1,0251	1,0278	1,0316	1,0345	1,0380	1,0405	1,0424	1,0427
0,4123	0,17	1,0266	1,0297	1,0321	1,0358	1,0388	1,0420	1,0445	1,0463	1,0467
0,4243	0,18	1,0315	1,0344	1,0367	1,0402	1,0428	1,0461	1,0488	1,0504	1,0507
0,4359	0,19	1,0368	1,0393	1,0418	1,0447	1,0472	1,0503	1,0527	1,0545	1,0547
0,4472	0,20	1,0418	1,0444	1,0464	1,0494	1,0517	1,0546	1,0569	1,0588	1,0589
0,4583	0,21	1,0472	1,0498	1,0518	1,0543	1,0563	1,0590	1,0612	1,0628	1,0631
0,4690	0,22	1,0528	1,0550	1,0567	1,0593	1,0611	1,0638	1,0656	1,0671	1,0674
0,4796	0,23	1,0586	1,0606	1,0621	1,0644	1,0660	1,0682	1,0701	1,0715	1,0716
0,4899	0,24	1,0648	1,0662	1,0677	1,0697	1,0710	1,0730	1,0746	1,0760	1,0782
0,5000	0,25	1,0706	1,0721	1,0734	1,0761	1,0763	1,0779	1,0793	1,0805	1,0807
0,5099	0,26	1,0769	1,0782	1,0792	1,0808	1,0816	1,0830	1,0841	1,0852	1,0854
0,5196	0,27	1,0833	1,0944	1,0853	1,0864	1,0871	1,0881	1,0890	1,0899	1,0901
0,5292	0,28	1,0899	1,0908	1,0914	1,0923	1,0928	1,0934	1,0941	1,0946	1,0949
0,5388	0,29	1,0966	1,0972	1,0976	1,0982	1,0986	1,0989	1,0993	1,0998	1,0999
0,5477	0,30	1,1035	1,1037	1,1039	1,1042	1,1043	1,1046	1,1048	1,1049	1,1049
0,5566	0,31	1,1106	1,1106	1,1108	1,1104	1,1102	1,1101	1,1101	1,1101	1,1101
0,5657	0,32	1,1179	1,1178	1,1173	1,1168	1,1184	1,1169	1,1166	1,1165	1,1154
0,6745	0,33	1,1253	1,1246	1,1241	1,1233	1,1226	1,1218	1,1214	1,1209	1,1208
0,5831	0,34	1,1329	1,1320	1,1312	1,1300	1,1290	1,1279	1,1272	1,1286	1,1264
0,6918	0,35	1,1407	1,1394	1,1384	1,1388	1,1386	1,1341	1,1332	1,1324	1,1321
0,6000	0,38	1,1488	1,1470	1,1487	1,1436	1,1423	1,1408	1,1394	1,1383	1,1379
0,6083	0,37	1,1567	1,1548	1,1532	1,1610	1,1493	1,1472	1,1467	1,1445	1,1439
0,8164	0,38	1,1650	1,1627	1,1609	1,1583	1,1564	1,1540	1,1523	1,1506	1,1501
0,8248	0,39	1,1734	1,1709	1,1688	1,1658	1,1636	1,1609	1,1590	1,1573	1,1665
0,6325	0,40	1,1821	1,1793	1,1768	1,1735	1,1711	1,1680	1,1660	1,1641	1,1630
0,8403	0,41	1,1909	1,1677	1,1881	1,1813	1,1786	1,1764	1,1732	1,1710	1,1698

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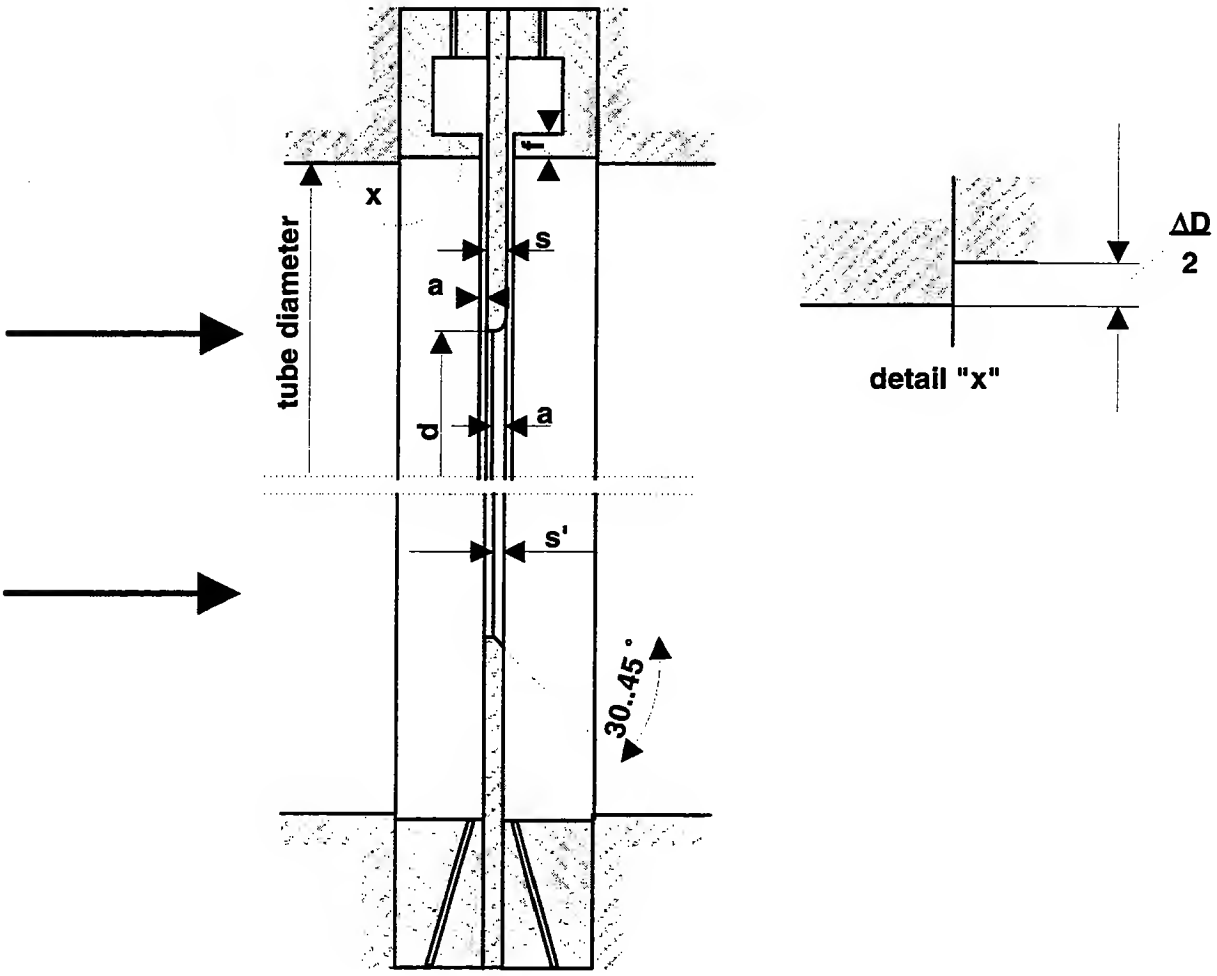
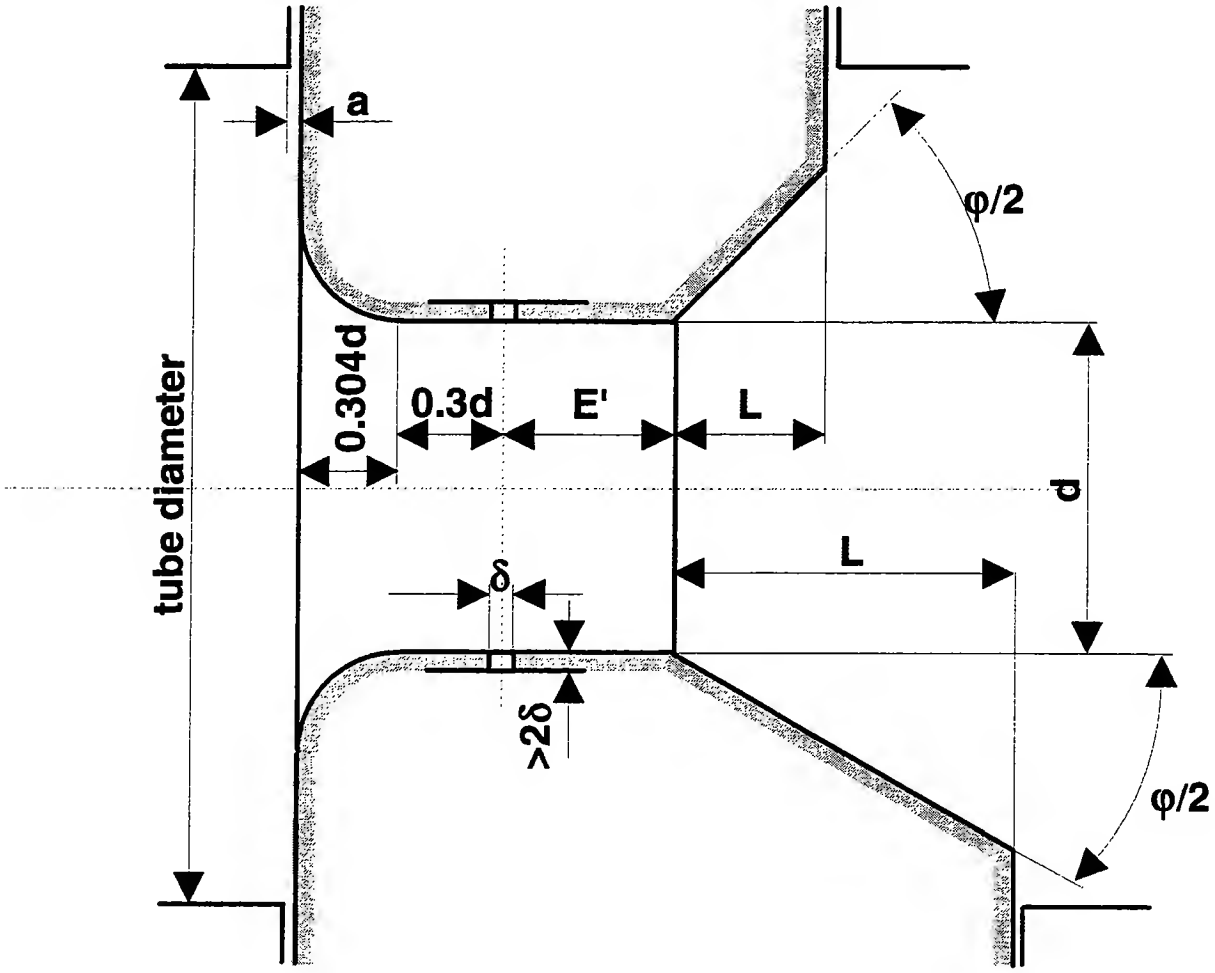
4.2 Standard Orifices and Venturi-Nozzles

Orifices and Venturi-nozzles are standardized in the same way as above-mentioned standard nozzles.

The following figure shows the two standard orifices distinguished only in the pressure taps.



The following figure shows the two used Venturi nozzles distinguished only in the length of the diffuser part.



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5. CALCULATION OF DENSITY

5.1 Requests, Calculation of Real Conditions

The methods based on the law of Bernoulli needs the knowledge of the density ρ of gas. For simple gases like dry air this request makes normally no problem. In case of kiln gases e.g. the density must be evaluated or approximated to the real condition.

$$p_0 = 760[Torr] = 760[mmHg] = 1.01325 * 10^5 \left[\frac{N}{m^2}\right] or [Pa]$$

$$T_0 = 273.16[°K] = 0[°C]$$

$$1[mmHg] \cong 133.322 \left[\frac{N}{m^2}\right] or [Pa]$$

In case of air and in function of the relative humidity φ is almost invariant:

Table 1

φ [%]	ρ_o [kg/m^3]
100	1.290
80	1.290
60	1.291

Thus, the actual gas density ρ_G can be found with the following correlation:

$$\rho_G = \rho_0 \frac{T_0 \cdot p_G}{T_G \cdot p_0} = 1.29 \frac{273.16 \cdot \rho_G}{T_G \cdot 1.01325 \cdot 10^5}$$

where p_G = actual static gas pressure [N/m^2]
T_G = actual gas temperature [K]

P_G can be measured by adding the differential pressure between tube and the ambient to the barometric pressure.

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5.2 Gas Compositions

Mainly in the field of kiln, the gas compositions can be very different so that their density must be determining. The greatest influences to the density come from the components carbondioxide (CO2) and water content (H2O, humidity). Often both components are unknown and can't be measured easily, so that the user depend from approximations.

5.2.1 Calculation of a Heterogeneous Gas

Density of dry gas composition: ρ_{ndr}

Component	Content [%]	Factor	Part of	Unit
CO_2		1.977		[kg/m^3,N, dr.]
O2		1.429		[kg/m^3,N, dr.]
CO		1.25		[kg/m^3,N, dr.]
N2		1.257		[kg/m^3,N, dr.]
	Σ 1.000	ρ_{ndr} =		[kg/m^3,N, dr.]

Density of wet gas composition:

$$\rho_{nw} = \frac{\rho_{ndr} + W_o}{1 + f_{dr}}$$

Where W_0 = Water content [kg/m^3 n,dry]

$f_{dr} = \frac{W_o}{0.804}$ [m^3/m^3 n,dry]

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6. PRACTICAL WORK IN GAS FLOW MEASUREMENT

6.1 Test Equipment

The measurement of the velocity profile and the volume flow is shown on a special test equipment which is sketched in the following figure.

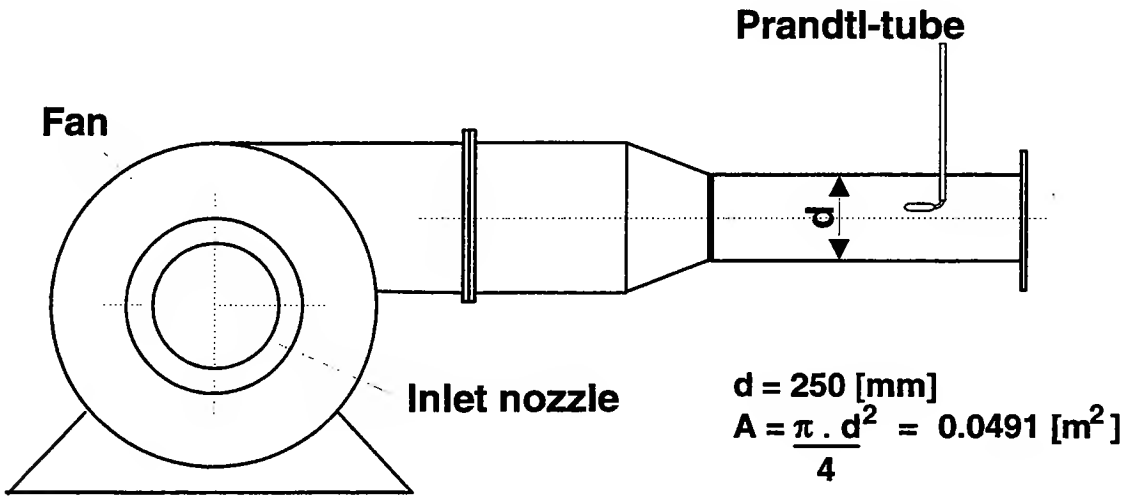

Fig. 1

The following measuring methods are demonstrated and applied:

1) **Prandtl Tube:** The velocity will be measured on several points of the cross sectional area by means of which the average velocity and the volume rate will be calculated.
2) **Anemometers:** At the outlet of the test equipment the mean velocity will be measured with two different anemometers.
3) **Fan Characteristics:** With the knowledge of the characteristic curves of the fan given by the supplier, the volume rate is determined by the pressure difference of the fan and the revolutions of the fan.
4) **Inlet Nozzle (Piezometer):** At the suction side of the fan an inlet nozzle is attached. This nozzle is calibrated and the volume rate can be calculated by means of the pressure difference between the nozzle inlet and ambient.

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6.2 The Prandtl Tube

Prandtl tubes are mainly used for speed measurements of gaseous mediums. The application of a Prandtl tube is suitable for following measurements:

- Higher Temperatures (Until 500°C, depending of construction materials)
- Few dust loaded gas flows (approx. 50 g/m^3 max.)
- Only for higher gas velocities (approx. 10 m/s min.)
- The density of the medium must be known

The Prandtl tube itself is a very simple and cheap measuring instrument. Because the measuring is selectively for an integral gas flow measurement several measuring points must be measured. Therefore, the time consumption for the measurement itself and the evaluation is considerable.

Most application in cement industry for Prandtl tubes are:

- Air output of grate coolers
- Riser ducts
- Tertiary air ducts
- etc.

Generally, the application of Prandtl tubes are indicated when other instruments e.g. anemometers as a result of high temperatures or other reasons cannot be used.

If a Prandtl tube is put into a flowing medium, the following pressures are produced:

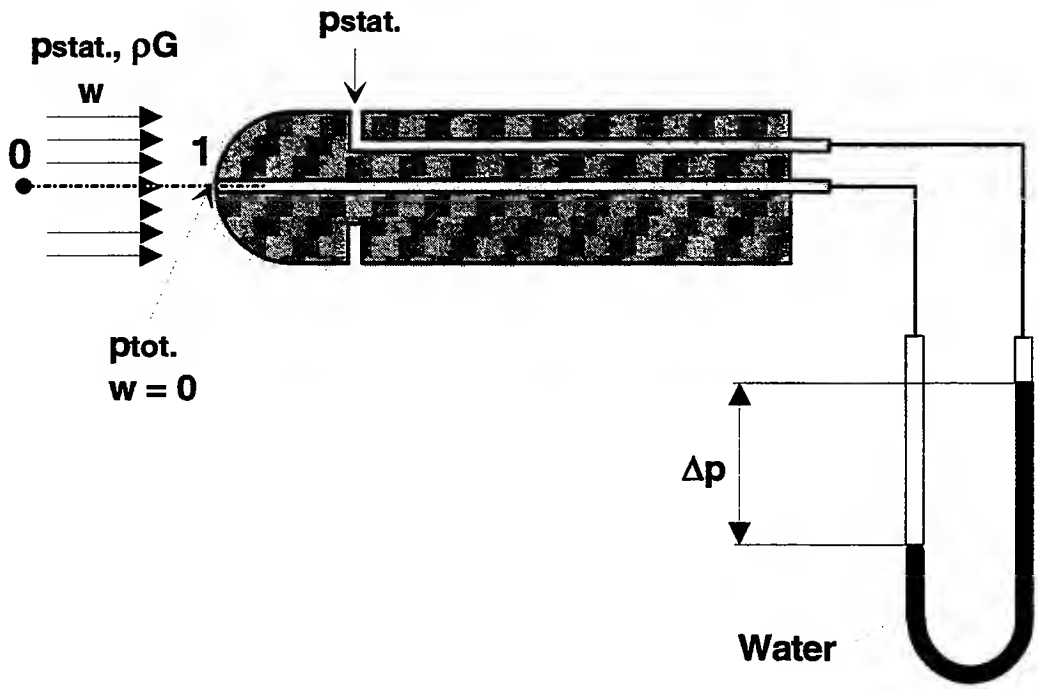

Fig.2

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The total pressure P_{tot} acts on the ball-shaped measuring head, composed of the static pressure P_{stat} and the dynamic pressure. The Bernoulli-equation formulated from the point 0 (ambient point) to the point 1 (pressure point) states:

$$p_{stat} + \frac{\rho_G}{2} w^2 = p_{tot} \quad \begin{matrix} total\,pressure \\ \uparrow\ \uparrow \\ dynamic\,pressure \\ static\,pressure \end{matrix} \qquad (1)$$

The flow speed is only influenced by the dynamic pressure:

$$w = \sqrt{\frac{2}{\rho_G}(p_{tot} - p_{stat})} \qquad (2)$$

Its value is obtained from the difference of the total pressure P_{tot} and the static pressure P_{stat} read on a differential pressure gauge.

$$(p_{tot} - p_{stat}) = \rho_W \cdot g \cdot h \qquad (3)$$

From equation (2) and (3)

$$w = \sqrt{2 \cdot g \cdot \frac{\rho_W}{\rho_G} \cdot h} \qquad (4)$$

PS: Prandtl tubes are insensitive to deviations from the flow direction up to an angle of about ± 10 ° only (See chapter 3, Fig 2)

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6.2.1 Calculation of the Gas Density

Theoretically, the real gas density ρ_G, considering the humidity, should be calculated by determination of the water content or the dew point. The influence of the humidity is only relevant in case of high temperature and high water contents, e.g. exhaust gas of the kiln.

In the case at hand the density of moist air on standard conditions

$$p_0 = 760[\text{Torr}] = 760[\text{mmHg}] = 1.01325 \cdot 10^5 \left[\frac{N}{m^2}\right] \text{or[Pa]}$$

$$T_0 = 273.16[°K] = 0[°C]$$

$$1[\text{mmHg}] \cong 133.322 \left[\frac{N}{m^2}\right] \text{or[Pa]}$$

$$1[\text{mbar}] \cong 10^2 \left[\frac{N}{m^2}\right] \text{or[Pa]}$$

and in function of the relative humidity φ is almost invariant:

Table 1

φ [%]	ρ_o [kg/m^3]
100	1.290
80	1.290
60	1.291

Thus, the actual gas density ρ_G can be found with the following correlation:

$$\rho_G = \rho_0 \frac{T_0 \cdot p_G}{T_G \cdot p_0} = 1.29 \frac{273.16 \cdot p_G}{T_G \cdot 1.01325 \cdot 10^5} \qquad (5)$$

where p_G = actual static gas pressure [N/m^2]
T_G = actual gas temperature [°K]

P_G can be measured by adding the differential pressure ΔP_s between the tube and the ambient to the barometric pressure.

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6.2.2 Numerical Calculation of Density

Barometric pressure	P_a	=	..... [mbar]
		=	..... $\cdot 10^5$ [N/m^2]
Static pressure	P_s	=	..... [mmW])
		=	..... $\cdot 10^5$ [N/m^2]
Gas pressure	P_G	=	$p_a + P_s$ =..... $\cdot 10^5$ [N/m^2]
Gas temperature	t_G	=	.....[°C]
	T_G	=	.....+ 273 [°K]

$$\rho_G = 1.29 \frac{273.16 \cdot p_G}{T_G \cdot 1.01325 \cdot 10^5}$$

$$\rho_G = 1.29 \frac{273.16 \ldots\ldots \cdot 10^5}{\ldots\ldots 1.01325 \cdot 10^5} = \ldots\ldots \left[\frac{kg}{m^3}\right]$$

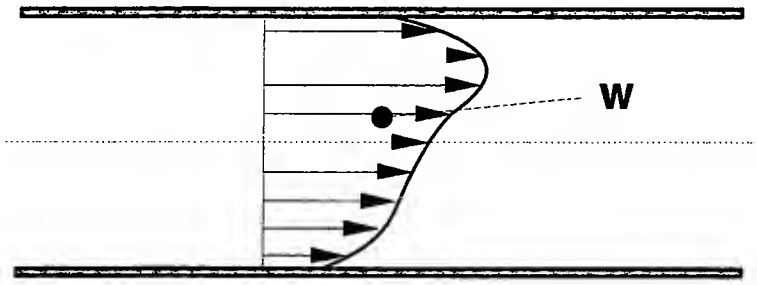
6.3 Calculation of the Gas Velocity

With the knowledge of the gas density ρ_G the gas velocity can be calculated by the following equation (6)

1000

.....

The velocity in the given tube is not constant over the cross section and a certain velocity profile appears:



For this reason the average velocity must be found. This is made by an integration of the local velocities over the cross sectional area.

In order to split the integration in discrete values, the cross sectional area was shared in equal-sized concentric sections, each including four measuring points (two on each measuring axis). The locations where the pressure measurement are to be carried out depend on the desired total number of measuring points to be made on each measuring axis (see Table 2 and Figure 4).

Table 2

	12 measuring points	10 measuring points	8 measuring points	6 measuring points
1	0.022 x D	0.025 x D	0.032 x D	0.043 x D
2	0.067 x D	0.080 x D	0.105 x D	0.146 x D
3	0.118 x D	0.145 x D	0.194 x D	0.296 x D
4	0.177 x D	0.225 x D	0.323 x D	0.704 x D
5	0.250 x D	0.340 x D	0.677 x D	0.853 x D
6	0.356 x D	0.660 x D	0.806 x D	0.957 x D
7	0.644 x D	0.775 x D	0.895 x D	
8	0.750 x D	0.855 x D	0.968 x D	
9	0.823 x D	0.920 x D		
10	0.882 x D	0.975 x D		
11	0.933 x D			
12	0.979 x D			

(D = Diameter)

For this measurement six points are taken

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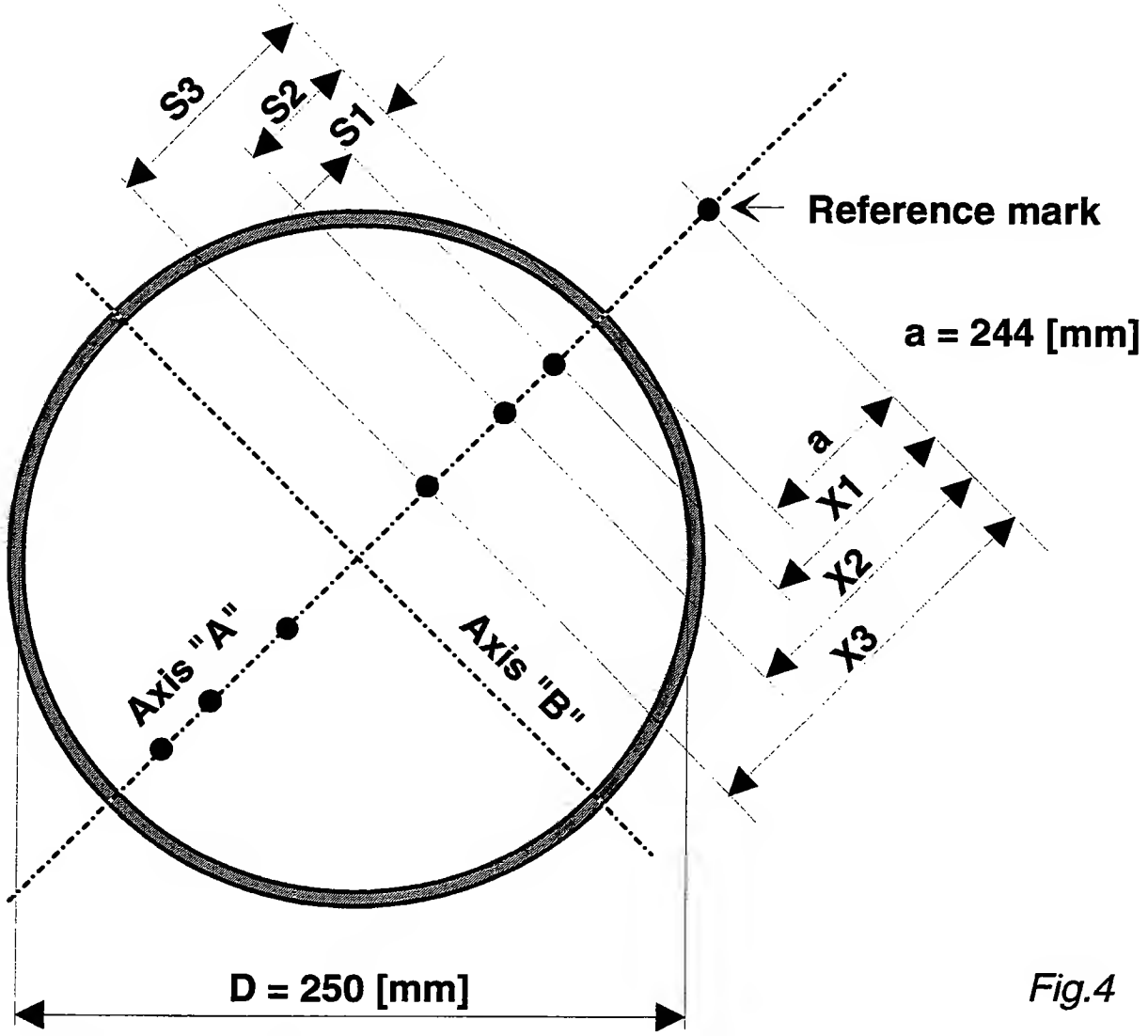

Fig.4

Distance from inside wall

$s_1 =$ 9.043 · D = 11 mm
$s_2 =$ 0.146 · D = 37 mm

Distance from a reference mark

$x_1 = S_1 + a = 235$ mm
$x_2 = S_2 + a = 261$ mm

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Table 3

Axis	Point	Distance from inside wall [cm]	Distance from reference mark [cm]	Δh* [mm W]	Δh [m W]	Velocity (see 6) $w = ... \cdot \sqrt{h}$ [m/s]
A	1	1.1	23.5			
A	2	3.7	26.1			
A	3	7.4	29.8			
A	4	17.6	40.0			
A	5	21.3	43.7			
A	6	23.9	46.3			
B	1	1.1	23.5			
B	2	3.7	26.1			
B	3	7.4	29.8			
B	4	17.6	40.0			
B	5	21.3	43.7			
B	6	23.9	46.3			
					$\bar{w}=\frac{\sum w}{12}=$	

6.4 Calculation of the Gas Flow Rate

a) Operating conditions

$$\dot{V}=A_T \cdot \bar{w}=0.0491 \cdot ..... = ... \left[\frac{m^3}{s}\right]$$

b) Standard conditions

$$\dot{V}_0=V_G \frac{\rho_G}{\rho_0}= ..... \cdot \frac{.....}{1.29} = ... \left[\frac{Nm^3}{s}\right]$$

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6.5 Anemometers

6.5.1 Type A (Vane Wheel Anemometers)

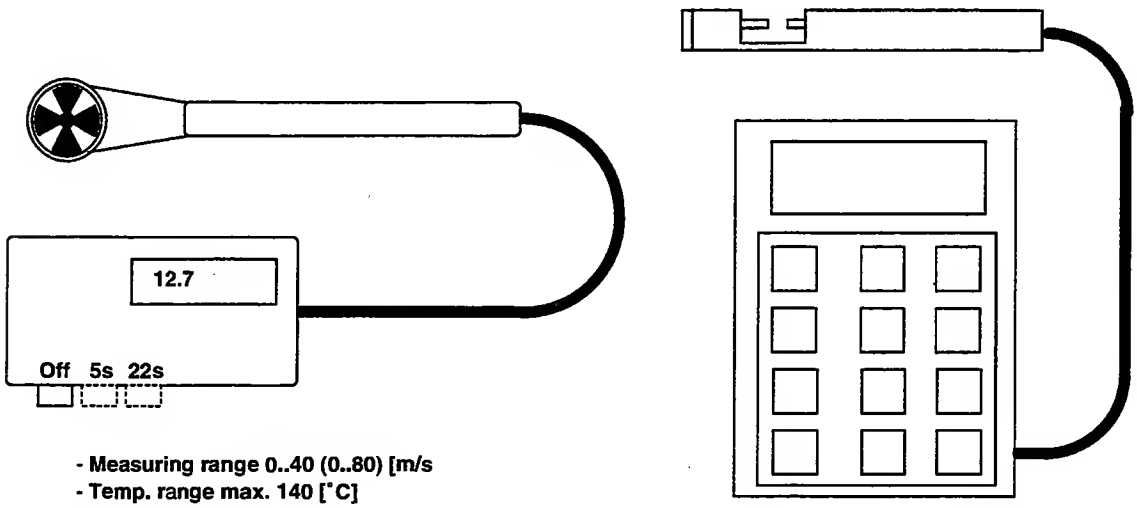

Fig.5 **A: Vane wheel Anemoneter** **B: Thermal gasflow meter**

The revolutions of the vane, being proportional to the gas velocity, have directly been measured by an electronic sensor. The instrument integrates the number of revolutions by a certain time and the indication is in meter per second:

$$w_A = \ldots\ldots\ldots\ldots \left[\frac{m}{s}\right]$$

$$\dot{V} = A_T \cdot w_A = 0.0491 \ldots\ldots\ldots\ldots = \ldots\ldots\ldots\ldots \left[\frac{m^3}{s}\right]$$

Vane wheel anemometers are independent of gas condition.

6.5.2 Type B Thermal Anemometers

Thermal gas flow meters were based on the principle of heat transfer from an electrical heated body to the ambient medium (gas) depending from the relative velocity between body and medium. The probe consists of a heated temperature sensor and two additional temperature sensors. The first sensor measures the temperature of the heated sensor and the second the temperature of the medium. The temperature difference between heated sensor and the medium contains the information for calculating the gas velocity with the aid of an integrated micro processor.

The indication of thermal gasflow anemometers is depending from gas conditions:

- specific heat
- conductivity
- density

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The calibration is normally based on the medium air. For other gases (kiln gases) correction factors should be used. Tables with such factors are available from the suppliers.

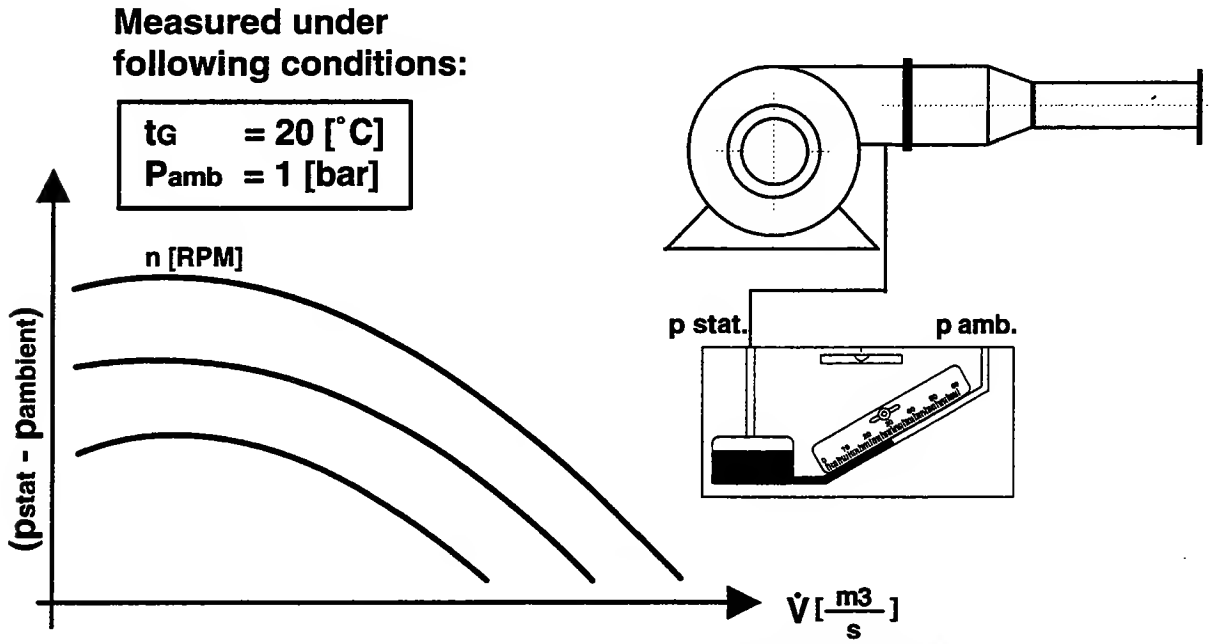
6.5.3 Vortex Anemometers (not used in practical work)

When a object is inserted in a flowing fluid then a vortex tail, known as Karman vortex street, arises. Karman found (about 1912) out, that depending on the velocity a constant relation exists between vortex interval and vortex sequence behind a cylindrical body in a current. He also observed that at constant flow velocity the vortex frequency downstream remains practically constant.

Vortex anemometers are independent of gas conditions like density, pressure and temperature until 180°C. Vortex measuring probes have no moving parts. They are robust, reliable and corrosion-resistant. Even in rough conditions as dust loaded fluids a high fatigue strength is ensured.

6.6 Characteristic Curves of the Fan

The supplier establishes the fan characteristics by measuring the difference between the static pressure at the fan outlet and the total pressure at the fan inlet in function of the revolutions of the fan and the volume rate. This measurement is given by the following diagram.



PS: For most fans in cement plants, it is satisfactory to assume that the fan's static pressure is the difference between the static pressure at the inlet (here atmospheric pressure) and outlet of the fans.

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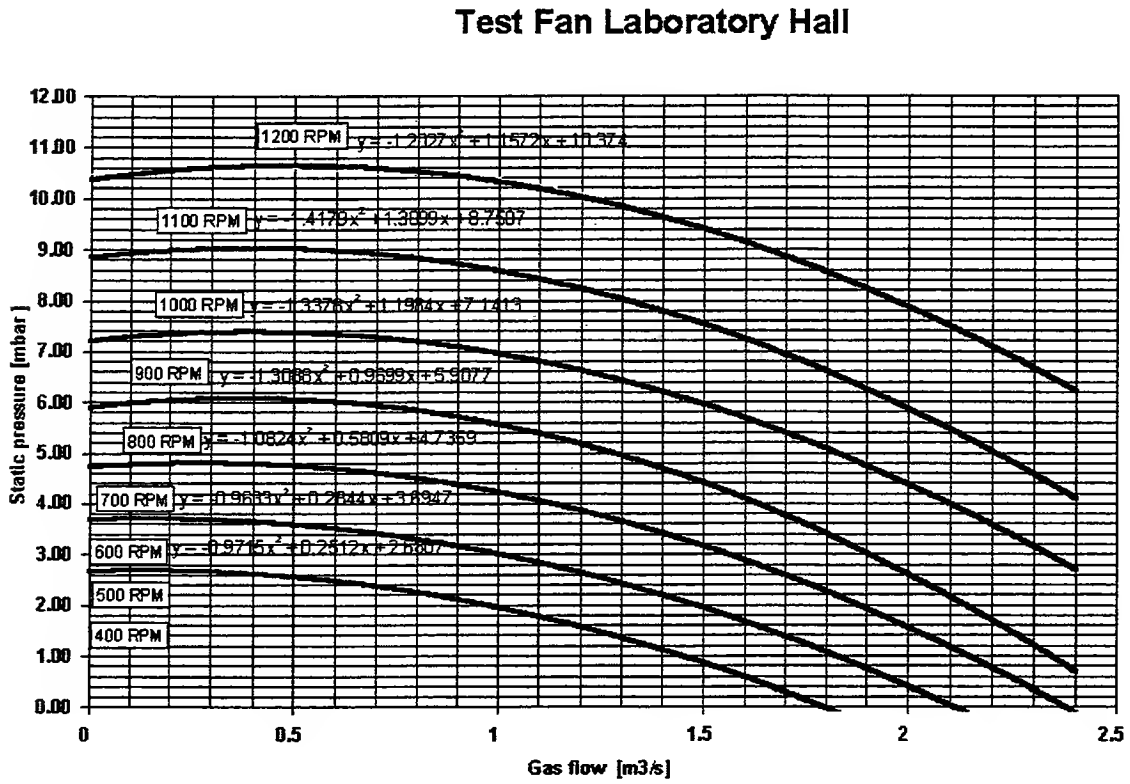
6.6.1 Measurements

n = ... [rpm]

Δp = ... [mbar]

From the given diagram by the supplier: $\underline{V_f = \ldots\ldots\ldots [m^3/s]}$



6.7 INLET NOZZLE

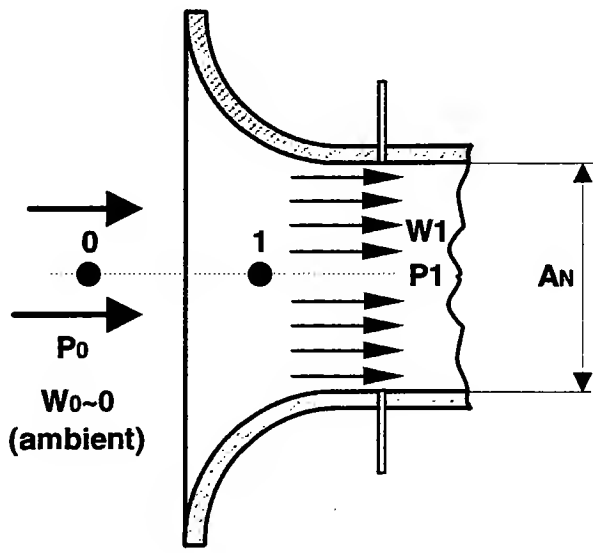


Fig. 8

Equation of Bernoulli:

$$w_0 = \frac{p_0}{\rho_G} + \frac{w_0^2}{2} = \frac{p_1}{\rho_G} + \frac{w_1^2}{2} \qquad (7) \quad (8)$$

$$w_1 = \sqrt{\frac{2 \cdot (p_0 - p_1)}{\rho_G}}$$

Continuity condition:

$$V_f = \cdot A_N \cdot w_1 = \cdot A_N \cdot \sqrt{\frac{2 \cdot (p_0 - p_1)}{G}} \qquad (9)$$

U - Tube manometer:

$$(p_0 - p_1) = \rho_W \cdot g \cdot \Delta h \qquad (10)$$

From equations (9) and (10) :

$$V_f = \alpha \cdot A_N \cdot \sqrt{2 \cdot g \cdot \frac{\rho_W}{\rho_G} \Delta h} \qquad (11)$$

α	=	0.93	[-]	(measured by the supplier)
A_N	=	0.1244	[m^2]	
ρ_W	=	1000	[kg/m^3]	
ρ_G	=	.....	[kg/m^3]	(see 2.2)
Δh	=	..... [mm W] · 0.001	[m WC]	
V_f	=	$0.93 \cdot 0.1244 \cdot \sqrt{2 \cdot 9.81 \frac{1000}{.....} .....}$		
V_f	=	.....	[m^3/s]	

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6.8 Manometers

6.8.1 Inclined Stationary Manometer

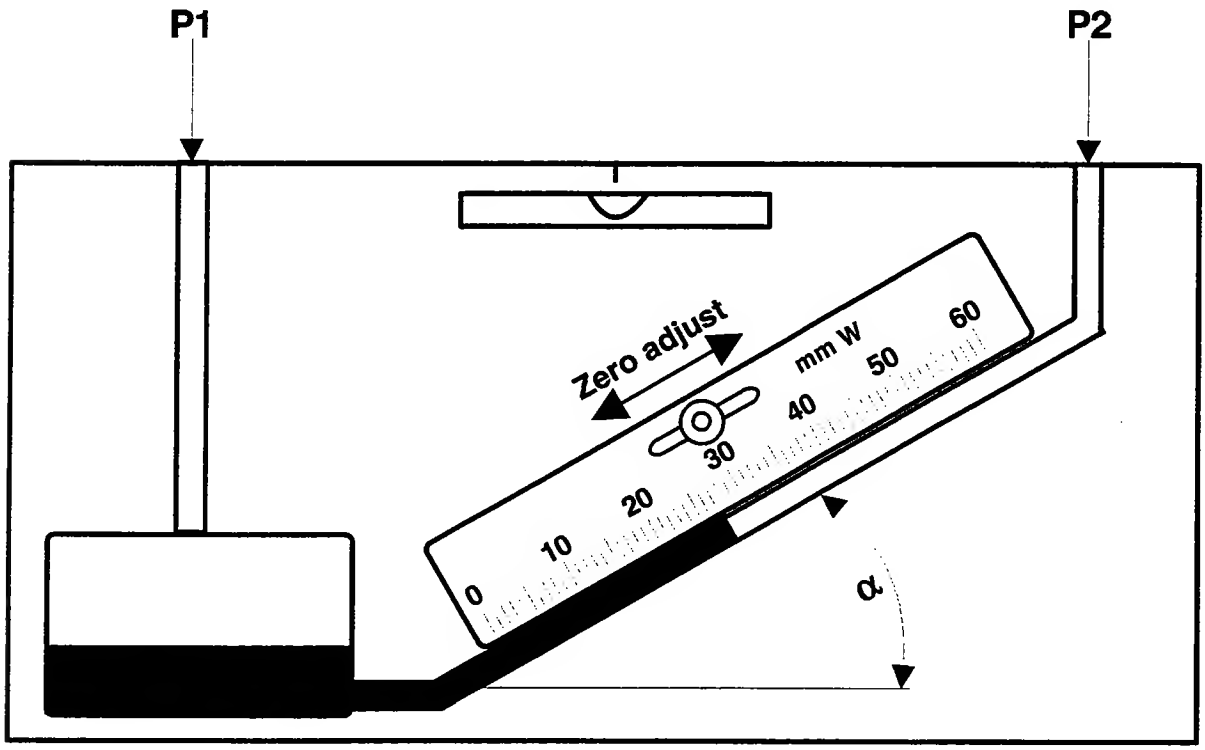

Fig. 9

PS: On the instrument itself mostly not the angle α is indicated but directly the value of sin α, e.g.

1 : 5 = 0.2 = 11.5°
1 : 2 = 0.5 = 30.0°

The angle of the used manometer is fixed and the scale is calibrated in mm W

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7. SYMBOL LIST OF GAS FLOW MEASUREMENT

A	[m^2]	Cross section area
d	[m]	Diameter of the smallest aperture
D	[m]	Diameter of the tube
h	[m]	Height of Liquidcolumn
m	[-]	Cross section area ratio $m = d^2 / D^2$
$\dot{m}$	[kg / s]	Mass flow rate
p	[N / m^2]	Pressure
$\dot{v}$	[m^3 / s]	Volume flow rate
w	[m / s]	Velocity
α	[-]	Correction factor for the contraction
φ	[%]	Humidity
ε	[-]	Correction factor if fluid is compressible
ρ	[kg / m^3]	Density
fdr	[m^3/m^3 dry]	Water content (Volume)
W_0	[kg/m^3 n,dry]	Water content (Mass)

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Binder 7

Engineering

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Chapter 1

Power Supply

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Power Supply

Fritz Richner

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1. INTRODUCTION

A cement plant is considered a big consumer of electrical energy. The energy consumption is comparable to the one of a small town, e.g. 15'000 habitants of a town require approx. 20 MW of electrical power - the equivalent of a cement plant with a production capacity of 2000 tons/day.

The cement production process is a continuous process and depends on a very reliable power supply system and distribution network. Any interruption in the power supply means a loss in production and causes a lot of trouble.

The reliability of a power supply depends on the:

- design of the supplying network
- atmospheric conditions
- strength of the supply

In the following a short description of the commonly used generation and systems is given.

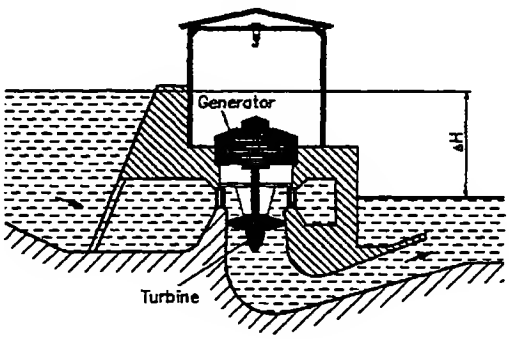
2. POWER GENERATION

The geographical location, the climate and the level of technical and economical development of a country are decisive factors for the selection on the type of power generation systems. A mix of several systems may be of advantage in order to cope with the fluctuating power requirements of industries and public consumers during the charging phases of the day or the year.

The following are the most commonly used power generating systems and their main characteristics.

2.1 Hydraulic Power Station

Figure 2.1.1 River power plant



- Low water level difference (<25m)
- Low water pressure
- Large water quantity
- normally continuously operated
- for base load power supply

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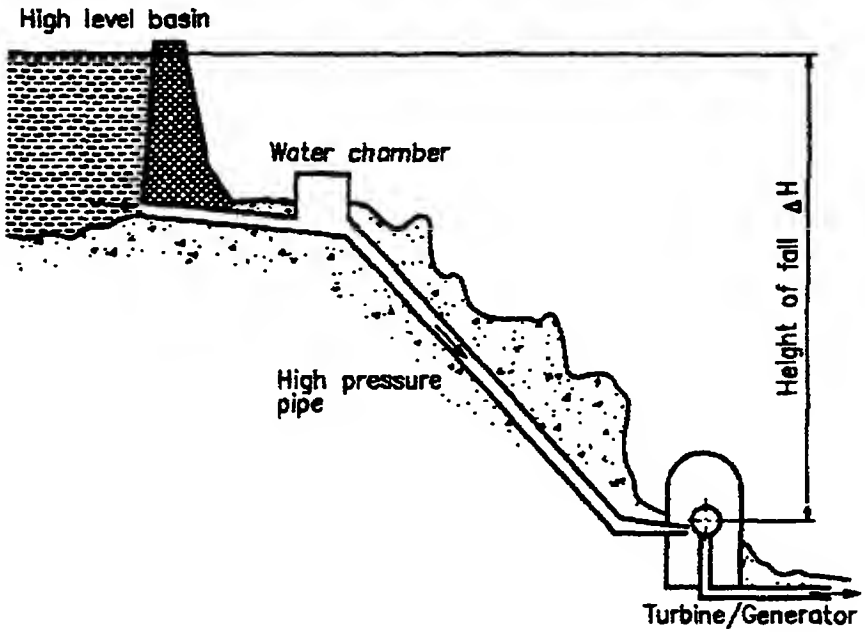
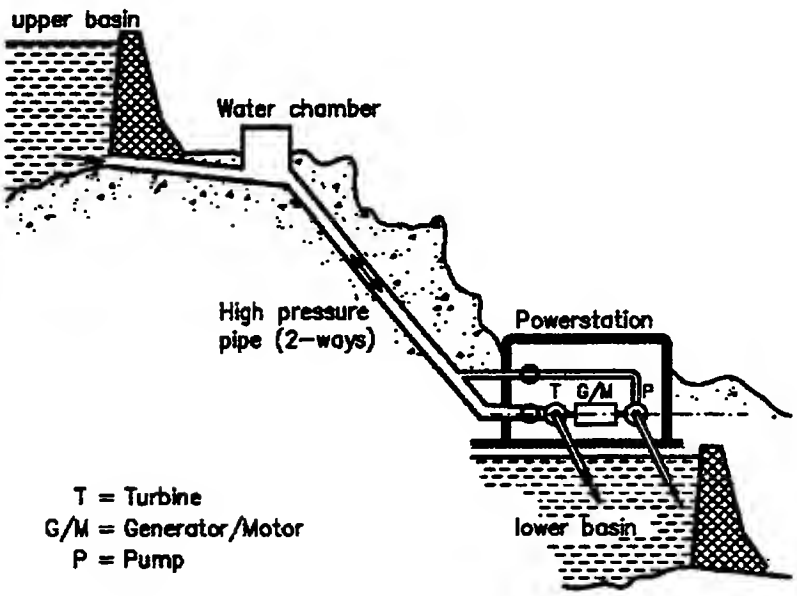
Figure 2.1.2 High level basin power plant



- Large water level difference
- High water pressure
- Low water quantity
- Simple ON/OFF operation
- for peak power demand

Figure 2.1.3 Pump storage power plant



- Large water level difference
- High water pressure
- Minimum water quantity
- Simple ON/OFF operation
- for peak power demand
- re-use of water

Function:

High power demand phase: (day-time)	Lower power demand phase: (night time)
◆ High water pressure drives turbine-generator set ◆ Water collected in lower basin	◆ external power drives motor to pump water from lower to upper basin.

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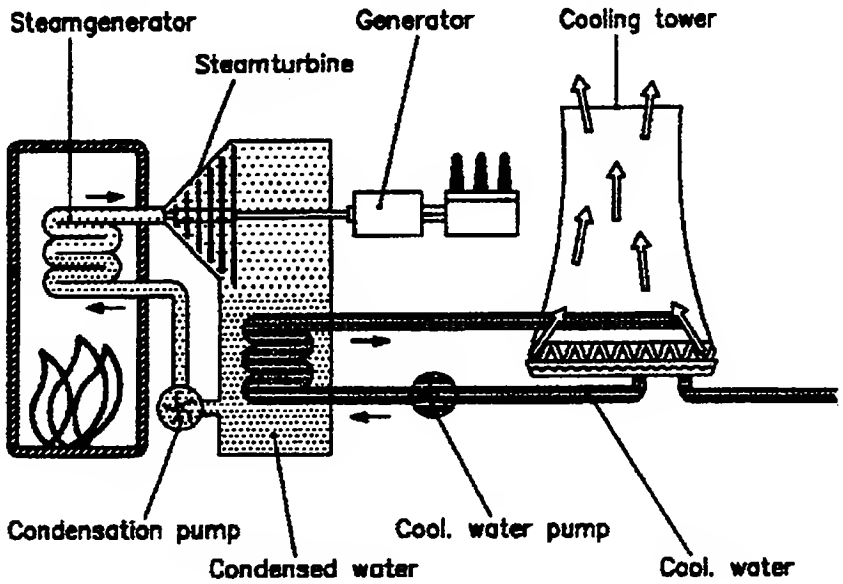
2.2 Thermal Power Plants

Figure 2.2.1 Fuel boiler systems



- Oil, gas or coal fired boiler
- High pressure steam drives steam turbine/generator (steam 200 bar/500°C)
- Efficiency <40%
- For base load demand
- Capacities: up to 800 MW p.u.

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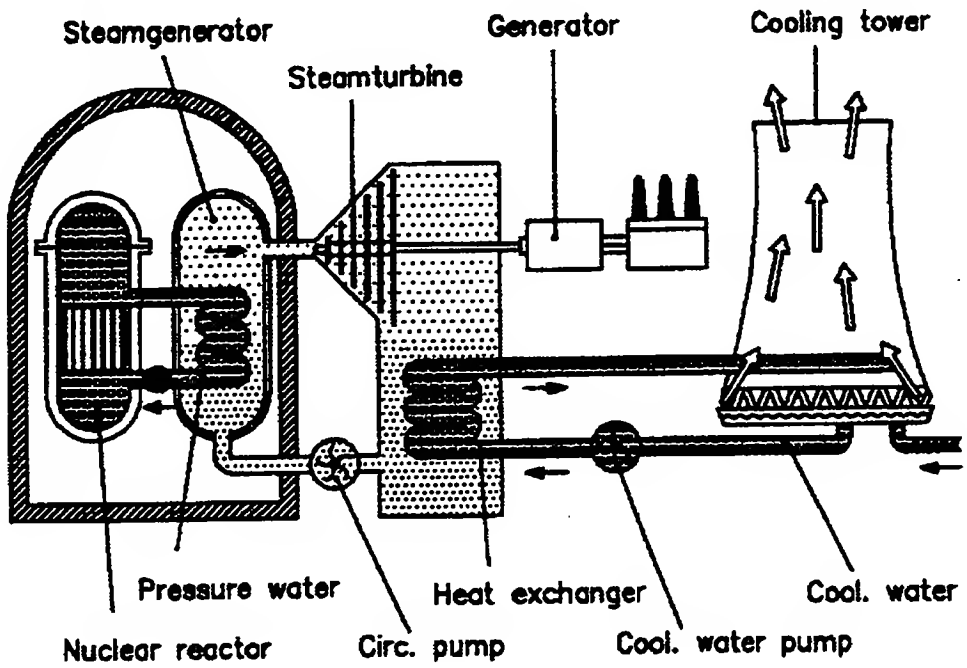
Figure 2.2.2 Nuclear power systems



- Boiling water reactor type
  - prim. circuit: 70 bar, 280°C
- Pressure water reactor type
  - prim. circuit: ~160 bar, 300°C
  - sec. circuit: ~60 bar, 270°C
- For base load demand
- Efficiency: < 35%
- Capacities: up to 1300 MW p.u.

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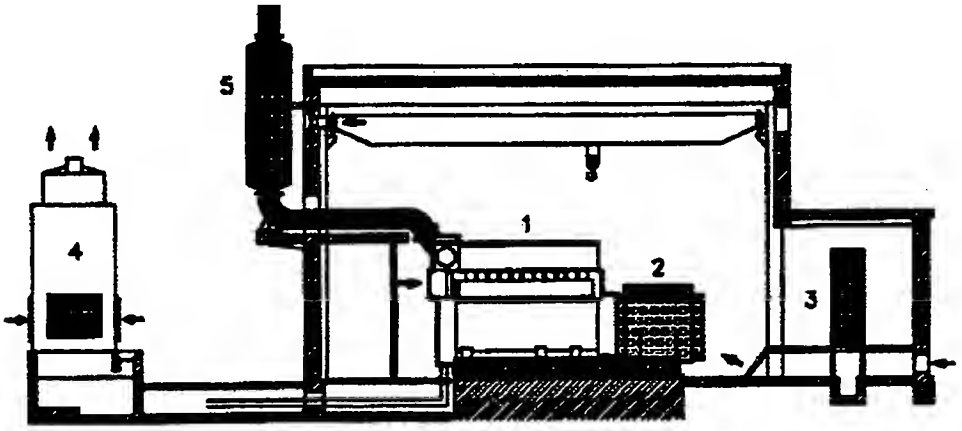
2.3 Industrial Type Power Plants

Figure 2.3.1 Diesel power units



- Individual unit sizes for 0.5 - 40 MW
- Indoor and outdoor units
- Fuel: heavy fuel oil
  light fuel oil
- Efficiency: < 40%
- With heat recovery system to heat up e.g. heavy fuel oil

1 = Diesel engine

2 = Generator

3 = Control system

4 = Cooling tower

5 = Exhaust silencer

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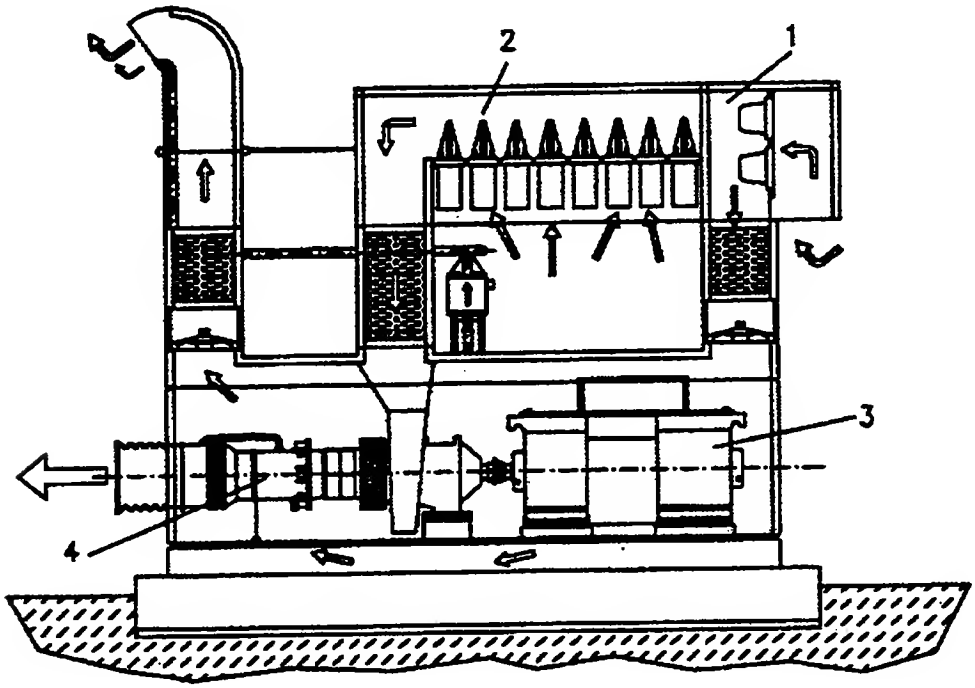
Figure 2.3.2 Gas turbine units



- Individual unit sizes of package units: 5 to 30 MW
- Combined cycle power plants up to 200 MW
- Complete packages for indoor and outdoor
- Fuel:
  - Natural gas
  - Diesel Oil
  - Heavy fuel oil
- Efficiency: < 30% (~ 50% for combined cycle P.P.)

1 = Air intake

2 = Air filter

3 = Generator

4 = Gas turbine

5 = Exhaust silencer

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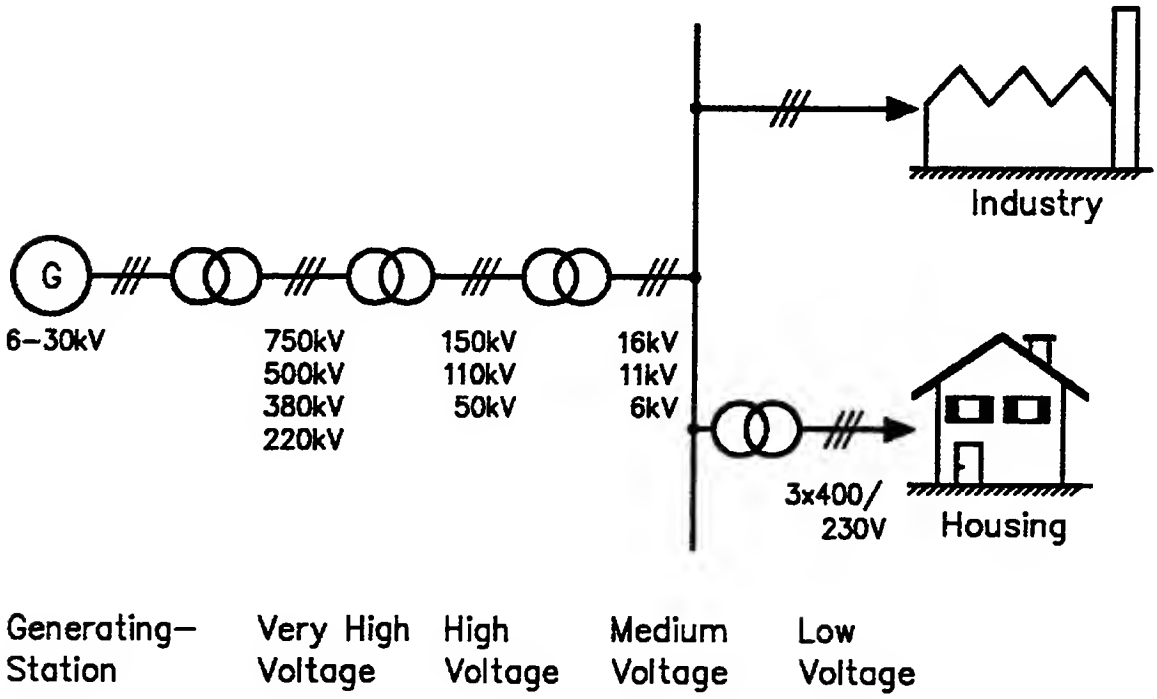
3. GENERAL DISTRIBUTION ARRANGEMENT

Electrical power generated in power plants need to be transported to the consumers. In order to avoid transmission losses, the AC-voltage is transformed at the generation station to extra high voltage levels (up to 750 KV~). At the consumer side, transformation to lower levels is required in order to feed the common industrial and household networks.

Figure 3.1 Typical transmission scheme



4. POWER TRANSMISSION

An extra high voltage (or maximum voltage) level network can for economical as well as for technical reasons only be operated by means of an overhead system. For short distances cables can be used for voltage up to approx. 400 kV.

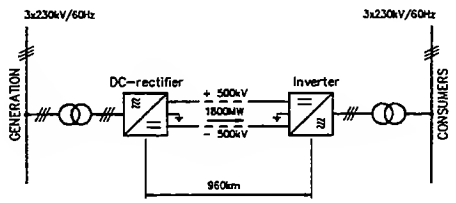
High voltage and medium voltage networks use both cables and overhead systems. In built-up areas cable connections are preferred whereas on open areas overhead lines are used mostly due to economical reasons.

4.1 Voltage levels:

- ♦ > 150 kV = extra high (or maximum) voltage
- ♦ > 50 kV = high voltage
- ♦ 1-50 kV = medium voltage
- ♦ ≤ 1 kV = low voltage
- ♦ < 50 V = extra low voltage

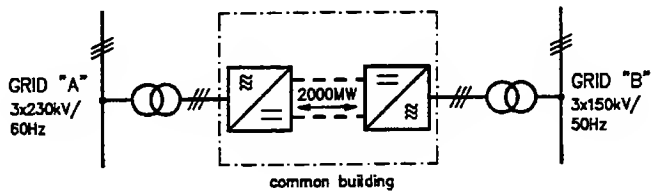
Loading of transmission lines in industrialised networks have in many cases reached their limits. Because of the high cost for construction of new transmission lines and delays due to environmental considerations, power supply companies planning to increase power transfer or to construct new extremely long transmission networks must today resort to new means. This includes so-called 'High Voltage Direct Current' links (HVDC-links). The 3-phase AC-supply is converted into a DC-voltage by means of static converter systems and at the other end of the transmission line re-converted by means of thyristor current converter into the 3-phase supply. HVDC-links reach voltage levels of up to ± 500 kV DC and transmit e.g. 3000 MW.

Figure 4.1.1 Concept of HVDC-link Nelson-River, Canada



HVDC-links are furthermore used for the coupling of two different networks systems, i.e. systems with different voltage levels and different frequencies (see Fig. 4.1.2). They are called 'short coupling'. In addition, these systems allow the interconnection of a stable with an unstable or weak network. A typical application is the connection of networks between the former eastern countries with western Europe.

Figure 4.1.2 Coupling of two different AC-systems by means of 'short HVDC-link'



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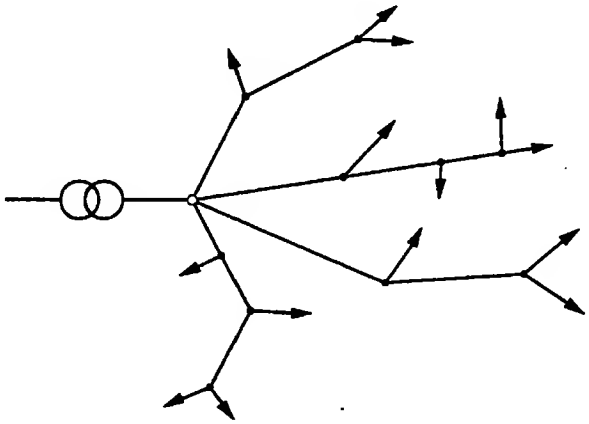
5. NETWORK SYSTEMS

5.1 Branch Power Supply

Branch networks are fed from one feeding point, a. generating plant or a main distribution station

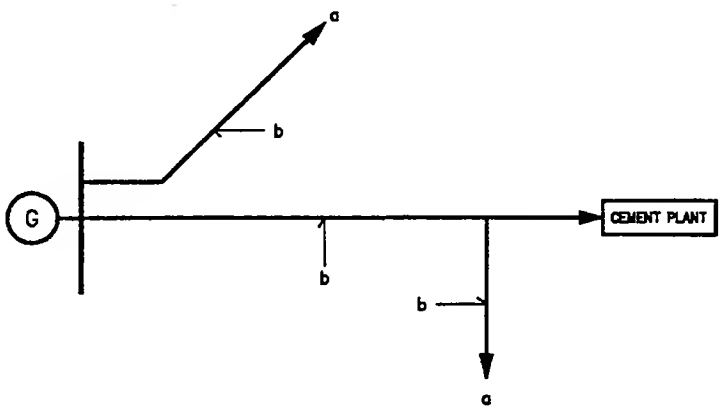
Figure 5.1.1 Branch power supply network



As can be seen from Fig. 5.1.2 below the power supply of the cement plant depends on one long branch line which has to be shared with another consumer. Any fault on the line caused by lightning, a falling tree, a. big bird, etc. will cause a black-out of the cement plant. Even a fault in the other consumers' network may trip the main circuit breaker and also disconnect the cement plant.

This unfortunate situation can be improved by trying to get a second supply line, preferably from a different source.

Figure 5.1.2 Branch power supply



G: power generating plant

a : power consumer

b: overhead power line

A branch power supply is not considered as a reliable source and ought to be backed up by installation of a generating station. The size of this (these) generator(s) will depend on the reliability of the power supply. A small emergency power generator (approx. 500 kW) has to be installed in any case to bring the plant under all circumstances to a safe standstill.

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5.2 Ring Power Supply

The power distribution network (Fig. 5.2.1) is called a ring. It guarantees a very high reliability and uninterrupted power to all connected consumers also in case of a failure in one of the supplying overhead lines.

Under normal operation ring networks are normally 'open' in the middle of the ring, i.e. the two sections of the ring are operated as a branch power supply. In case of a disturbance, the ring is 'closed' and the supply to all consumers is available.

Figure 5.2.1 Ring power supply
a) normal operation ('open')
b) disturbance, disconnector 'closed')

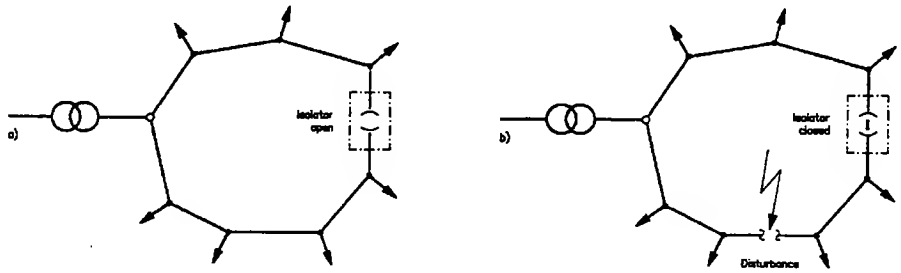
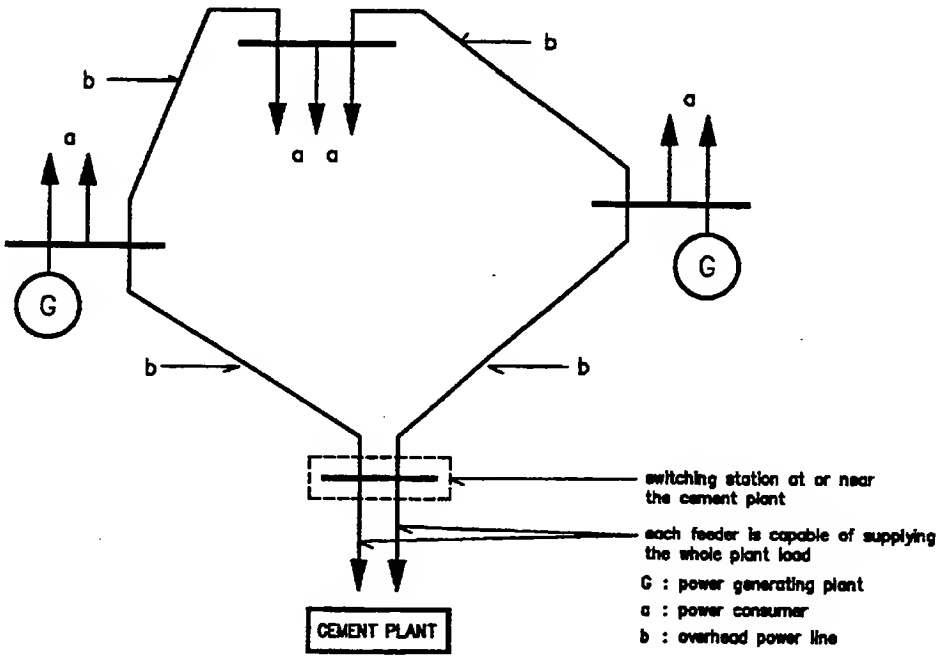


Figure 5.2.2 Preferred ring power supply system for cement plants



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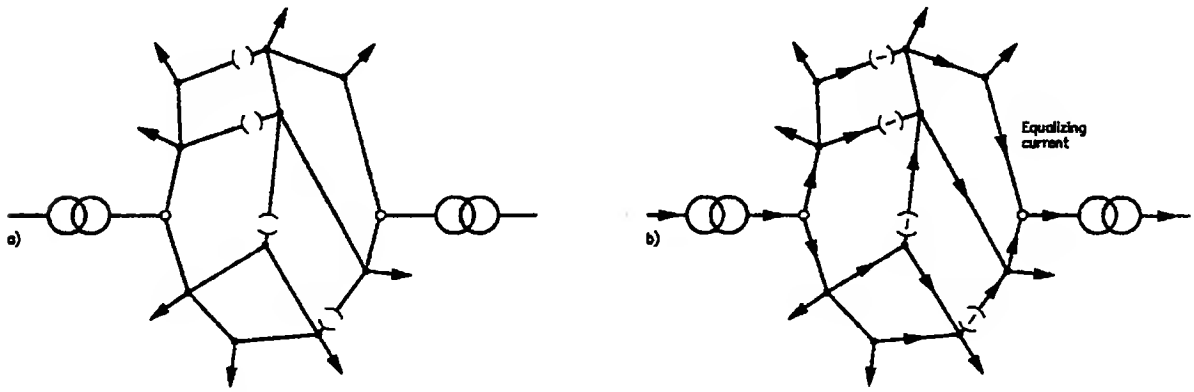
5.3 Mesh Networks

Interconnections within existing networks result in a so-called mesh network, which are normally fed from different sources i.e. generating stations. Availability of power as well as level of supply voltage are normally excellent, losses within the network are low.

Figure 5.3 Mesh network
a) 'open' operation (for LV-networks)
b) equalising current in network with different voltage levels



This type of networks requires high number of switching devices and protection equipment. Short circuit capacity is normally relatively high resulting in costly equipment. Mesh networks are applied for high voltage distributions as well as for low voltage distribution systems.

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6. ENERGY CONSUMPTION / TARIFF STRUCTURE

6.1 General

Electrical energy can practically not be stored, this in contrary to other forms of energy carriers such as gas, oil, coal, wood, water etc. Therefore, production of electrical energy and consumption is always balanced and the supply companies undertake every effort to cope with the continuously charging demand. A certain surplus of generating capacity must be available to avoid energy supply bottle-necks.

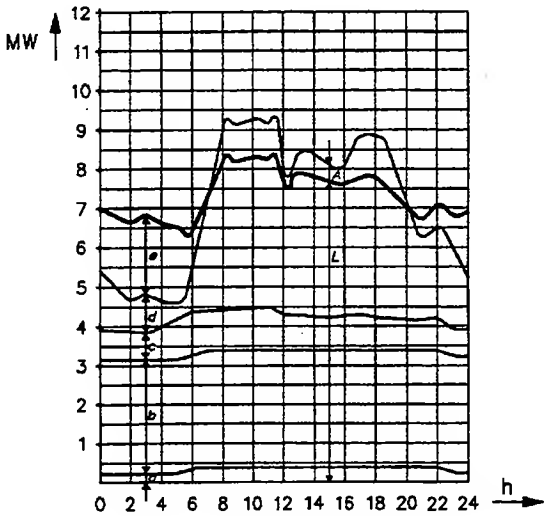
The application of different types of generating systems, the interconnection of large supply networks (Europe Interconnected grid from North of Norway to South of Italy) guarantee for a stable and continuous supply.

Small grid systems have the disadvantage to be weak, with unstable frequencies, heavy loads may lead to partial disconnection which may end in total black-outs for certain regions (Example: New York).

6.2 Typical Load Diagram

The graph Fig. 6.2 shows a typical demand over a period of 24 h and indicates the portions supplied by the various types of power stations (Example: Switzerland).

Figure 6.2 Load Diagram



Shares:

a =	convent. thermal power station	⎫
b =	Nuclear power stations	⎬ serving the base load
c =	River power plants	⎭
d =	High level basin power plants = peak load	
e =	imported energy	
L =	total power required	
A =	surplus produced, available for export	

The graph illustrates import phases during night-times (cheaper than own generation, partially used to 'refill' the high level basins) and export of peak-energy during day-time (expensive energy).

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6.3 Tariff Structures

It is of utmost importance for the utility companies to be able to calculate the predictable consumption requirements. This in order to provide a stable supply and to eliminate sudden high demands which may affect and weaken the supply systems. For this reason supply contracts contain increasingly more certain power consumption limitation, so called load shedding requirements. These limitations are also reflected in different tariff levels as well as in penalties. For obvious reasons electrical energy consumed during night-time is considerably cheaper and supply companies encourage consumers to use 'night-energy'.

Energy contracts need to be negotiated with supply companies, a careful study considering the needs of the consumer and its equipment is the basis for an economical contract.

Tariffs normally distinguish between base charges calculated on the basic power demand (kW) and energy costs based on the actual consumption (kWh). Energy costs are normally connected to the time of consumption, in certain areas costs vary from hour to hour.

Extra charges (penalties) may have to be paid if agreed on limits on power demand are exceeded.

6.3.1 Typical Tariff example

The following shows a typical tariff structure showing cost-split for energy, power and reactive energy:

Active energy consumption charge

	Winter	Summer
High tariff	0-25 Mio. kWh: 6,3 cts/kWh > 25 Mio. kWh: 6,1 cts/kWh	0-20 Mio. kWh: 4,5 cts/kWh >20 Mio. kWh: 4,3 cts/kWh
Low tariff	0-20 Mio. kWh: 4,6 cts/kWh > 20 Mio. kWh: 4,4 cts/kWh	0-10 Mio. kWh: 2,6 cts/kWh > 10 Mio. kWh: 2,4 cts/kWh

Power demand charge

$ 29.--/kW, calculated on maximum demand of a quarter of a year.

The maximum demand of a quarter is the mean value of the four highest week-maxima, of a full-time period of 60 minutes.

Reactive energy consumption charge

The total consumed reactive energy during high tariff phases is limited to 45,5% of the active energy consumed (corresponds to average power factor (cos phi = 0,91).

Additionally consumed reactive energy will be billed to 2.1 cts/kVarh.

Tariff phases

High tariff:	Monday-Friday	07.00 - 21.00 h
	Saturday	07.00 - 13.00h
Low tariff:	remaining time	
Summer:	April - September	
Winter:	October - March	

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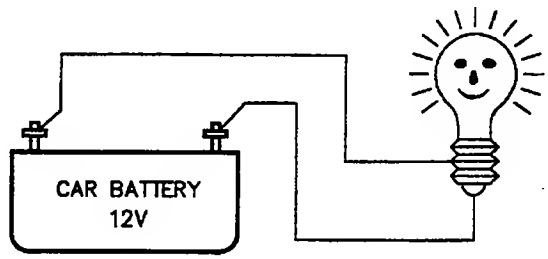
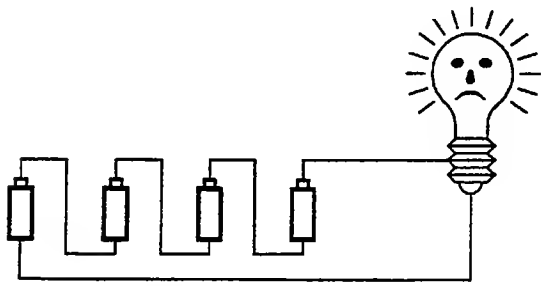
7. CAPACITIES OF POWER SUPPLIES

7.1 Weak and Strong Power Supplies

Figure 7.1 Weak and Strong Power Supplies



in the view of the little 12V lamp, the car battery is a strong power supply



the 4 little 3V batteries, even though they add up to 12V, are still a weak power supply

The supply voltage does not say anything about the strength of the supply. It is the available capacity of the network that determines the strength, or as it is called, the stiffness of a power supply.

The relation between the capacity of the source and the installed power at the consumer's plant is a measure for the stiffness of the electric power system.

The short circuit capacity - a theoretical figure expressed in MVA (Mega-Volt-Ampère) - is based on the assumption of a 3-phase short circuit between the source and the consumer. It is the product of the short circuit current and the line-to-line voltage.

$$S_K = I_{K3p} \times U_N \times \sqrt{3} \text{ (MVA)}$$

S_K : short circuit capacity of a 3-phase system

I_{K3p} : short circuit current of a 3-phase system

U_N : normal voltage

$\sqrt{3}$: factor for a 3-phase system

A strong power supply of approx. 300 MVA is normally required to operate a cement plant.

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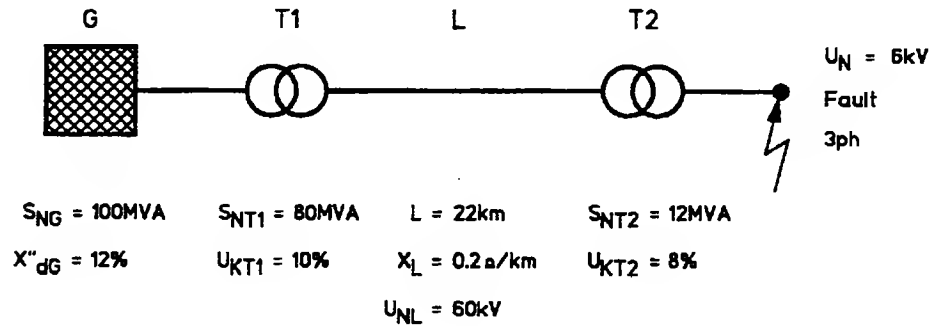
7.2 Calculation of Short Circuit Capacity

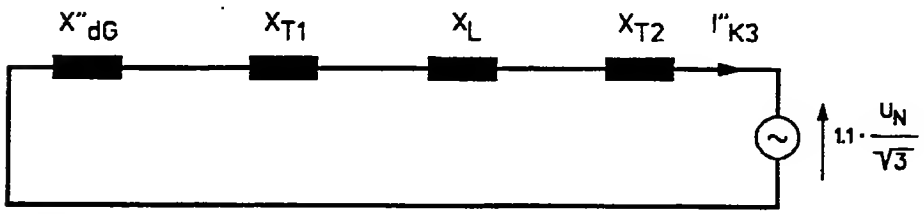
Calculation of symmetrical short circuit capacity S_{K3}' (MVA) resp. short circuit currents I_{K3}" (kA).

Network components:

S_N	=	rated apparent power	(MV)
U_N	=	rated system voltage	(kV)
E"	=	initial voltage of power supply	(kV)
c	=	factor E" / $U_N \approx$ 1.1	
$X"_d$	=	subtransient reactance	(%)
U_K	=	impedance voltage drop	(%)
X_L	=	impedance per conductor	(Ω/km)
$I"_K$	=	initial symmetrical short circuit current	(kA)
$S"_K$	=	initial symmetrical short circuit capacity	(MVA)

Figure 7.2 Network components





Calculation method: The so called '% / MVA system' is particularly useful for calculating short circuit currents in high voltage networks.

There is no need to reduce the impedance of different voltage levels to the voltage of the faulty part of the network.

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This method is especially used for rough estimations in the project phase.

$$X''_G = X''_{dG} \times \frac{100}{S_{NG}} = 0.12 \times \frac{100}{100} = 0.12\%/\text{MVA}$$

$$X_{T1} = U_{KT1} \times \frac{100}{S_{NT1}} = 0.10 \times \frac{100}{80} = 0.13\%/\text{MVA}$$

$$X_L = X_L \times L \times \frac{100}{U^2_{NL}} = 0.20 \times 22 \times \frac{100}{60^2} = 0.12\%/\text{MVA}$$

$$X_{T2} = U_{KT2} \times \frac{100}{S_{NT2}} = 0.08 \times \frac{100}{12} = 0.76\%/\text{MVA}$$

$$\sum X = \mathbf{1.04\%/MVA}$$

$$S''_{K3} = C \times \frac{100}{S_{NT2}} = 1.1 \times \frac{100}{1.04} = \mathbf{106\ MVA}\ (\text{at } 6\text{kV})$$

$$I''_{K3} = \frac{S''_{K3}}{\sqrt{3} \times U_N} = \frac{106}{\sqrt{3} \times 6} = \mathbf{10.2\ kA}\ (\text{at } 6\text{kV})$$

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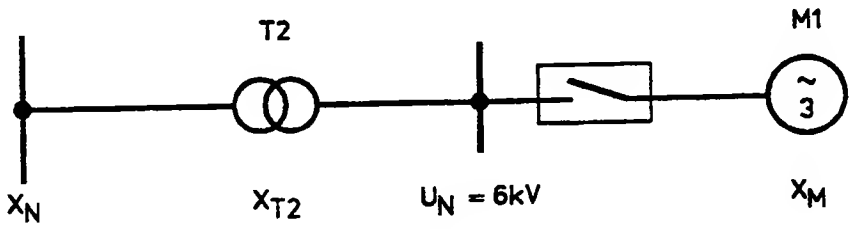
7.3 Calculation of Voltage Drop

Calculation of network voltage drop due to starting of a large motor.

Figure 7.3 Calculation of network voltage



a) Squirrel cage motor (direct starting) $I_S = 5 \times I_N$

$$X_N = C \times \frac{100}{S''_{K3}} = 1.1 \times \frac{100}{300} = 0.37\%/\text{MVA}$$

$$X_{T2} = U_{KT2} \times \frac{100}{S_{NT2}} = 0.08 \times \frac{100}{12} = 0.67\%/\text{MVA}$$

$$X_{M1} = \frac{100}{S_{NM1} \times I_S} = \frac{100}{3.5 \times 5} = 5.70\%/\text{MVA}$$

$$\sum X = = 6.74\%/\text{MVA}$$

$$U_{N6kV} = \frac{(X_N + X_{T2}) \times 100}{\sum X} = \frac{1.04 \times 100}{6.74} = \mathbf{15.4\%}$$

b) Slipring motor (rotor starter) $I_S = 1.2 \times I_N$

$$X_N = 0.37\%/\text{MVA}$$

$$X_{T2} = 0.67\%/\text{MVA}$$

$$XM1 = \frac{100}{3.5 \times 1.2} \quad 3.5 \times 5 = 23.80/\text{MVA}$$

$$\sum X = = 24.84\%/\text{MVA}$$

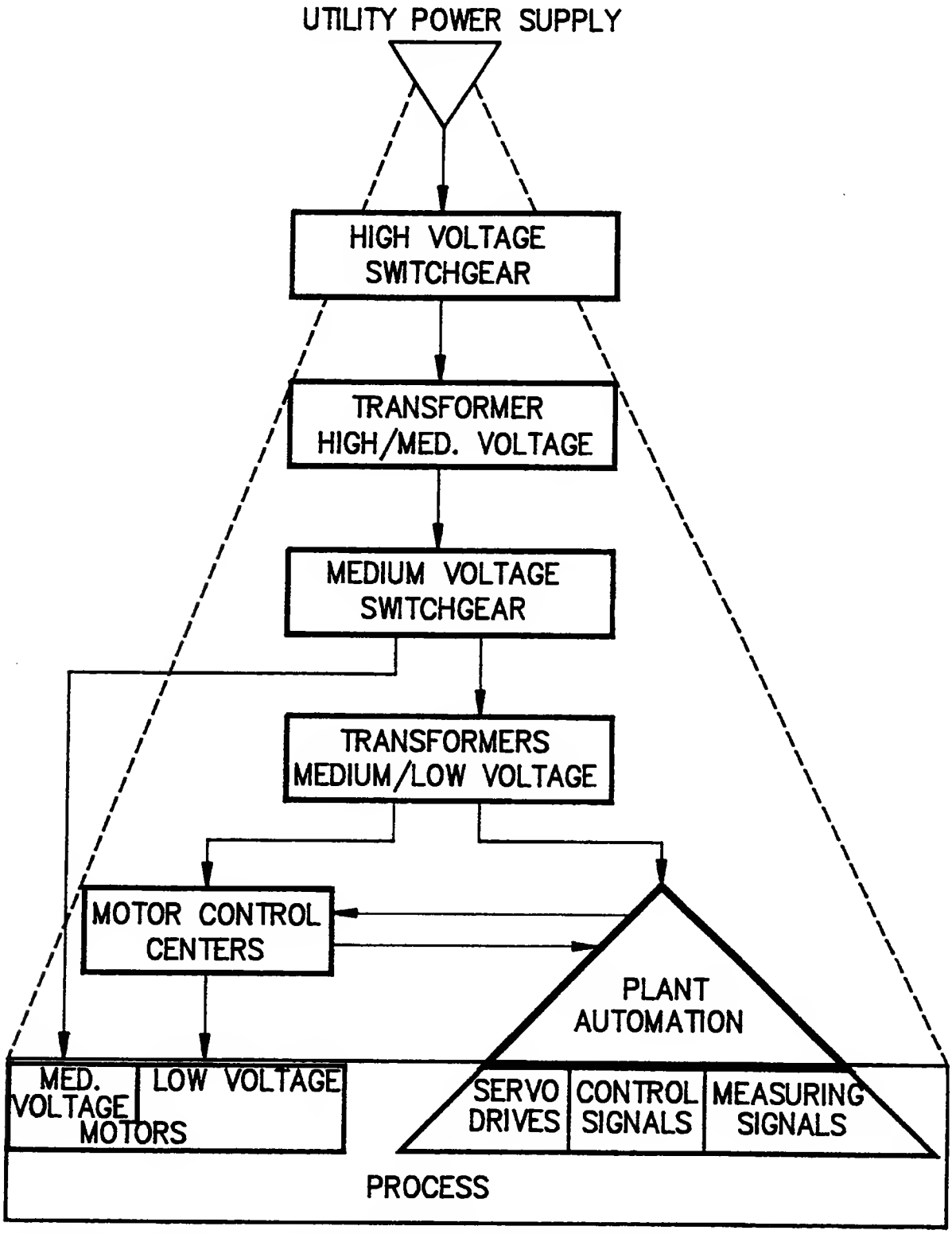
$$U_{N6kV} = \frac{(XN + X_{T2}) \times 100}{\sum X} = \frac{1.04 \times 100}{24.84} = \mathbf{4.2\%}$$

The normal output of a motor is guarantied within a voltage range of $U_N \pm 5\%$

In most cases a voltage drop at the motor terminal during the starting phase of 10% is acceptable and will not cause disturbances to other equipment.

For more important voltage drops closer investigations are required.

8. STRUCTURE OF ELECTRICAL SYSTEMS IN A CEMENT PLANT



POWER SUPPLY AND POWER DISTRIBUTION
ELECTRIC MOTORS
AUTOMATION SYSTEMS

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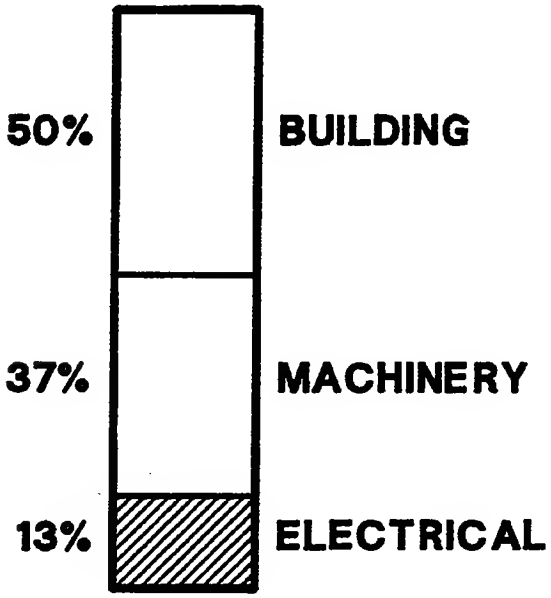
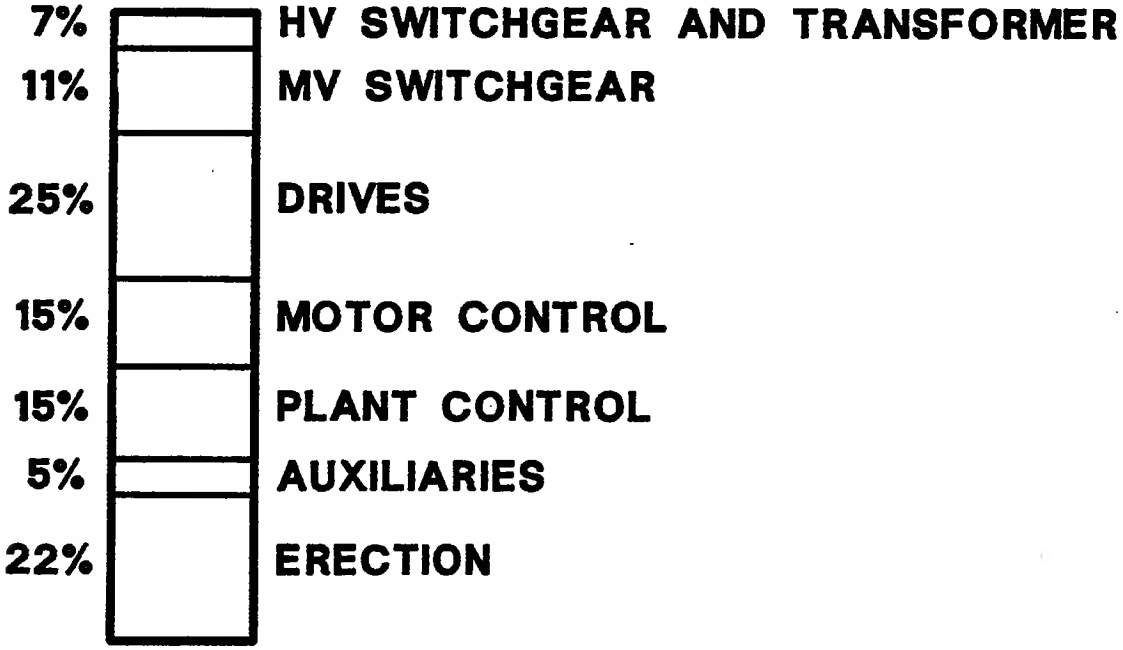
9. INVESTMENT COST OF A CEMENT PLANT

TOTAL COST



COST OF ELECTRICAL EQUIPMENT



NOTE: THE PERCENTAGE VALUES MAY VARY FROM PLANT TO PLANT

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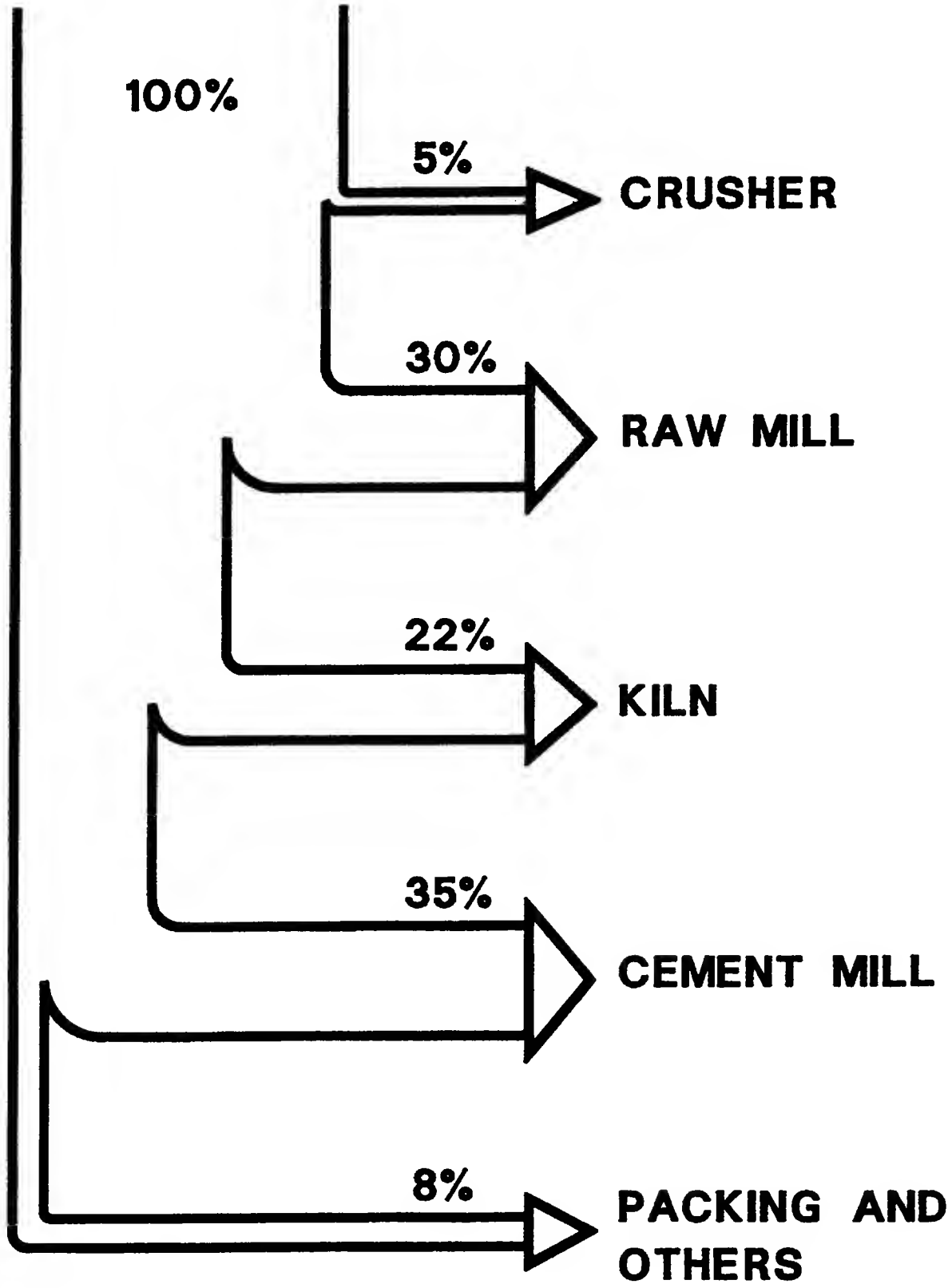
10. POWER CONSUMPTION OF A CEMENT PLANT



≈ 100 kWh/t.

≈ Peak x 1.3

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Chapter 2

Power Distribution

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Power Distribution

Fritz Richner

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1. SUPPLY VOLTAGES

In the cement industry, the supply voltages from the utility power companies are in the range of:

- 11 kV (for small plants)
  up to
- 150 kV (for large plants)

In power distribution systems we distinguish between equipment for:

- 'high voltage' > 50 kV
- 'medium voltage' 1-50 kV
- 'low voltage' ≥ 1 kV

The design of the equipment (e.g. circuit breaker) varies according to the requirements of the different voltage levels and power ratings.

Note: The medium voltage range is very often also called 'high voltage', e.g. 6 kV high voltage motors.

2. HIGH VOLTAGE TRANSFORMER STATION

Depending on the layout of the cement plant and on the type of switchgear installed this station is located on the periphery or in the centre of a cement plant. For safety reasons a cement plant is preferably fed by two incoming cables or overhead lines. The transmission, voltage can vary between 11 kV and 150 kV and is normally fixed by the power supplier. The high voltage will be transformed to 4 to 11 kV by means of preferably two transformers, one of as stand-by.

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For power distribution system-arrangement see Fig. 2.1. For power distribution single line diagram see Fig. 2.2.

Figure 2.1 Example of a Power Distribution System-Arrangement

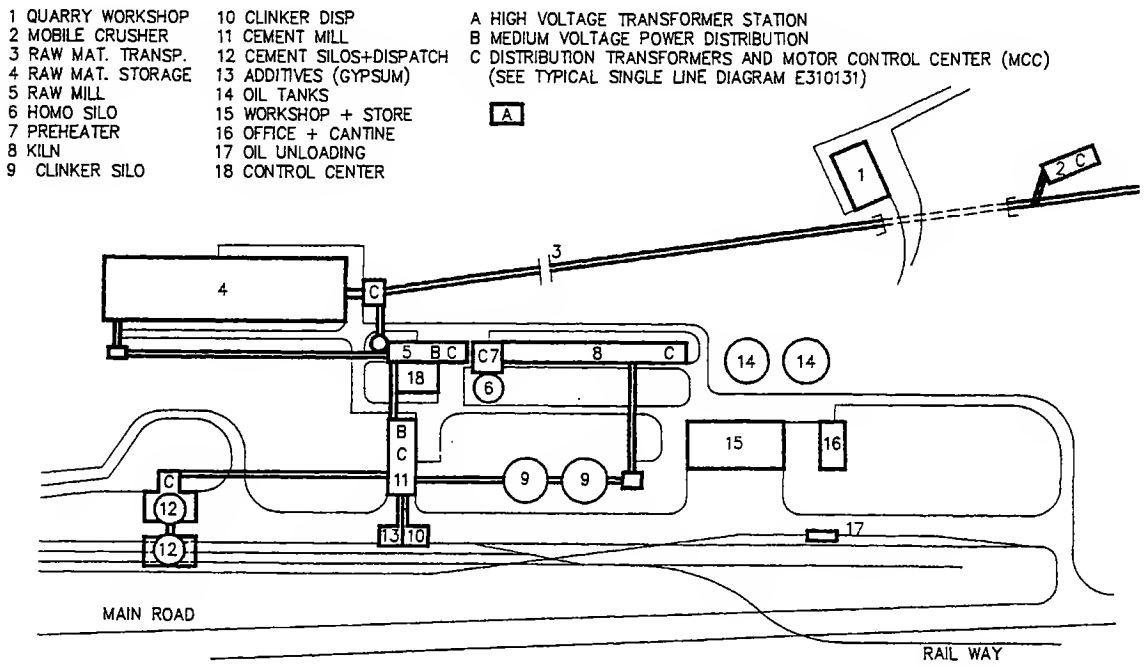
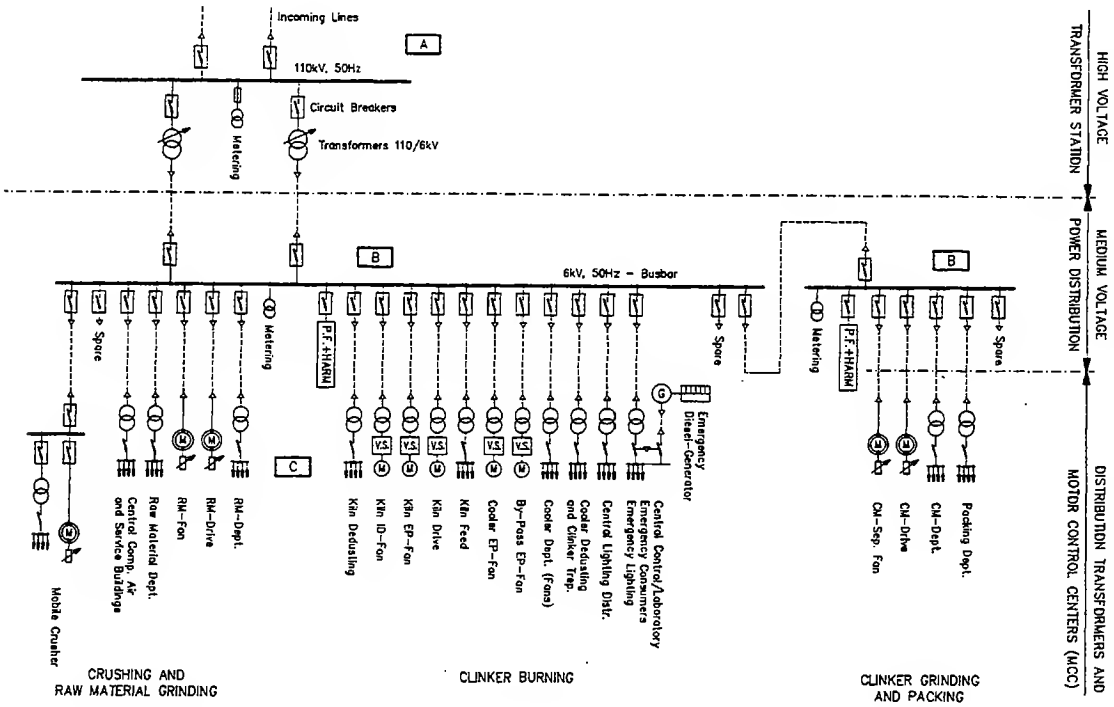


Figure 2.2 Example of a Power Distribution Single Line Diagram



(see typical power distribution system arrangement dwg. F44623)

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3. MEDIUM VOLTAGE POWER DISTRIBUTION

The medium voltage power distribution is usually located in an electrical room in the centre of a cement plant, this is order to minimise cabling and installation costs (centralised MV-distribution).

However, depending on the layout of the plant, the MV-power distribution may be arranged in a decentralised manner, i.e. the MV-distribution might be located in load centres common with the LV-distribution transformers and motor control centres. For comparison of 'centralised' and 'decentralised' arrangements refer to single line diagram Fig. 3.1.

Decentralised arrangements may result in the application of a higher quantity of circuit breakers, control equipment, larger electrical rooms, a more sophisticated protection scheme and requires a somewhat more costly maintenance. Costs for MV-cabling may however be smaller.

The medium voltage level normally ranges from 4 kV up to 11 kV. There is a strong tendency to use the higher voltage of 11 kV; 50 Hz, resp. 13.8 kV; 60 Hz, because a higher working voltage results in smaller cable cross-sections (i.e. lower investment costs) and less voltage drops (i.e. less energy losses).

The distribution station is equipped with a main busbar, the incoming circuit breakers and the different outgoing circuit breakers to the distribution transformers and 'high voltage' motors. Distribution transformer stations are located in the different load centres, namely the crushing plant, the raw meal grinding plant, the kiln plant, the cement grinding plant and the packing plant.

All high and medium voltage equipment (transformers, cables and motors) has to be protected against overload, short circuit, earth fault, over voltage, etc. to guarantee a selective fault isolation.

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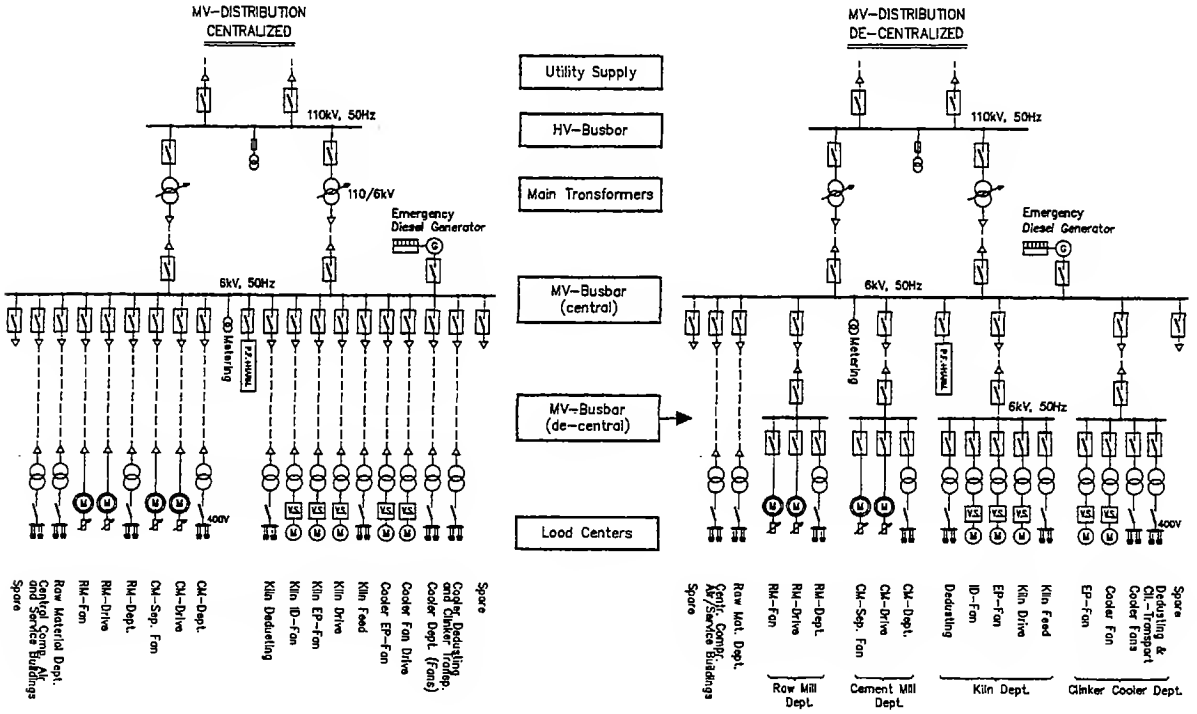
Figure 3.1 Centralised & de-centralised medium voltage distribution typical single-line-diagram



Load Centers: Contain Distribution Transformers, Motor Control Centers, VS-Motor Controls, etc.

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4. DISTRIBUTION TRANSFORMERS AND MOTOR CONTROL CENTRES (MCC)

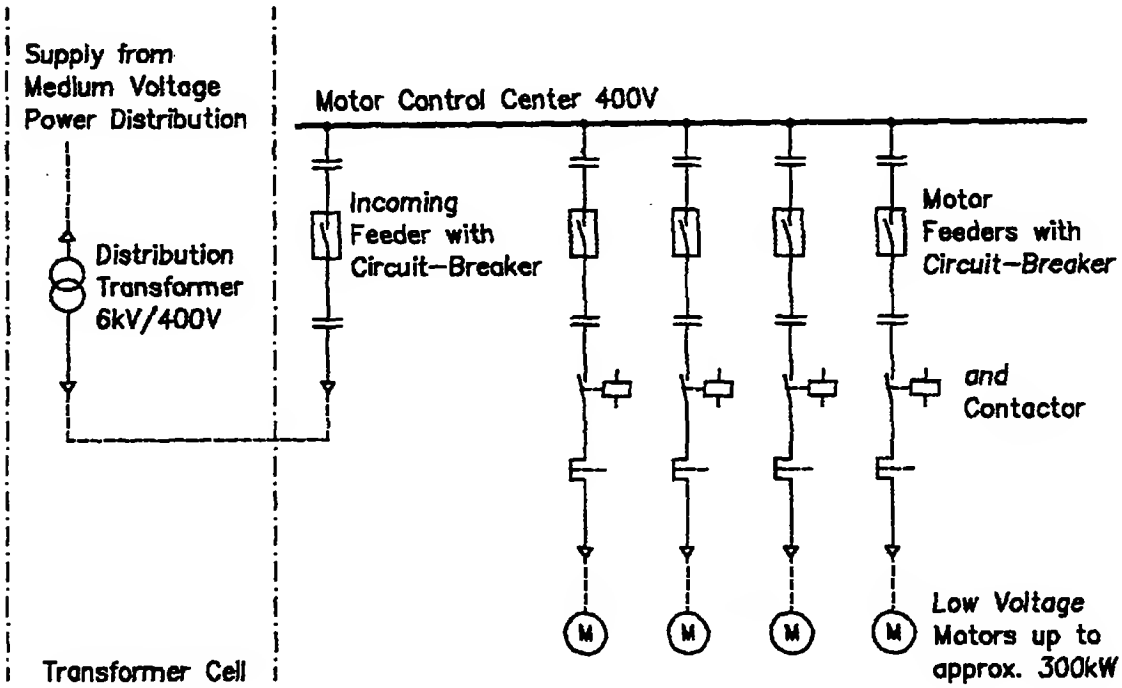
Distribution transformers and MCC's are located in the different load centres of the plant.

The transformers have a capacity of 630 up to 2000 kVA depending on the power requirement and a low voltage level of 400 V up to 660 V.

Distribution transformers are normally installed indoors in individual transformer cells.

The low voltage switch gear and MCC's are located in electrical rooms of the respective plant buildings.

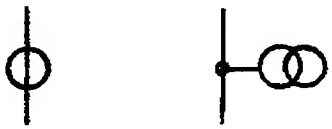
Figure 4.1 Single line diagram 'Example of a load centre with distribution transformer and MCC'



5. HIGH VOLTAGE EQUIPMENT

The high voltage transformer station generally consists of following components:

Lightning arrestors (protect the substation against over voltage such as lightning)



Current and voltage transformers (measure the amount of energy consumed)

Off-load isolator (isolate a circuit at no load e.g. for maintenance purposes)

On-load isolator (disconnect a circuit at max. double-rated current)

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Earthing switches (connect a circuit to the earth for safety reasons, during maintenance activities)

Circuit breakers (disconnect a circuit under load and in case of fault)

Busbars (form the electrical power circuit)

Step-down transformer (transforms incoming voltage to the plant power distribution level e.g. 110 kV to 7 kV)

The following 3 types of transformer stations are mainly used:

- Outdoor transformer station with conventional switch gear:

  Characteristics:
  - biggest surface area required
  - medium investment costs for high voltage switch gear
  - time consuming for maintenance (cleaning)

- Indoor transformer station with conventional switch gear:

  Characteristics:
  - medium surface area required
  - lowest investment costs for high voltage switch gear
  - medium time required for maintenance

- Indoor transformer station with metal-clad, SF_6, gas insulated or vacuum type switch gear:

  Characteristics:
  - smallest surface area required
  - highest investment costs for high voltage
  - switch gear
  - needs the least of maintenance

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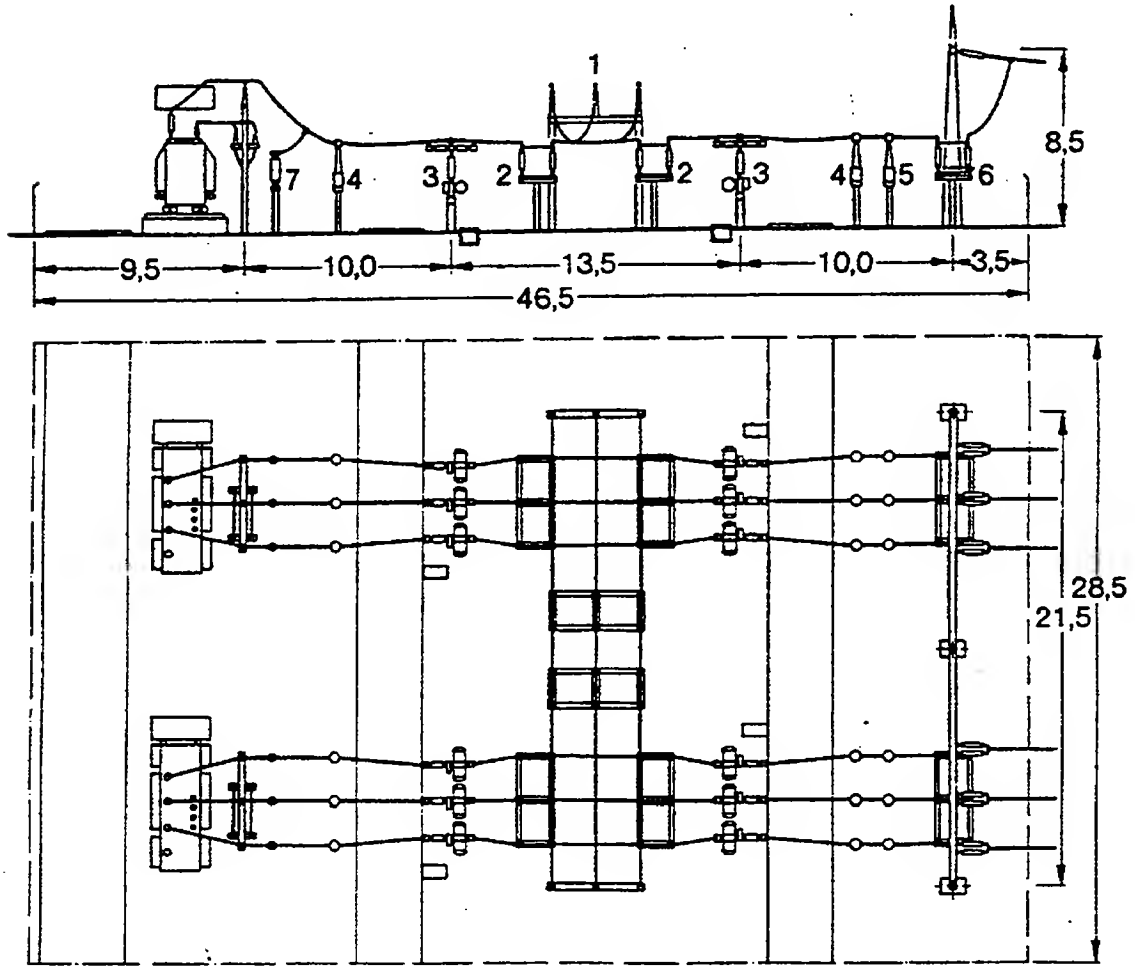
Figure 5.0.1 110 kV Outdoor Transformer Station
Typical arrangement with conventional switch gear



Legend:

1	Busbar system
2,6	Off-load isolator
3	Circuit breaker
4	Current transformer
5	Voltage transformer
7	Lightning arrestor
8	Power transformer

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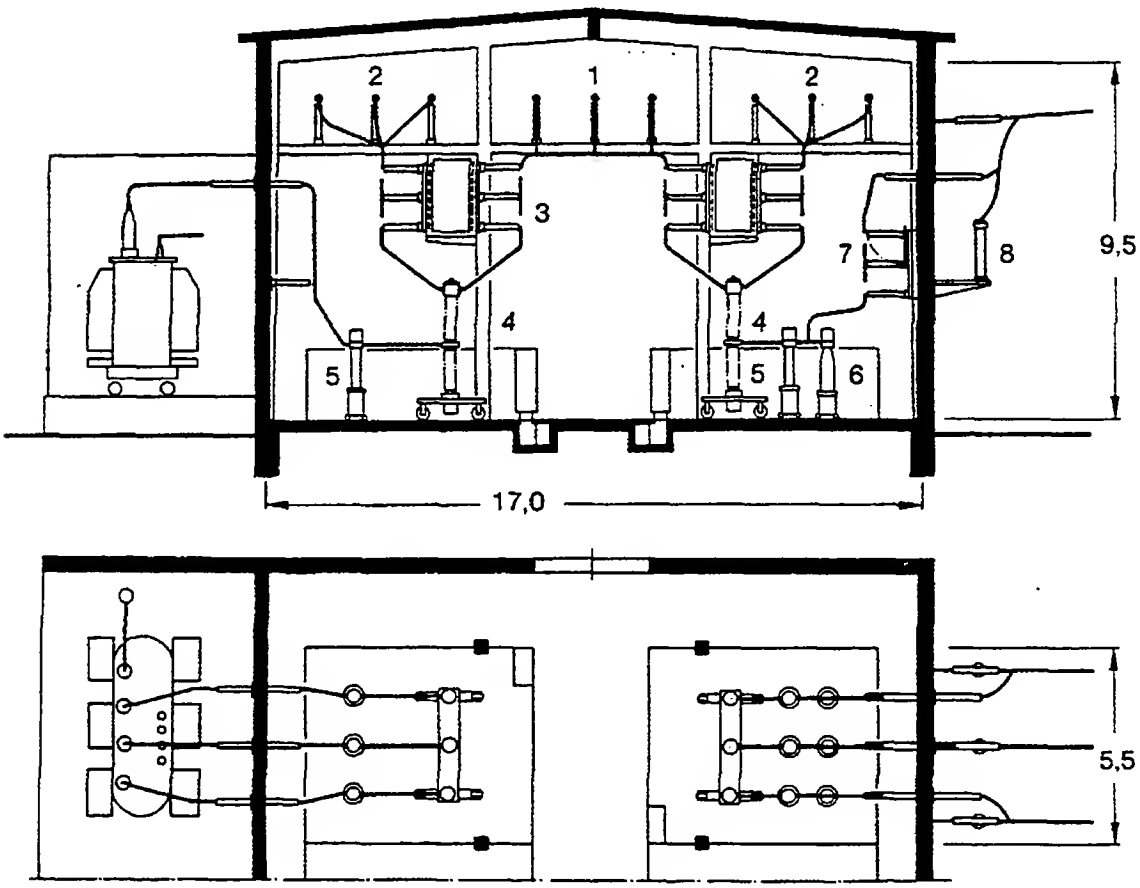
Figure 5.0.2 110 kV Indoor Transformer Station
Typical arrangement with conventional switch gear



Legend:

1, 2	Busbar systems (double busbars)
3, 7	Off-load isolator
4	Circuit breaker
5	Current transformer
6	Voltage transformer
8	Lightning arrestor
9	Power transformer

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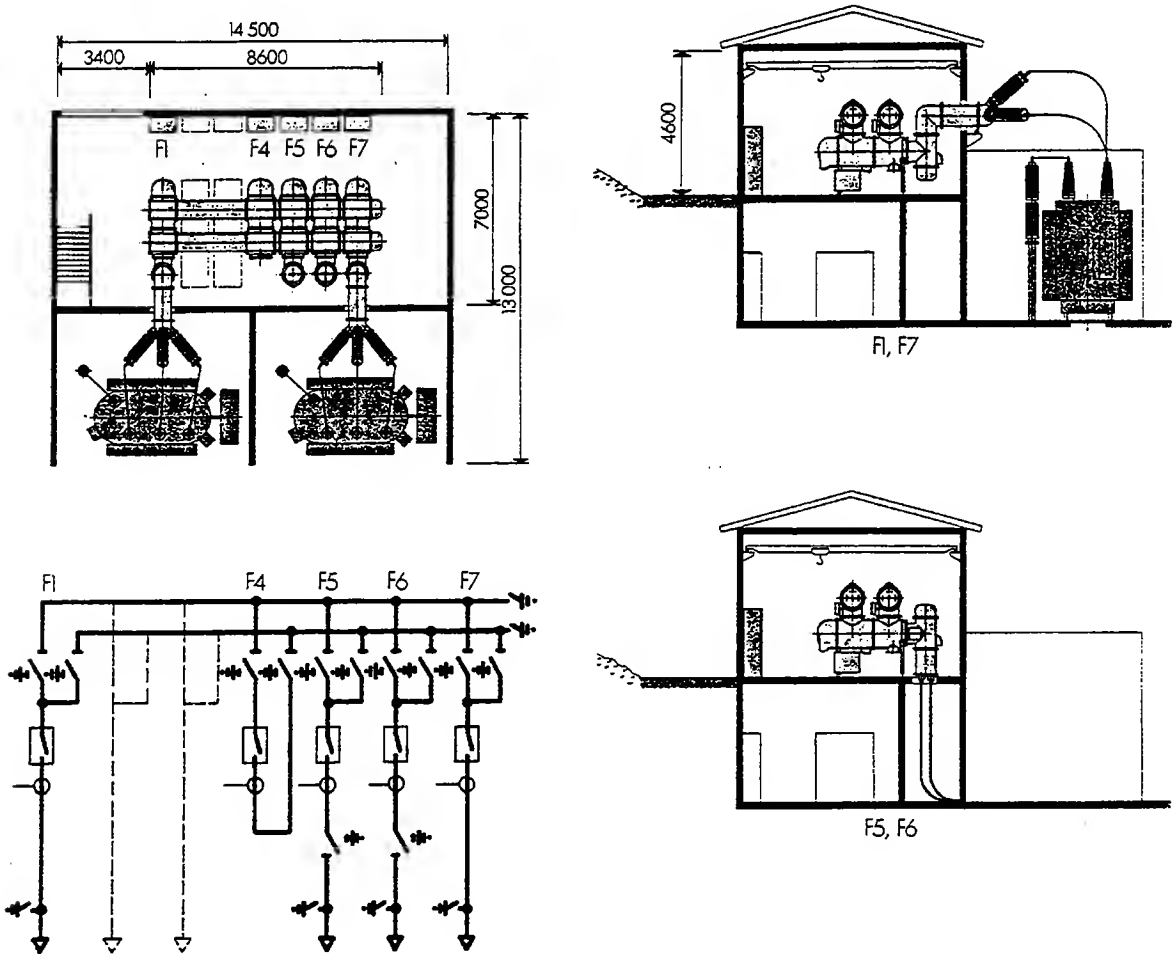
Figure 5.0.3 110 kV Indoor Transformer Station
Typical arrangement with metal-clad, SF_6 gas insulated switch gear



Legend:

F5, F6	Incoming feeders consisting of off-load isolators, circuit breaker, earthing switches, current and voltage transformers
F1, F7	Outgoing feeders consisting of off-load isolators, circuit breaker, earthing switches, current transformers
F4	Buscoupler (double busbars)

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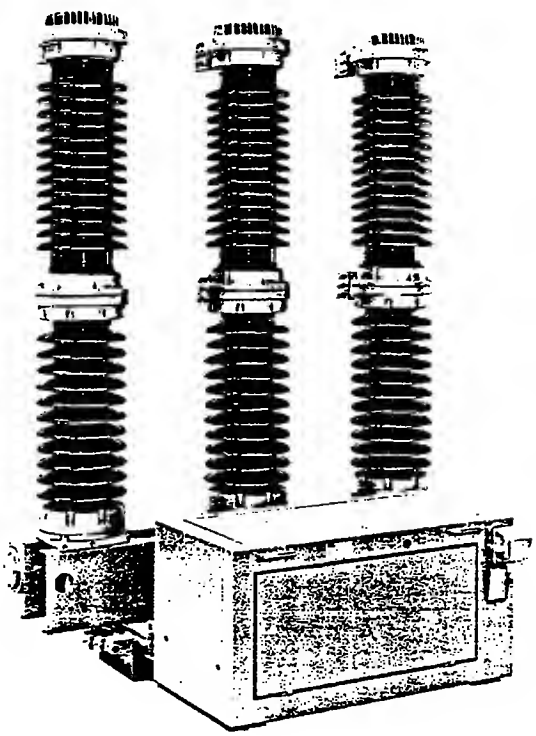
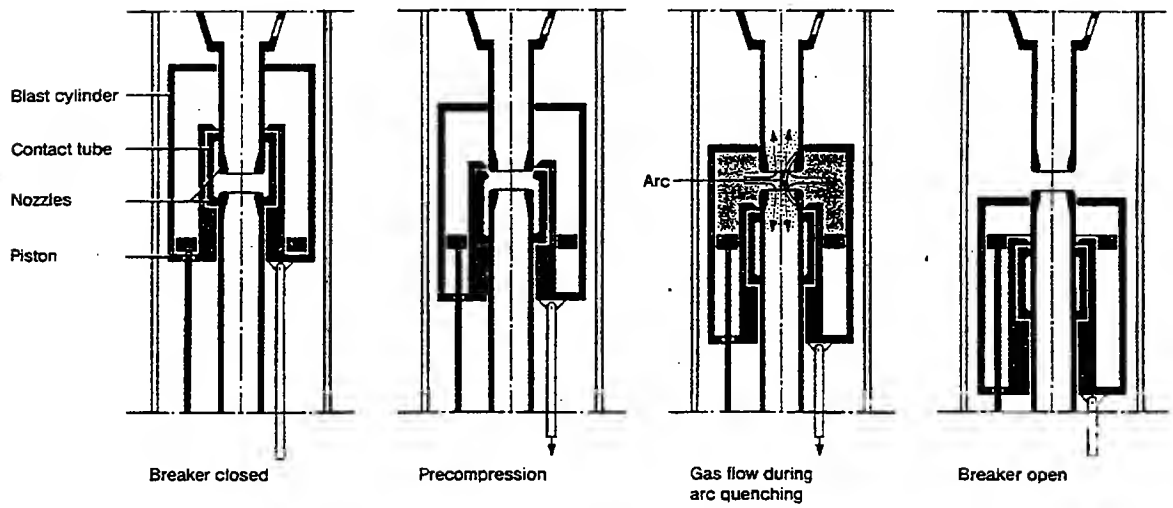
5.1 Circuit Breakers

In high voltage installations mostly SF_6 gas circuit breakers are applied. The low oil content circuit breakers are still used but manufacturing will run out in the next years.

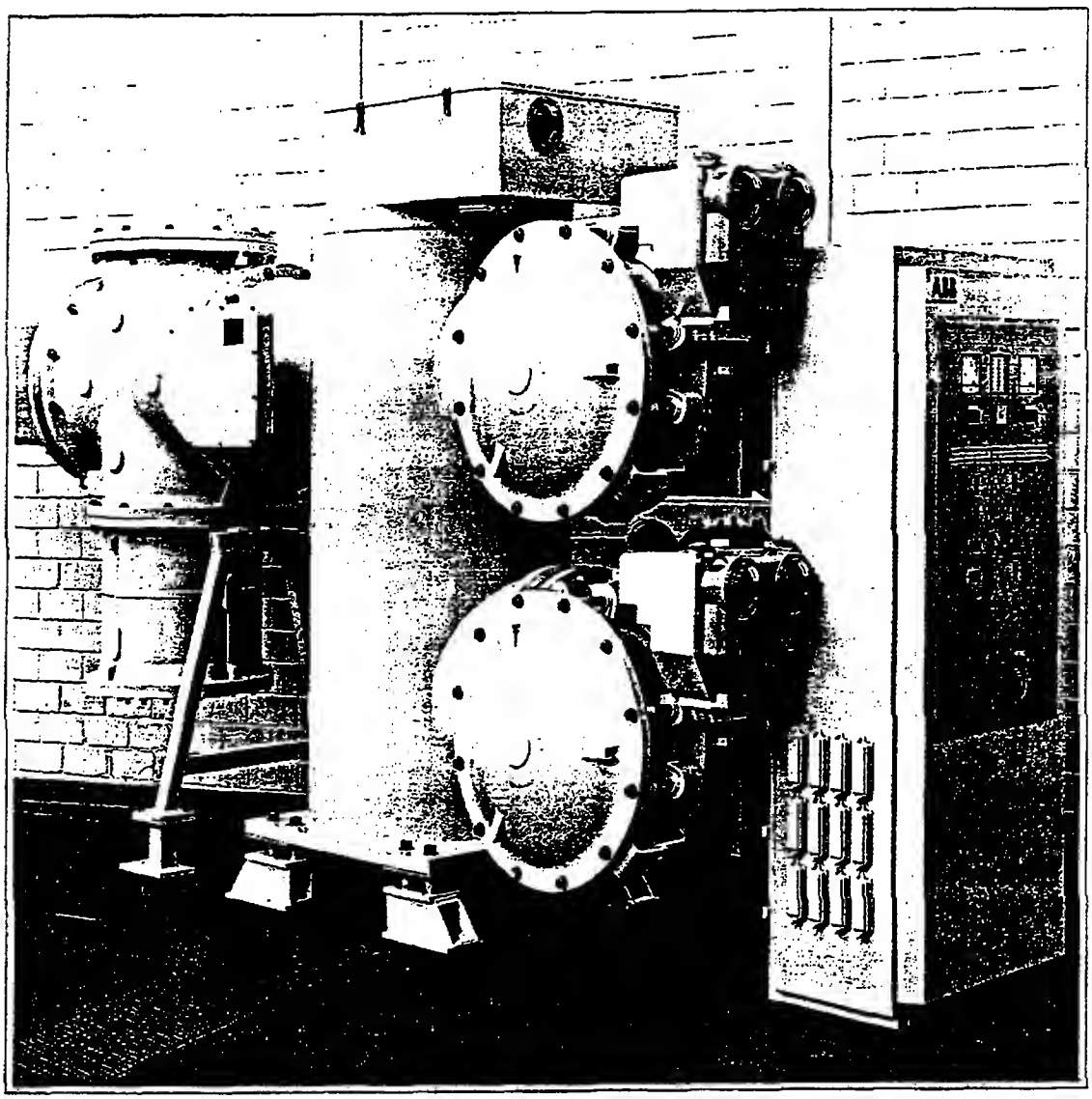
Figure 5.1.1 SF6 Gas Circuit Breaker
3-pole circuit breaker with operating mechanism



5.1.2 Schematic function of arc quenching mechanism of a circuit breaker pole



5.1.3 Metal-clad, SF6 Gas-Insulated Circuit Breaker



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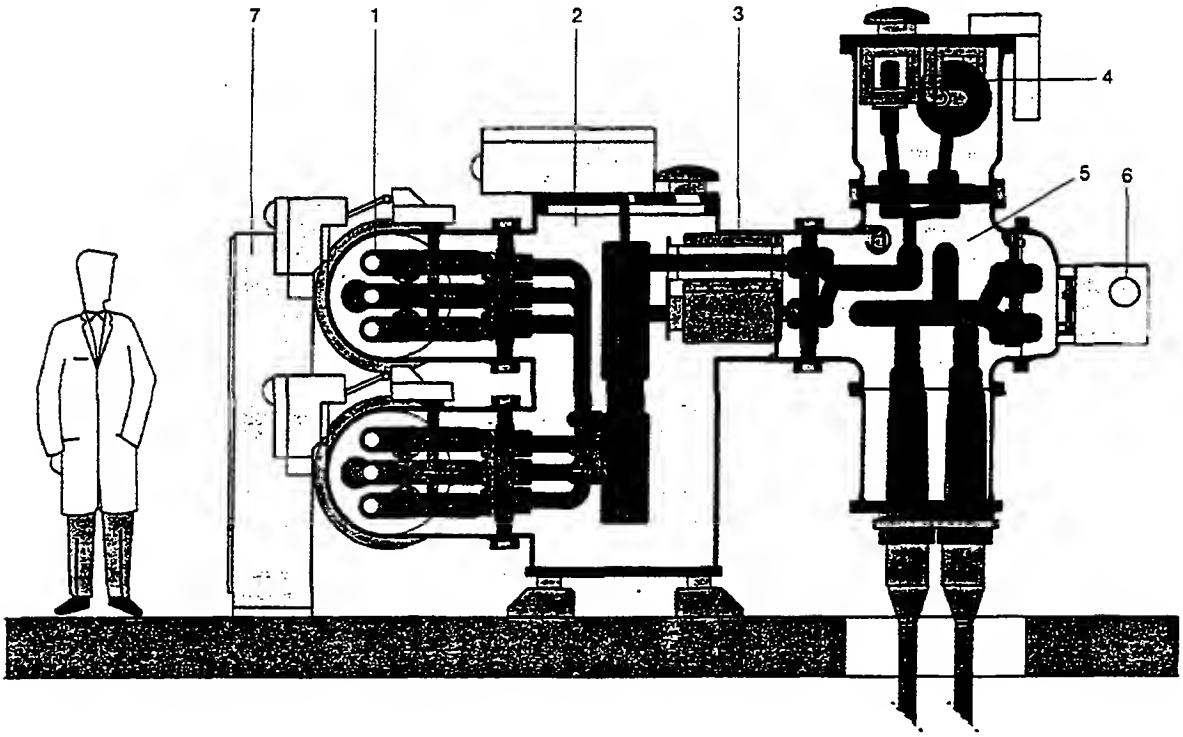
5.1.4 Cross section of double busbar cable feeder:



Legend

1	Busbar with combined disconnector/earthing switch
2	Circuit breaker
3	Current transformer
4	Potential transformer
5	Cable end unit with combined disconnector/earthing switch
6	Fast acting earthing switch
7	Control cubicle

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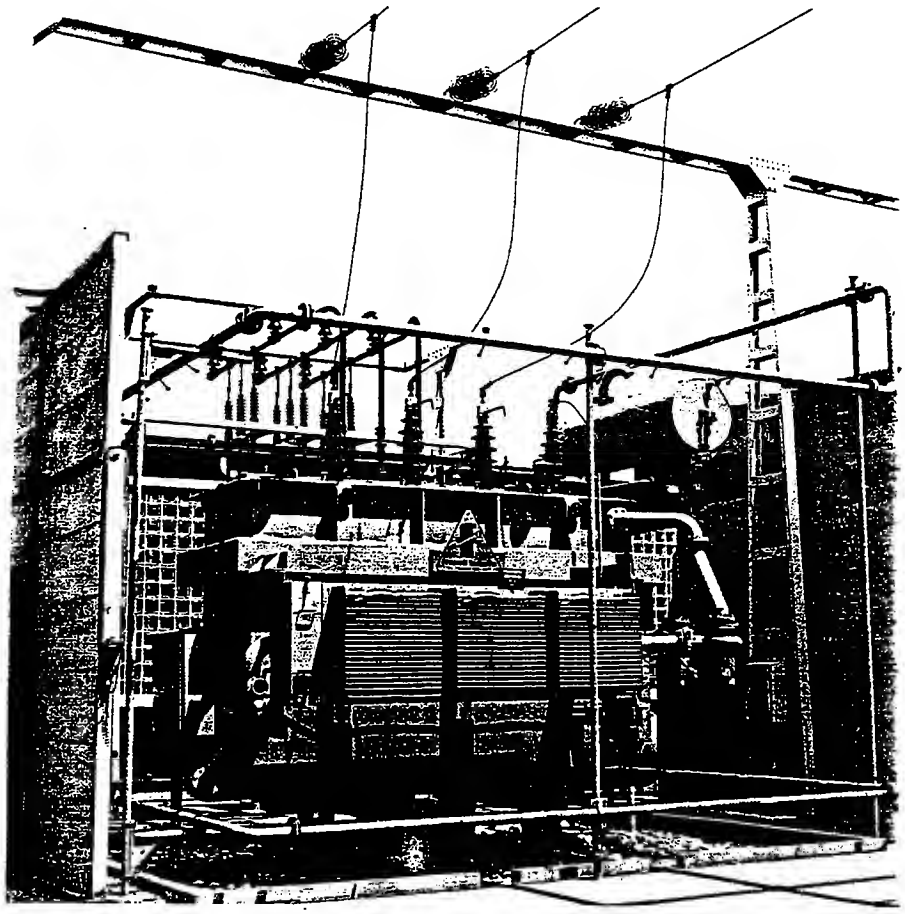
5.2 High Voltage Transformer

Large high voltage transformers have an efficiency of up to 99%. They are always filled with oil which still copes best with all insulation and temperature problems. To meet the requirement for a constant voltage level in the cement factory under various loads and with varying voltage levels of the utility power supply, the transformers should be equipped with a tap switch which automatically increases or decreases the secondary voltage in steps of about 1 to 1.5%. Large transformer are equipped with air or water cooling equipment (a 20 MVA transformer with an efficiency of 98% still produces a flow of 400 kW of heat). Temperature sensors and the so-called 'Buchholz-Relay' protect the transformer against overload and insulation failures. The Buchholz-Relay detects gas bubbles which collect at the highest point of the transformer, and thus gives a good indication of insulation or local over temperature problems, in the transformer. The vector-group of a transformer is often Yd5, where 'Y' stands for primary star connection, 'd' for secondary delta connection, and '5' gives the phase relation between the two systems.

Figure 5.2 High voltage transformer



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6. MEDIUM VOLTAGE EQUIPMENT

Medium voltage power distributions consist mainly of the same components as described in paragraph 5. They are generally of the indoor type. Today draw-out type cubicles are commonly used. Different equipment can be mounted on identical 'trucks'. The trucks are easy to handle and allow a quick replacement in case of a failure. The figure below shows a medium voltage distribution station.

Figure 6.0.1 Medium voltage substation

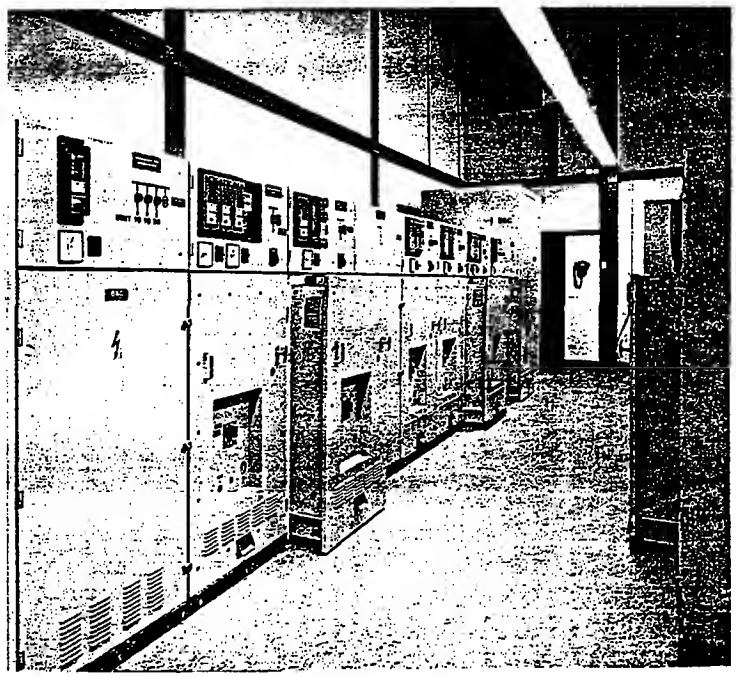
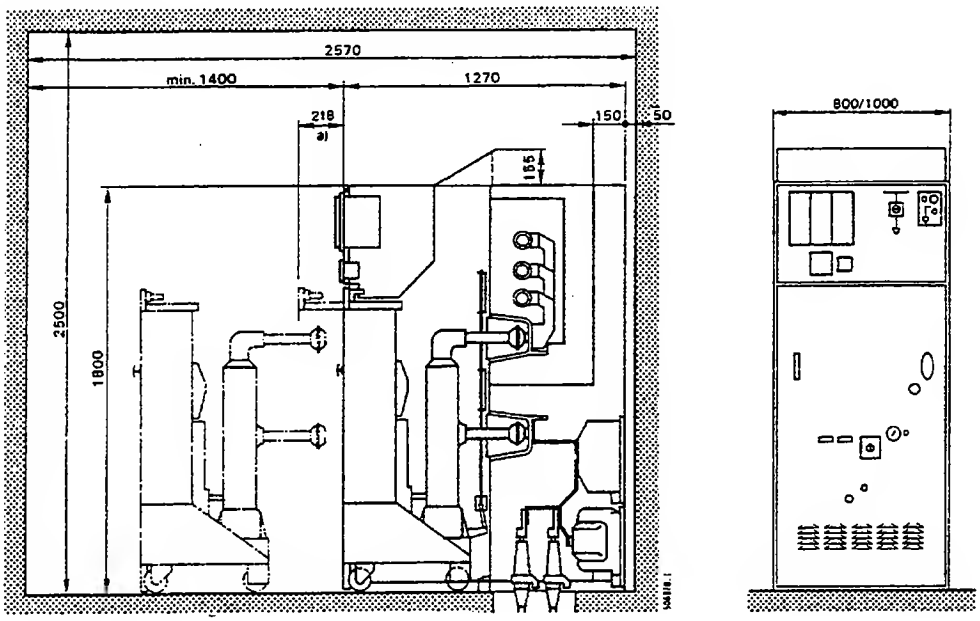


Figure 6.0.2 Typical arrangement with metal-clad SF_6 switch gear

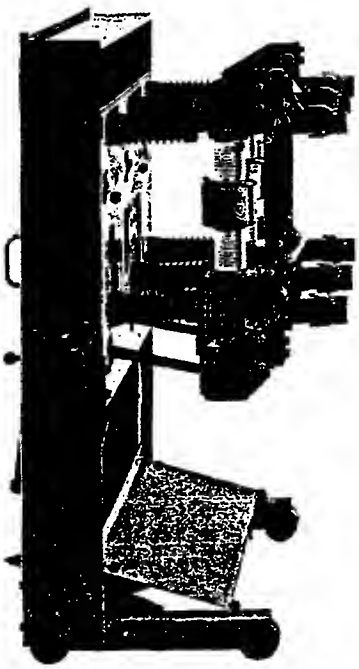


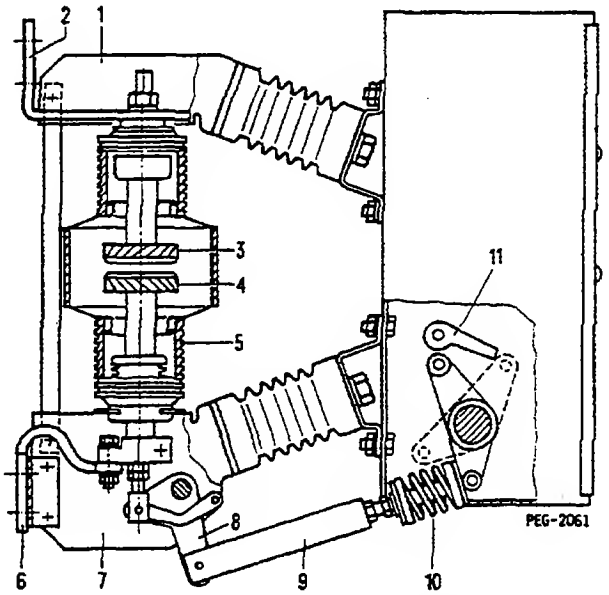
6.1 Medium Voltage Circuit Breakers

Vaccum- and SF_6 circuit breakers are normally installed in the medium voltage power distribution systems.

Low oil content circuit breakers are still manufactured but are less used due to higher equipment costs.

Figure 6.1.1 Truck mounted vacuum circuit breaker





1 Upper interrupter support
2 Upper terminals
3 Fixed contact
4 Moving contact
5 Ceramic insulator
6 Lower terminal
7 Lower interrupter support
8 Angled lever
9 Insulated actuating rod
10 Contact pressure spring
11 Trip pawl

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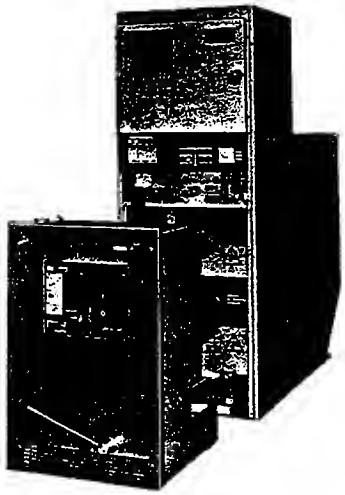
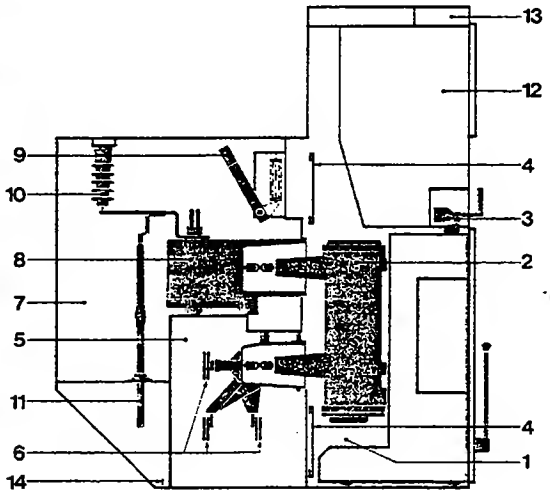
6.1.2 Truck mounted SF_6 circuit breaker





Legends
1 Breaker compartment
2 circuit breal
3 LV plug-in connection between breaker and LV auxiliaries
4 metal safety shutters covering fixed disconnecting contacts
5 Busbar compartment
6 Busbars
7 Cable-connection compartment
8 Current transformers
9 Earthing switch
10 Potential dividers
11 HV solid-dielectric cables
12 LV auxiliaries compartment
13 LV wiring trough
14 Earthing bus

Properties of vacuum- and SF_6 circuit breakers

- **Application:**
  Protection and switching operation for
  - lines
  - cables
  - transformers
  - motors
  - capacitors
- **Criterias:**

	Vacuum	**SF_6**
Interrupter service life	20'000 to 30'000 C-0 operations	10'000 to 20'000 C-O operations
Service interval	Lubrication of mechanism(max. 10 years)	Lubrication of mechanism (max. 10 years)
Overhaul of interrupter	Interrupter to be replaced	Interrupter can be reconditioned
Switching of lines, cables, transformers, capacitors	Well suited	Well suited
Switching of motors	Well suited but measures may be necessary to limit over voltages	Well suited normally no measures necessary to limit over voltages

6.2 Medium Voltage Contactors

The psychical principles are similar to a circuit breaker, except that the contactor cannot interrupt short circuit currents. The contact system is optimised for high numbers of rated current operations.

High rupture capacity current limiting fuses in conjunction with the contactor are therefore required for the short circuit protection.

Overload protection is ensured by separate relays.

Fused contactors can be used as motor and transformer feeders.

Examples for max. fuse rating 250 A, 6 kV:

- distribution transformers max. 2000 kVA
- motor with max. starting current 1350 A and max. starting time 10 sec. - max. 1690 kW.

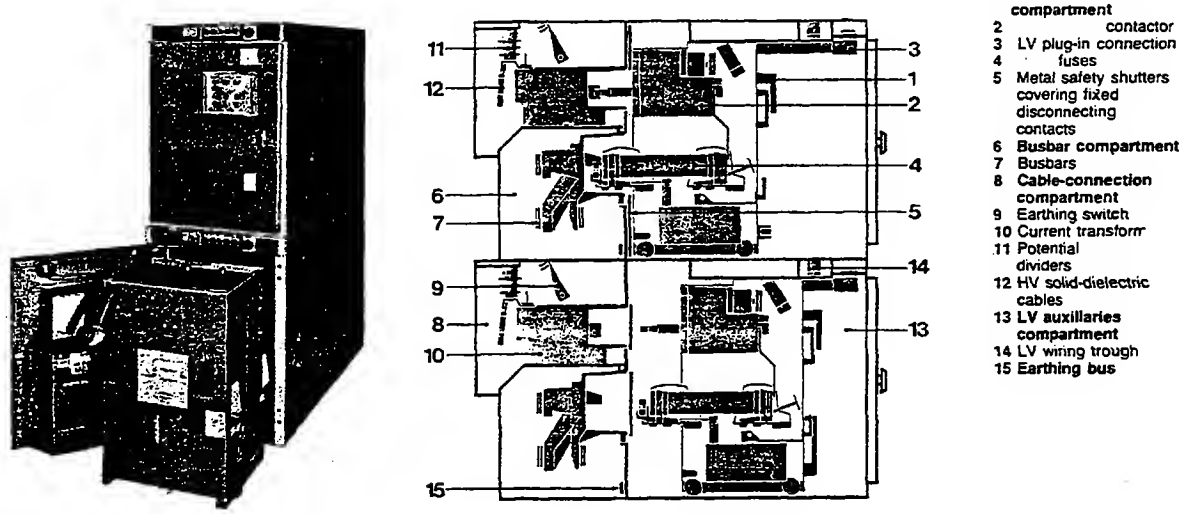
Advantages of fused contactors:

- very compact design
- more economical than circuit breakers

Disadvantages of fused contactors:

- tripping mechanism with auxiliary contact required for each fuse
- limited application (voltage, current)
- fuses to be replace

Figure 6.2 Contactor panel (double tier)



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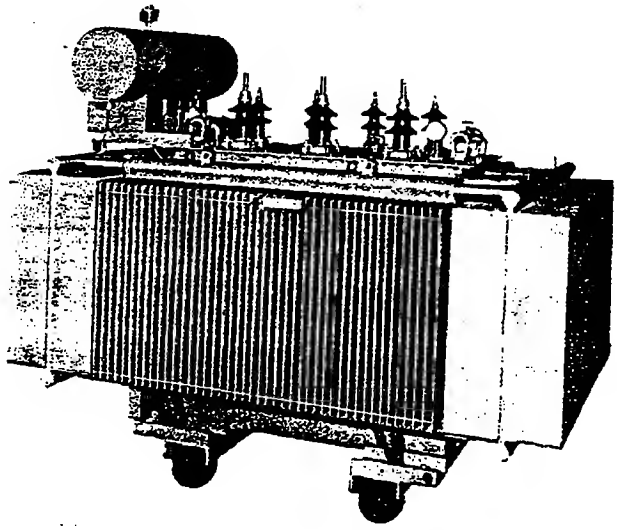
6.3 Medium Voltage Transformers

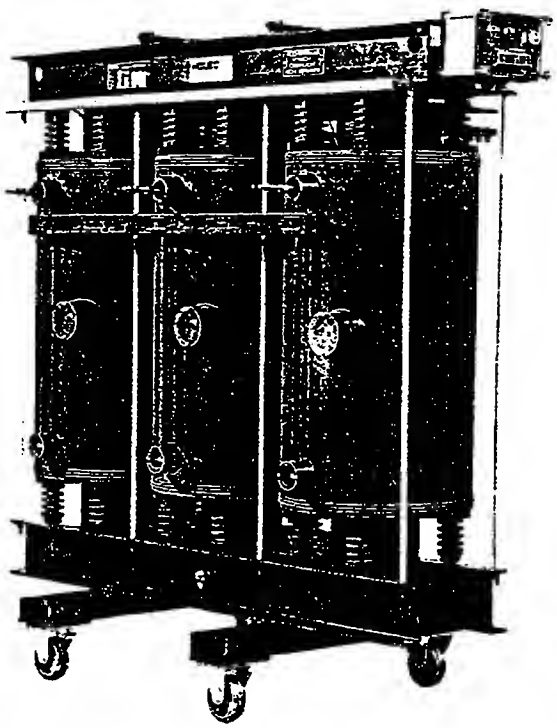
These transformers too are normally mineral oil-immersed. For special applications silicone oil is used which has less heat development during fire than mineral oil (10%).

Figure 6.3.1 Oil-immersed three-phase distribution transformer with oil conservator (hermetically sealed transformers without conservator are normally used up to 1000 kVA, resp. up to 20 kV)



So called 'dry' transformers which use a synthetic resign as insulation are built up to 10 MVA. They are more expensive than oil transformers.

Figure 6.3.2 Dry-type three-phase distribution transformer



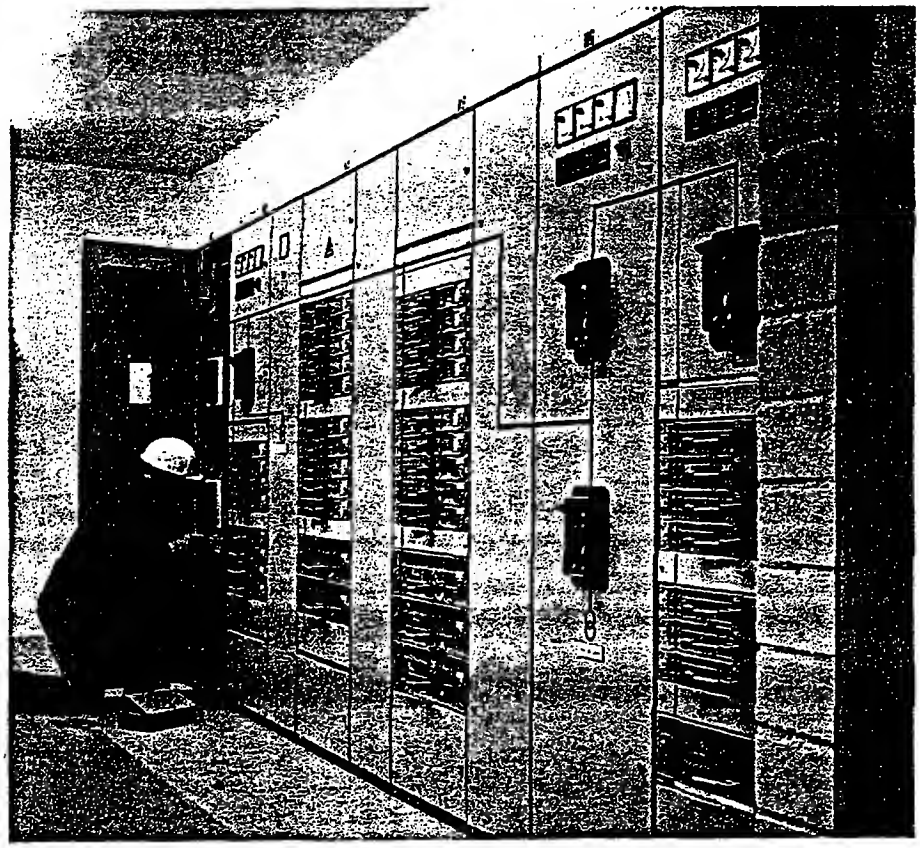
7. LOW VOLTAGE EQUIPMENT

Low voltage switch gear is located in electrical rooms in the different load centres of the plant.

7.1 Motor Control Centres

The motor control centres comprise all equipment for the remote control of the low voltage consumers (motors etc.). A fuseless and standardised execution of the feeder is preferred.

Figure 7.1 Motor control centre with outgoing feeders of the draw-out, fully-plugged design



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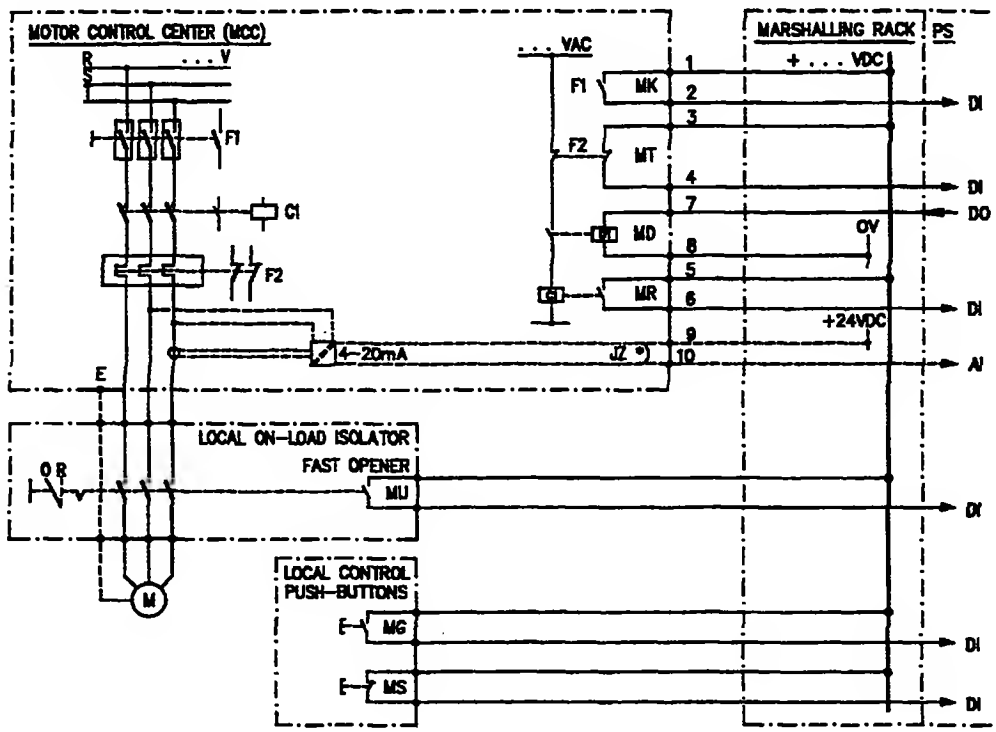
7.2 Motor Control Interface

The power- and control circuit wiring is normally standardised for each type of feeder.

For maintenance and repair purposes start/stop push buttons and OFF/READY isolator switches are located near each individual drive.

The DC-control circuits are connected via multi-core cables to the marshalling rack of the respective process stations.

Figure 7.2 Typical motor feeder circuit diagram



Legend:

MG.	Motor Local Go (Start)	DI:	Digital Input
MS:	Motor Local Stop	DO:	Digital Output
MU:	Motor Local Isolated	AI:	Analogue Input
MT:	Motor Thermal Overload	D1:	Interface Relay ... VDC
MR:	Motor Run	C1:	Contactor
MK:	Motor OK	F1:	Circuit Breaker
JZ:	Motor Power *) Option	F2:	Thermal Overload Device
MD:	Motor Start		

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8. CABLES

The cable installation is an important part in a cement plant. The investment costs are in the range of 10 to 15% of the total costs for electrical equipment.

Types of cables

Power cables for

- High voltage (e.g. 110 kV) for incoming feeder from power company
- Medium voltage (e.g. 6 kV) for medium voltage power distribution of the plant
- Low voltage (e.g. 400 V) for low voltage power distribution of the plant

Control cables for

- Low voltage (e.g. 220 V) for control circuits
- Extra low voltage (e.g. 24 V=) for process control, instrumentation, communication.

Figure 8.1 Medium voltage power cable (3-core)

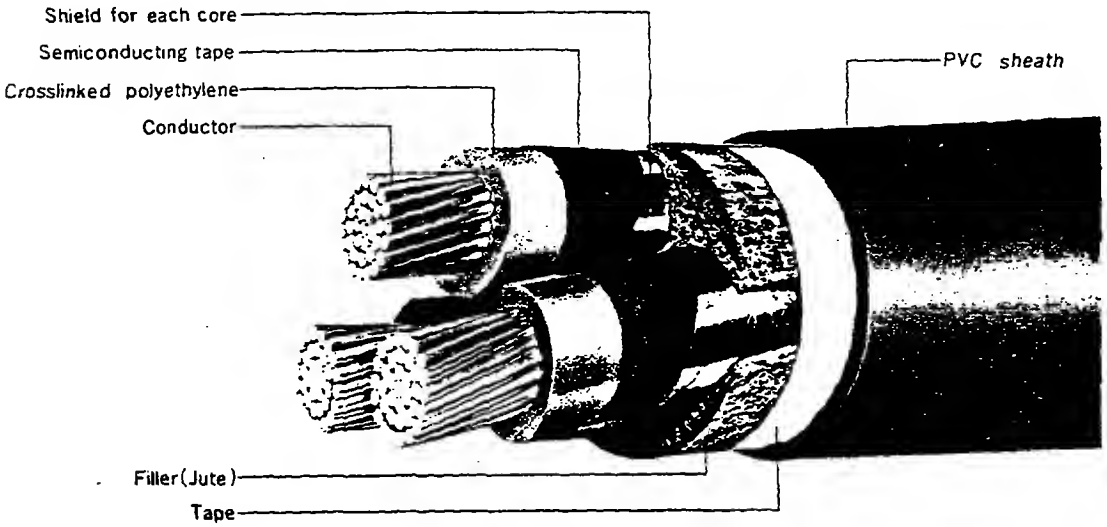

Figure 8.2 Low voltage power cable (4-core)

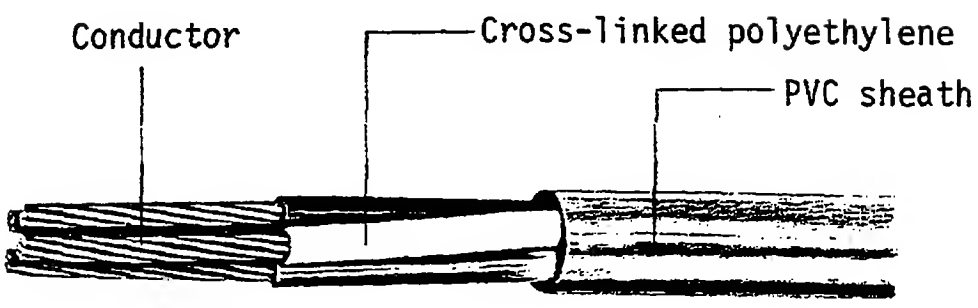

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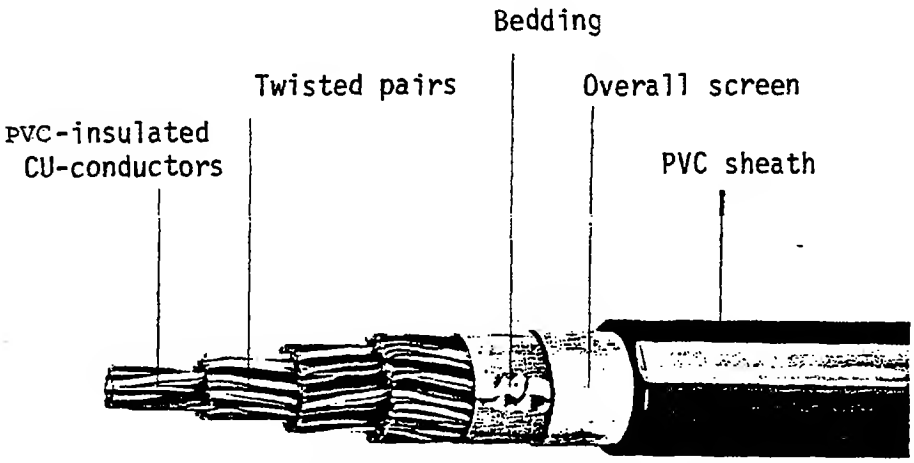
Figure 8.3 Extra low voltage cable (multi-core)



Polyethylene resp. cross-linked polyethylene (XLPE) insulated cables are the most widely installed power cables today in a cement plant. Ethylene-propylene-rubber (EPR) insulated cables will in future replace the PE resp. XLPE-cables.

Copper conductors are preferable to aluminium conductors due to the simpler installation method.

For special applications, cables with flame retardant, non-corrosive sheath material (EVA) can be installed.

Dimensioning of cables

Power cables have to be carefully dimensioned in respect of

- Current carrying capacity by taking into account
  - ambient temperature
  - kind of installation (ground, air grouping)
- Voltage drop (power loss)
- Thermal and dynamic short-circuits strength.

Installation of cables

In cement plants most of the cables are mounted on cable trays inside the buildings, on bridges, in walk-through cable tunnels or they are installed in pipe systems. Therefore, additional mechanical protection by armouring is generally not required.

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9. POWER FACTOR AND ITS IMPROVEMENT

9.1 General

The induction motor is the largest producer of reactive power in a cement plant and shall be used here as an example.

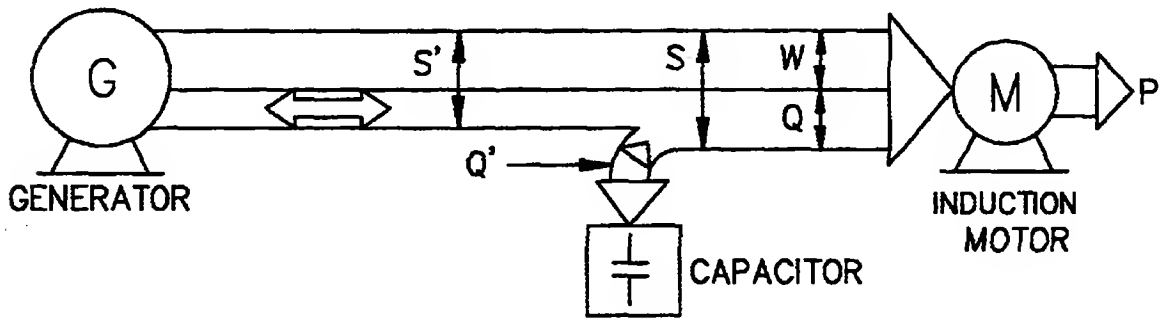
The induction motor draws two power components from the supplying network, or as shown below, from the generator, i.e.:

- the active power is transformed by the motor into mechanical energy
- the reactive power is transformed by the motor into magnetic energy; but with every change of polarity (with the frequency of the supplying network), the magnetic energy is transformed back to electrical energy.

in other words, it flows back and forth between the motor and the generator.

It can easily be seen that the reactive power is a burden on the generator and the supplying cables. It appears as a current like the active current and causes losses in the cables, transformers and in the generator.

Figure 9.1.1 Typical power flow in motor circuit



Legend:

S:	apparent power	kVA
Q:	reactive power	kVar
W:	active power	kW
P:	mechanical power	kW

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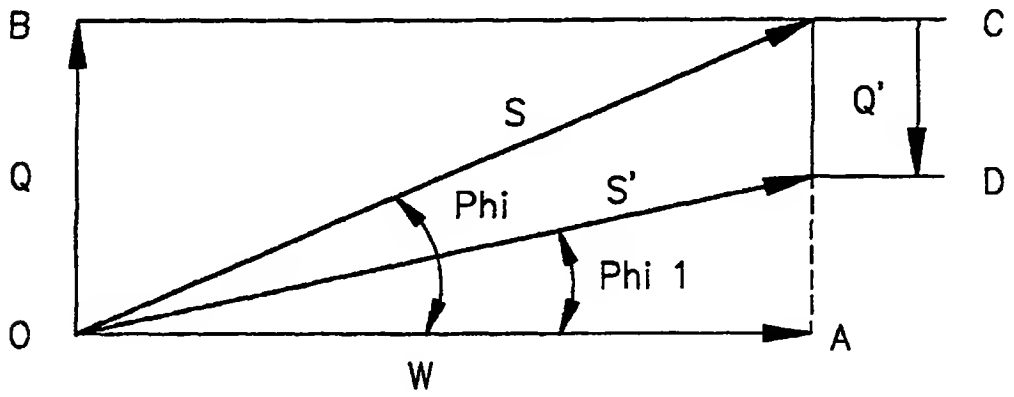
Figure 9.1.2 Vector diagram



Legend:

OA:	Active power W	(kW)
OB:	Inductive reactive power Q	(kVar)
OC:	Apparent power S	(kVA)
CD:	Capacitor reactive power Q	(kVar)
Phi:	Phase angle, uncompensated	
Phi_1	Phase angle, compensated	
Cos phi:	$\frac{W}{S}$	

The electricity authorities generally demand a minimum power factor (cos phi) to limit the losses in their own power distribution system.

One of the most common tools used to improve the power factor is the capacitor. A capacitor transforms reactive power into electrostatic power and back to reactive power with the frequency of the network.

The capacitor can be compared with an expansion tank.

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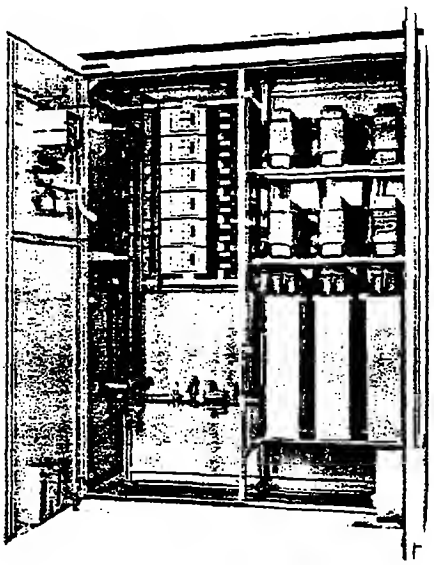
9.2 Power Factor Correction

The plant power factor is normally corrected in three different modes.

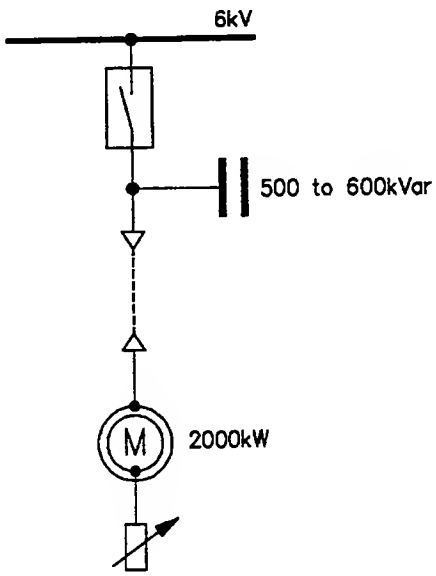
- For low voltage motors:
  They can be compensated through automatically regulated reactive current compensation plants situated at each low voltage power distribution.

Figure 9.2.1 Automatically controlled capacitor bank with line reactors (to cope with the harmonic content)



- For high voltage motors with constant speed:
  To correct the power factor, capacitor banks suitably sized for each individual high voltage motor can be connected and disconnected from the medium voltage power distribution with the respective motor circuit breaker.

Figure 9.2.2 Direct compensated high voltage motor



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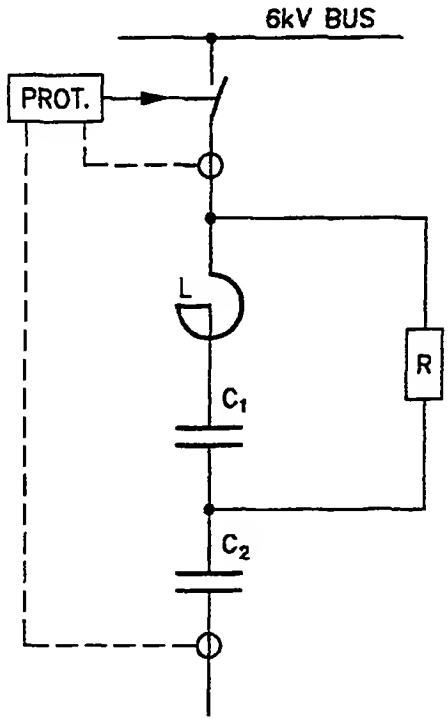
- For large variable speed drives:
  It is recommended to compensate large variable speed drive systems through central reactive power compensation plants, correcting the power factor as well as higher harmonics generated by the various kinds of converters (one plant per medium voltage busbar section).

  Prior to the installation of a compensation plant, a detailed network analysis must be carried out.

Figure 9.2.3 Circuit diagram of a harmonic absorber and power factor compensation circuit (1 MVAr)



Legend:

L:	Inductance,	14.1 mH
R:	Resistor	50 Ω
C_1:	Capacitor	720 µF
C_2:	Capacitor	90 µF
PROT:	Protection	(MCX 912)

10. ENERGY- / POWER-METERING

The power distribution scheme of a cement plant shall comply with the process requirements. Independent process departments receive independent power supplies and distributions. Therefore, metering equipment shall be installed at:

- the H.V. incoming feeder (metering for energy invoice)
- the M.V. outgoing feeders to the individual process departments
- the M.V. outgoing motor feeders
- the L.V. outgoing non-process feeders
- the L.V. outgoing main motor feeders

This allows for detailed information such as:

10.1 Metering for energy invoice

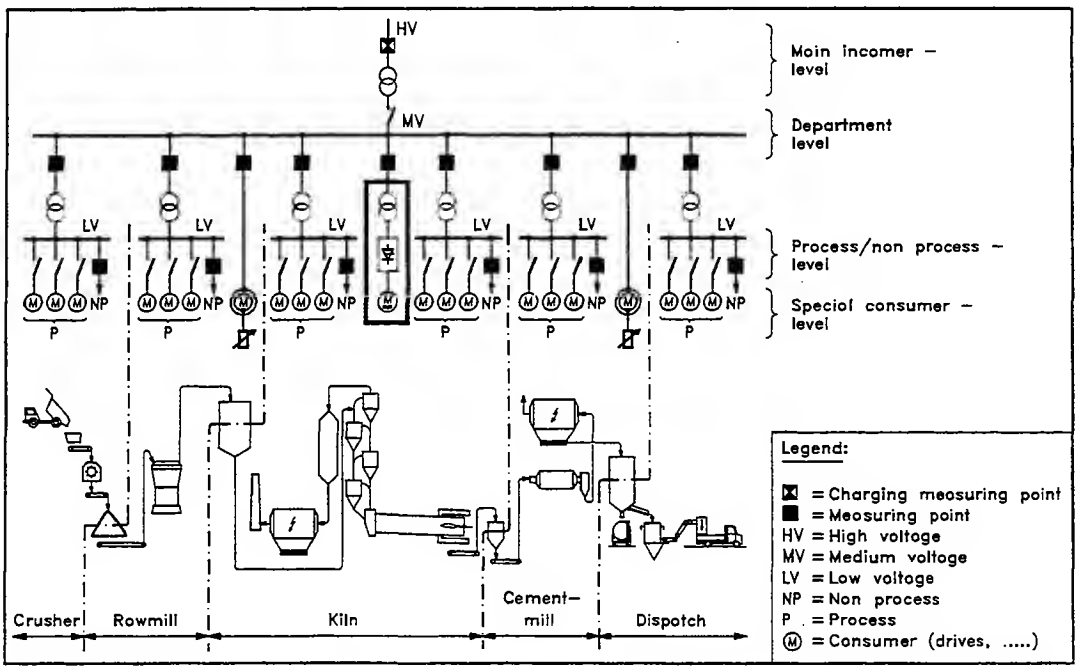
- total active energy consumption (kWh)
- total reactive energy consumption (kVarh)
- power factor (cos phi)
- total power demand (kW)

10.2 Metering for internal use only

- energy consumption per department (kWh)
- specific energy consumption per department (kWh/t)
- power demand per department (kW)

For a typical arrangement of metering equipment see Fig. 10.2.

Figure 10.2 Energy-/power metering in a cement plant



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11. SAFETY PRECAUTIONS ON ELECTRICAL EQUIPMENT

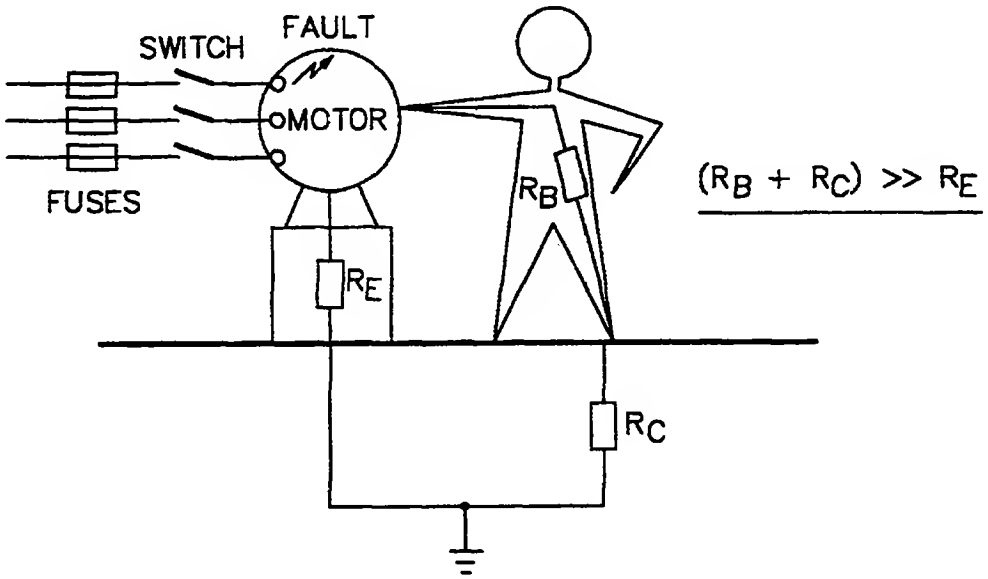
11.1 Introduction

The use of electrical equipment, from the main feed down to the hidden auxiliary servo-motor at the far end, spreads potential dangers all around the factory. The application of the safety precautions and regulations is the duty of all employees.

All electrical systems are grounded to earth to reduce the shock hazard to personnel and to provide a path to ground for currents induced in the system by lightning strokes.

11.2 'Touch' Voltage

Figure 11.2 Equivalent circuit of a person exposed to 'touch' voltage



Touch voltage is defined as the potential difference between a grounded point and a point on the earth's surface equal to a person's normal maximum horizontal reach. Fault current flowing into the earth via the grounded casing of the motor will develop a voltage drop across R_E, representing total ground system resistance. A person touching the faulty motor will be safe as long as his body resistance R_B and his contact resistance to the earth R_C are much bigger than R_E. The body resistance R_B varies greatly, even on the same person, between approx. 1,300 Ohm on a hot day in a humid atmosphere and approx. 3,000 Ohm in dry weather and with dry hands.

The maximum permissible voltage the body can be exposed to without immediate danger is 65 V. This corresponds under worst conditions to a maximum current of:

$$\frac{65}{1300} = 0.05\,A$$

These values of 65 V and 0.05 A are laid down in the German VDE regulations. In many countries, however, the voltage considered to be safe is 50 V.

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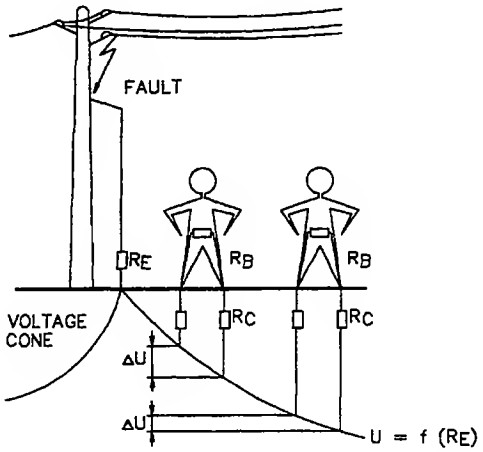
11.3 'Step' Voltage

Figure 11.3 Equivalent circuit of a person exposed to 'step' voltage



'Step' voltage is another hazardous condition caused by distributed voltage gradients. It is defined as the potential difference between two points on the earth's surface separated by a distance of a person's pace (about 1 m). The figure above illustrates such a condition. Again, fault current flowing into the earth via the grounded pylon will develop a voltage drop across R_E representing the total ground system resistance. Voltage appearing across portion ΔU will determine the magnitude to which the body will be exposed. Keeping the total ground system resistance low will reduce the value of ΔU for safety purposes.

11.4 Safety Precautions in High Voltage Equipment Rooms

The electrical equipment in the plant has to be protected in such a way that no one can touch any live parts. National as well as international safety codes have set up strict rules for all equipment, its insulation, wiring and earthing.

In electrical rooms, strictly accessible to trained electricians only, different regulations apply which allow for live parts to be exposed.

Special precautions therefore have to be taken in case of alterations and/or maintenance in such rooms. Under no circumstances should artisans of other trades (e.g. masons) be allowed to work in such rooms without the supervisions of an electrician. Temporary barricades may also be required to isolate work areas to prevent accidental contact with energised high voltage parts.

11.5 Preventive Maintenance

Each cement plant has a variety of mobile equipment which is temporarily connected to the power or light network by means of cord connections. Such flexible connections are subjected to abnormal wear; the electrical staff must, there- fore, pay special attention to proper maintenance and repair.

The grounding system has to be measured at regular intervals to assure its low resistance and to detect any faulty or corroded connections.

It is recommended to protect all plug sockets by differential current earth leakage breakers.

In areas where flammable or potentially explosive goods are stored or handled, the electrical installations have to be flame-proof or explosion-proof. Such equipment may be necessary within sections of the oil treatment plant or near natural gas installations.

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11.6 Precautions against Electric Fires

An American factory insurance company has compiled statistics showing that wiring alone is responsible for almost half the electric fires, and that over half the losses could have been avoided by correcting minor wiring defects.

Sufficient attention is normally paid to the proper protection of oil filled transformers, thus limiting damage because of fire.

Wiring, however, covers large areas of every structure; it is attached to or part of all types of machinery, and is exposed to almost every conceivable environment; heat, cold, dust, moisture, oil, vibration, corrosive liquids and gases. Cables are grouped in large steel enclosures, floor trenches, junction boxes, pits, manholes, and tunnels. Fires in such places are often well-advanced before being discovered. The limited accessibility impedes the application of extinguishing agents.

The almost universal use of PVC-covered and sheathed cables can increase the damage of such fires because of the extensive development of fumes of hydrochloric gases which combine to hydrochloric acid in the presence of moisture.

As a protection against such damages and prevention of plant interruption caused by wiring fires, the following means should be considered:

- installation of sprinklers
- division of long cable tunnels or trenches into partitions
- proper insulation of hot stream or oil pipes using the same tunnel
- proper protection against sparks during welding operations
- sufficient separation and ventilation of cables subjected to high loads
- marking of escape routes in trenches

11.7 Maintenance of Temporary Installations during Construction and Erection

Special attention has to be paid to the proper handling and maintenance of temporary installations. Cables suspended on steel structures are dangerous; if they tear they can set alive the whole structure. Mobile boom cranes, which can quickly change their working location, are frequently the cause of electrical accidents when they touch overhead lines or tear down suspended cables.

11.8 First Aid

First aid instruction posters are commonly available and prescribed in electrical rooms.

Since quick and correct action in case of an electrical accident is of vital importance, first aid training should take place at regular intervals.

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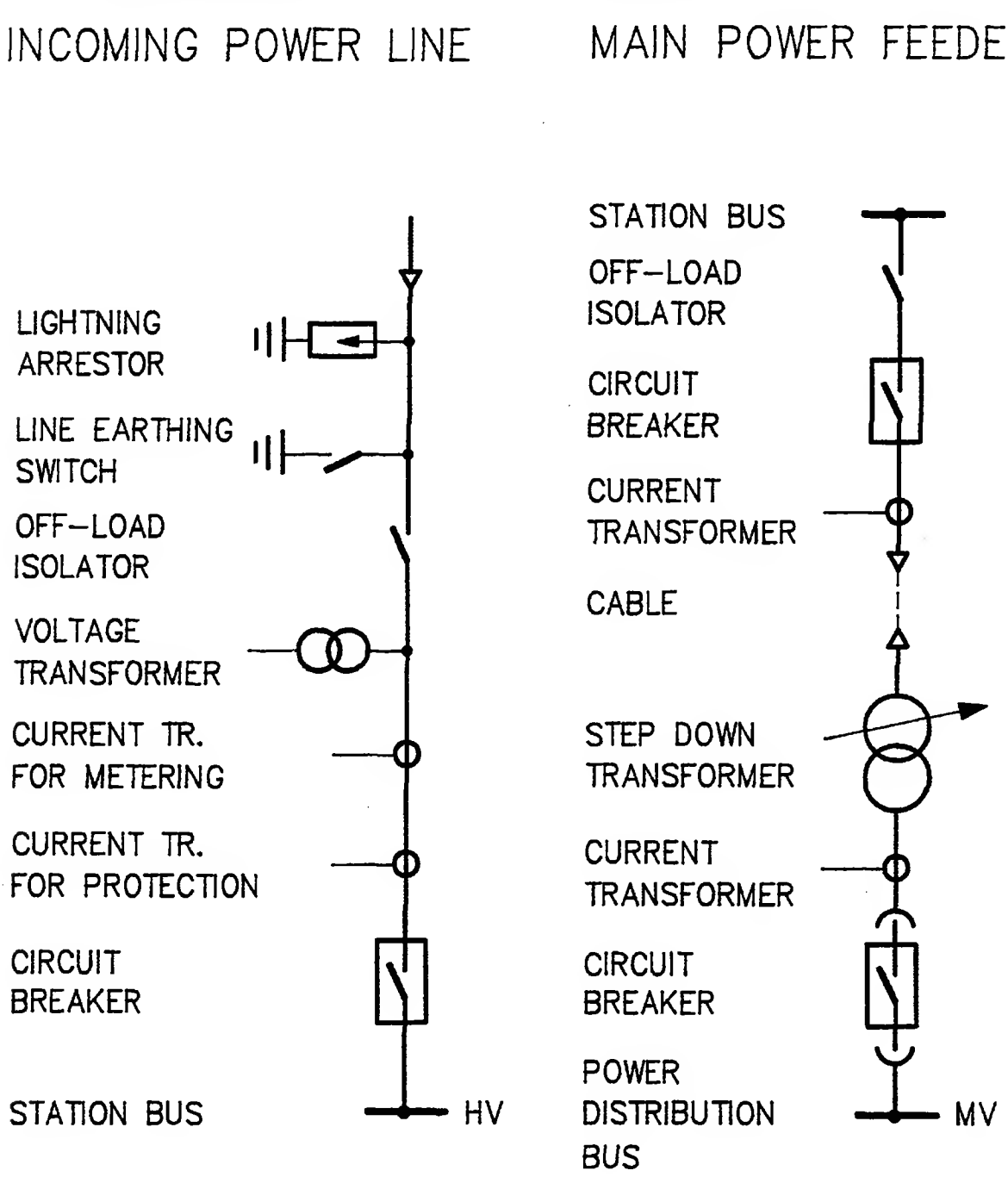
12. ANNEXES

12.1 Power Supply
Typical elements of a high voltage transformer station



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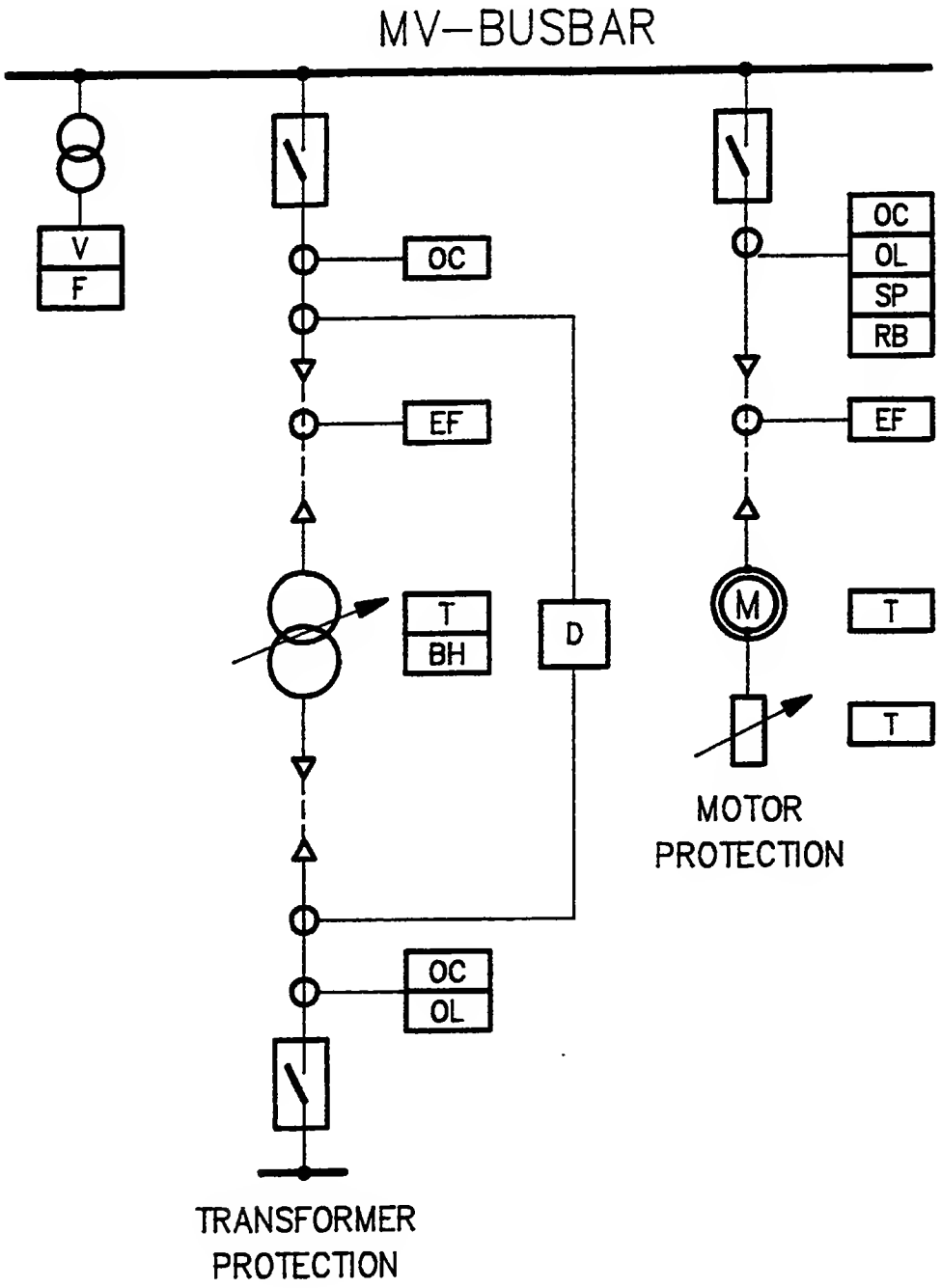
12.2 Power Distribution
Protective relays



LEGEND:

V	: VOLTAGE	RB	: ROTOR BLOCKAGE
F	: FREQUENCY	EF	: EARTH FAULT
OC	: OVERCURRENT	D	: DIFFERENTIAL PROTECTION
OL	: OVERLOAD	T	: TEMPERATURE
SP	: SINGLE PHASE	BH	: BUCHHOLZ

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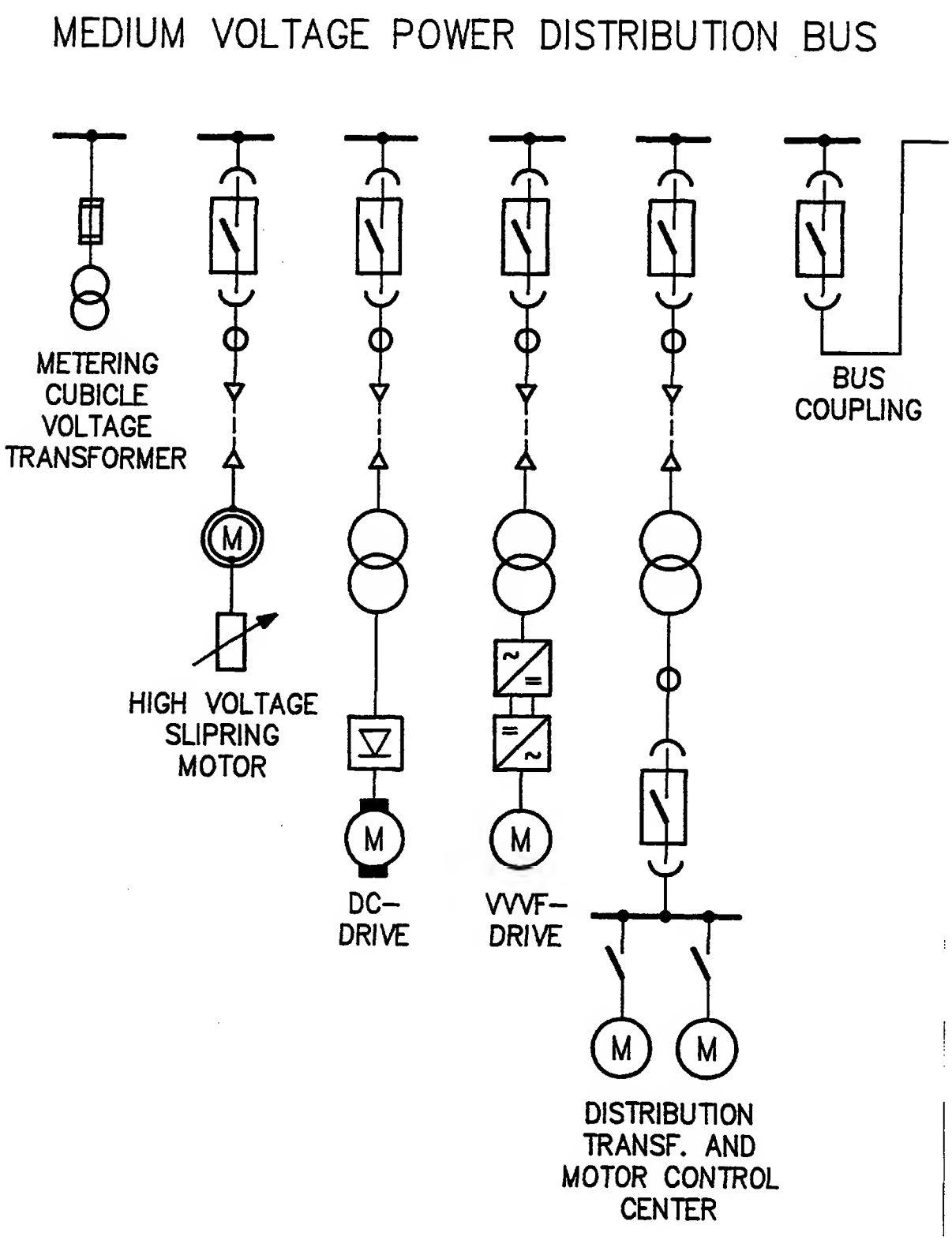
12.3 Power Distribution
Typical elements of a medium voltage power distribution



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Motors and Drive Systems

1.) Types of motors
2.) Protection forms of motors and cooling
3.) Characteristics of motors (torque)
4.) Maintenance aspects
5.) Variable speed drive systems (mechanical / electrical)
6.) Drive system efficiency

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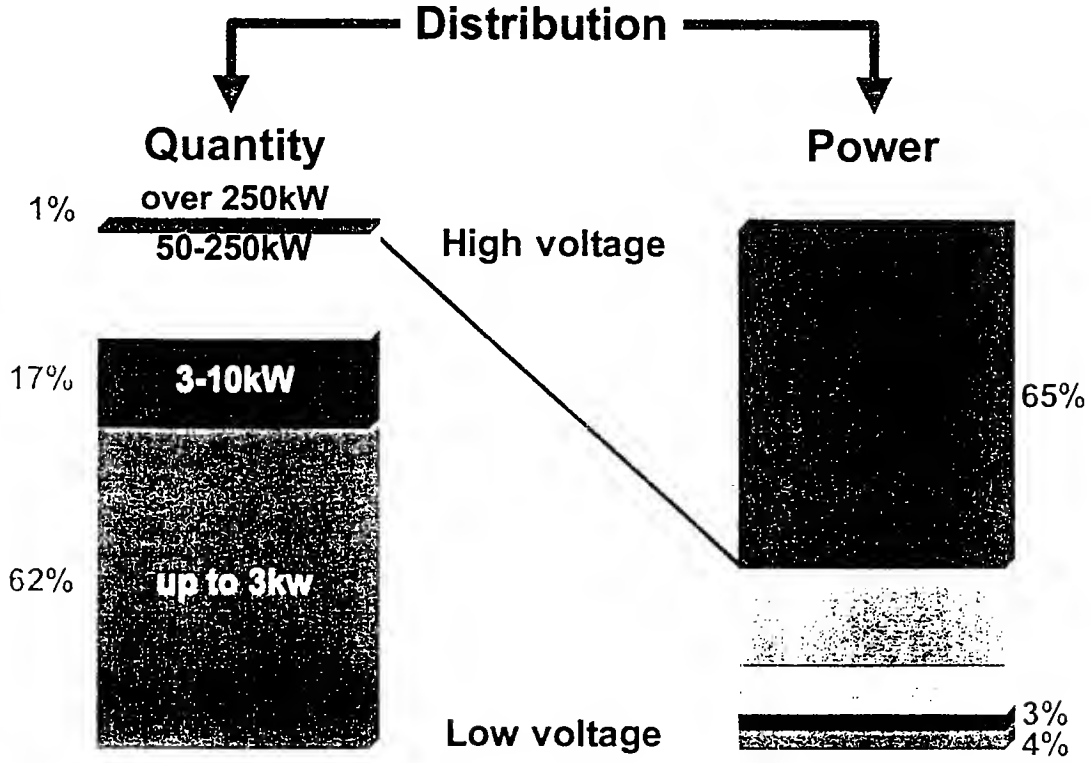
Electrical Drives



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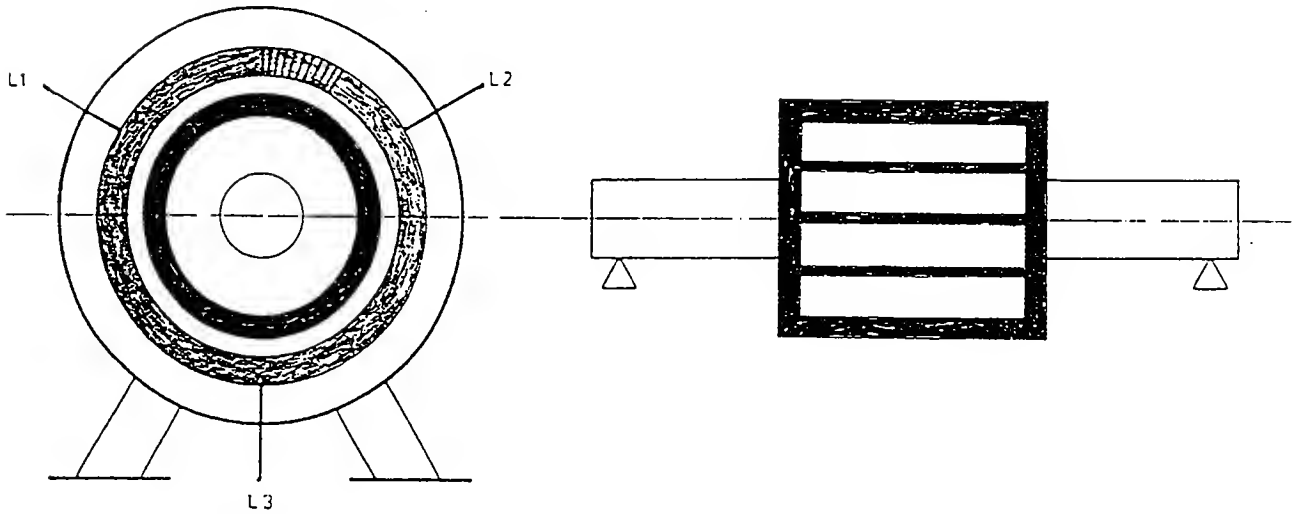
SQUIRREL CAGE MOTOR

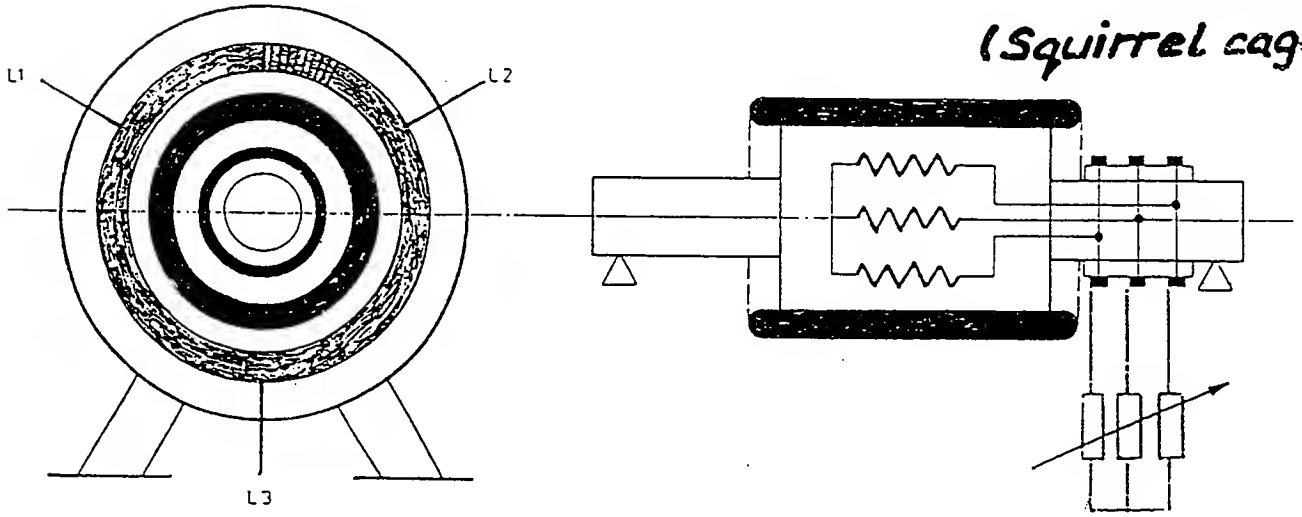


SLIPRING MOTOR
(HOUND ROTOR)

SLIPRING : STARTING ONLY

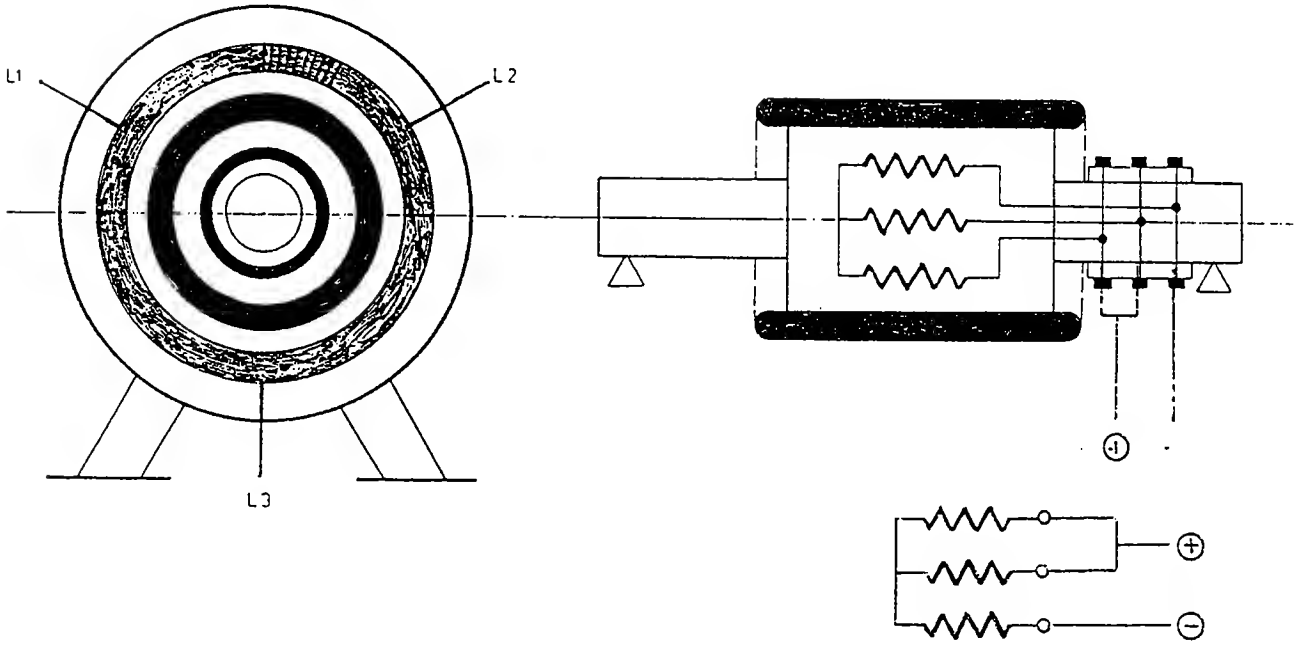
OPERATION : SHORT CIRCUIT

(Squirrel cage)

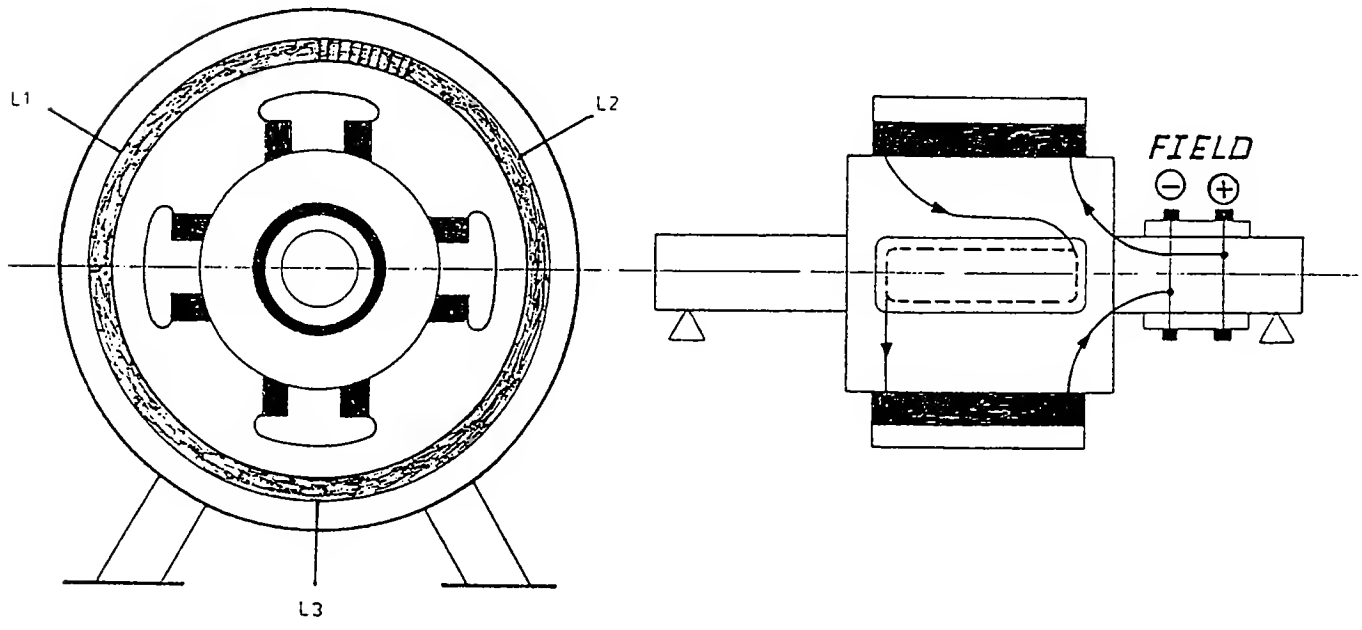


SLIPRING MOTOR
(WOUND ROTOR)

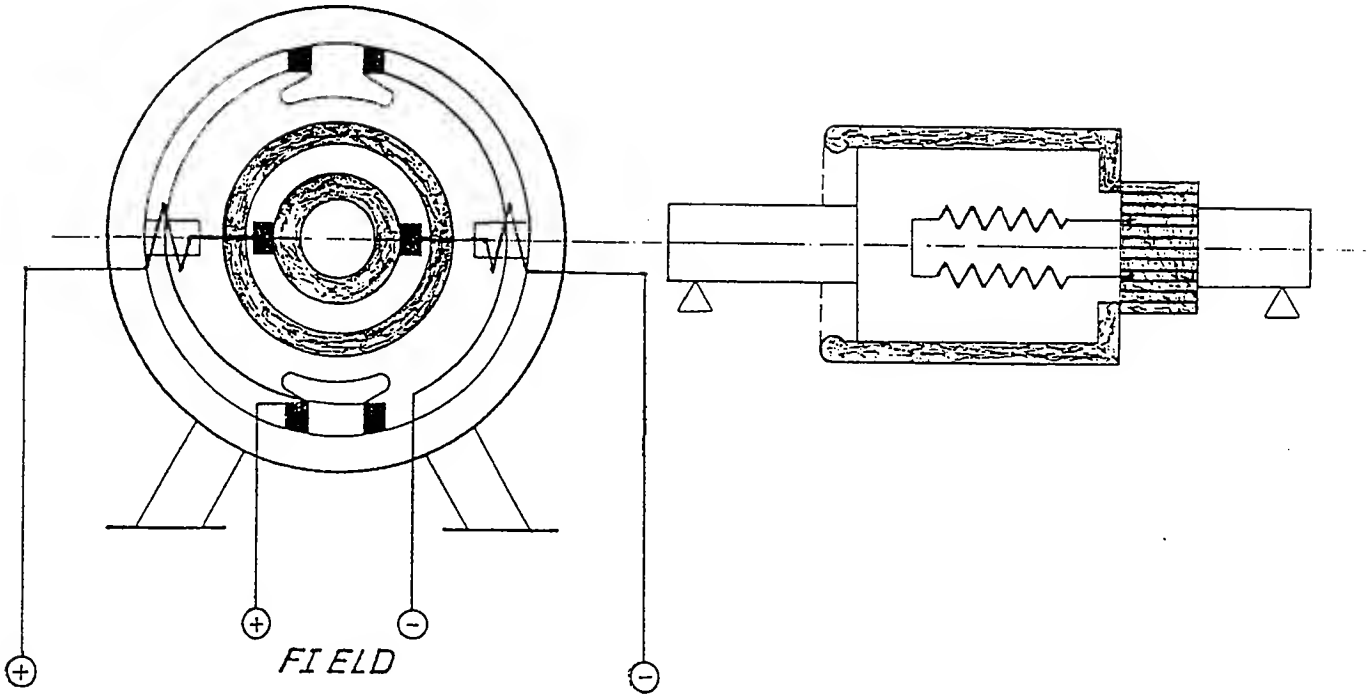
SYNCH. INDUCTION



SYNCHRONOUS MOTOR



DC - MOTOR



Motors / Drives **"Cement Seminar 2000"**

- Motors > 250 - 300kW → MV-motors (i.e. 6kV, 3kV)
- Motors < 250kW → LV-motors (i.e. 400V, 500V)

Two types of motors:

→ Constant speed motors
→ Variable speed motors

Motors / Drives **"Cement Seminar 2000"**

Constant speed motors → AC squirrel cage motor (most common drive in cement plant)

- To be considered:
  - Inrush current (5-7 times full load current)
  - Starting time → voltage drop on power distribution (< 10-15%)

↓ If problem

→ Slipping motor with rotor starter

Motors / Drives **"Cement Seminar 2000"**

Constant speed motors → AC synchronous motor (very seldom in cement plant)

- To be considered:
  - low starting torque
  - May be used for power factor compensation
  - Mostly used with converters

Motors / Drives **"Cement Seminar 2000"**

Variable speed motors:

→ AC-slipring motor with variable resistor or slip-recovery system for i.e. large fans
Note: speed depends of the load

→ DC-motor with thyristor converter for i.e. kiln drive
Note: speed is independent of the load

→ AC-squirrel cage motor with static frequency converter control for i.e. fans
Note: speed is independent of the load

$$\text{synchronous speed} = \frac{\text{frequency} \times 60}{\text{no of poles on motor}} \quad \text{(rpm)}$$

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Motors / Drives **"Cement Seminar 2000"**

Maintenance methods:

- Preventive maintenance based on vibration measurement
- Preventive maintenance based on thermography
- Preventive maintenance based on periodical inspection
- Corrective maintenance based on break-down

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Motors / Drives **"Cement Seminar 2000"**

Motors with brushes:

→ Most maintenance demanding equipment ↣ carbon dust
- Quality of the sliding contact dependent on atmosphere (humidity)
- Current density commutation (current density is directly proportional to temperature) ↣ leading to evaporation of the moisture in the air

→ **Patina or commutator skin**

- Normal brush wear: ca. 3-5mm per 1000 hours; to be frequently measured / recorded

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Motors / Drives **"Cement Seminar 2000"**

Brush wear:

→ Carbon dust has to be removed because conducts electricity; to be cleaned:
- → By vacuum cleaner
- → Slipring hub and brush assembly to be cleaned with a rag

Job to be done several times per month, depending on running hours

→ Brush has to move freely in brush holder
- → If not: increased brush wear and sparks will be created

→ Uneven sliprings have to be polished or skimmed off on a lathe

→ Commutator / slipring housing to be painted white: carbon dust, easily noticeable

Motors / Drives **"Cement Seminar 2000"**

Reason for sparks leading to excessive brush wear:

- Stuck brushes
- Stuck pressure finger
- Worn brushes
- Oil on contact surface
- Scratched contact surface
- Vibrations (max. 2-3 mm/sec.)
- Brush position (adjustment)
- Dust (abrasive dust)
- Carbon brush quality
- Too good contact → increased current → sparking due to overload
- Equipped with thyristor converter: break down of thyristor, diode fault in supply circuit, fault in control equipment or a fuse

Motors / Drives **"Cement Seminar 2000"**

One can distinguish between 4 colors of sparks on the commutator:

① Small violet sparks are almost harmless.

② Red sparks arise with overload and result in brush wear.

③ Blue sparks, shining with glowing particles, usually indicate inferior communtation. Both brushes and commutator will be damaged.

④ Green sparks, always together with glowing particles, indicate serious defects, causing rapid destruction of brushes and commutator.

Rotor-starters:

- To be checked on regular intervals
  - ➜ Contact tips to be visually inspected
  - ➜ Check insulation oil (can be regenerated)
  - ➜ Monitor temperature of oil-cooled or liquid filled starters
  - ➜ Liquid filled starters: level and concentration of electrolyte to be monitored

Testing insulation resistance on motor windings:

- To be done for new motors, long idle period, newly rewounded motors
  - ➜LV-motors: 500V-megger
  - ➜MV-motors: 1st: 500V-megger, if o.k.
    2nd: 1000-2500V-megger

$$\text{measured insulation resistance} > \frac{20 \times \text{operating voltage (V)}}{1000 \times 2 \times \text{output power (kW)}} = \text{M}\Omega$$

(Formula valid for 25°C, 500V-megger: one minute; if temperature is not to be corrected)

Condition monitoring of motor / drive system:

- Thermography (see chapter power distribution)
  - ➜ Used for motor / terminal box / frequency converter panels
- Vibration control
  - ➜ Shock pulse meter to check operating condition of ball and roller bearings
  - ➜ Used for preventive maintenance and long-term planning

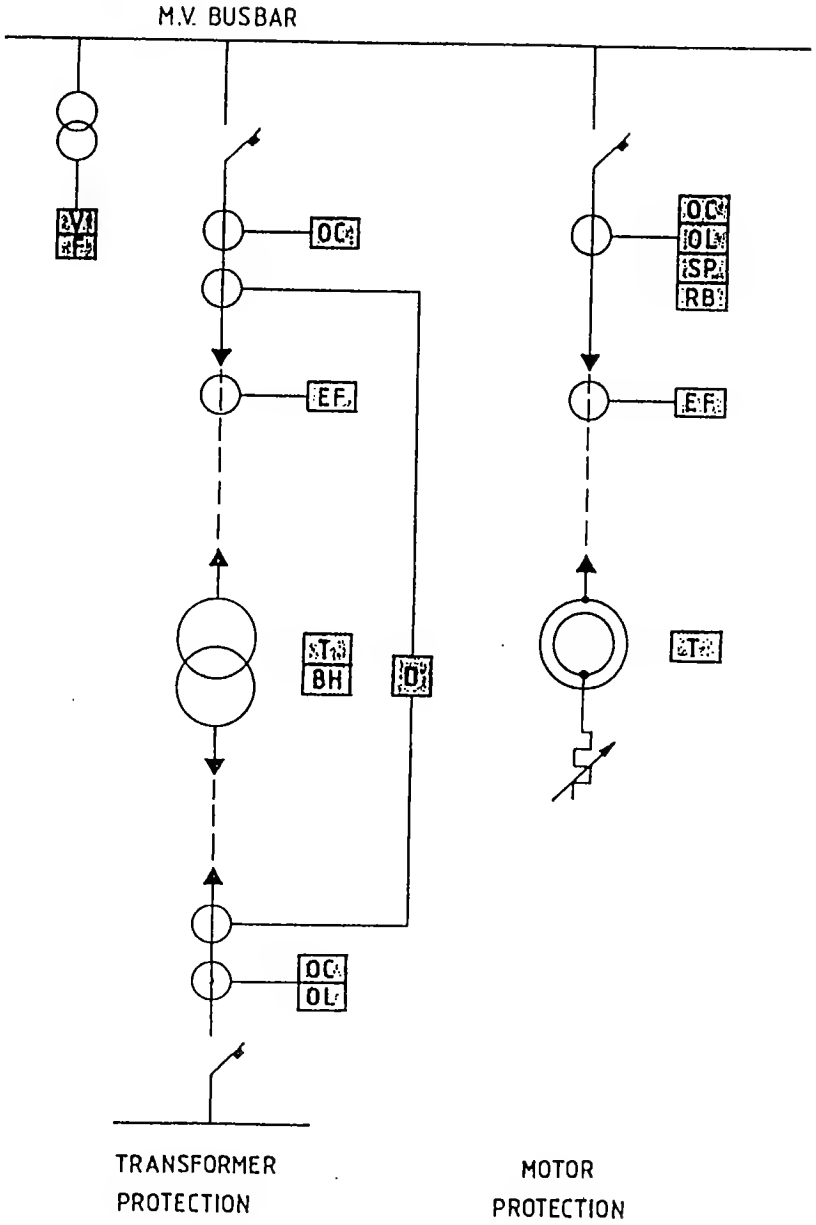
Low shock valve: good condition
High shock valve: sign bearing faults, bad installation, insufficient lubrication

- Cooling air installation for motors
  - ➜ Filters to be cleaned
  - ➜ Check if sufficient cooling air is available

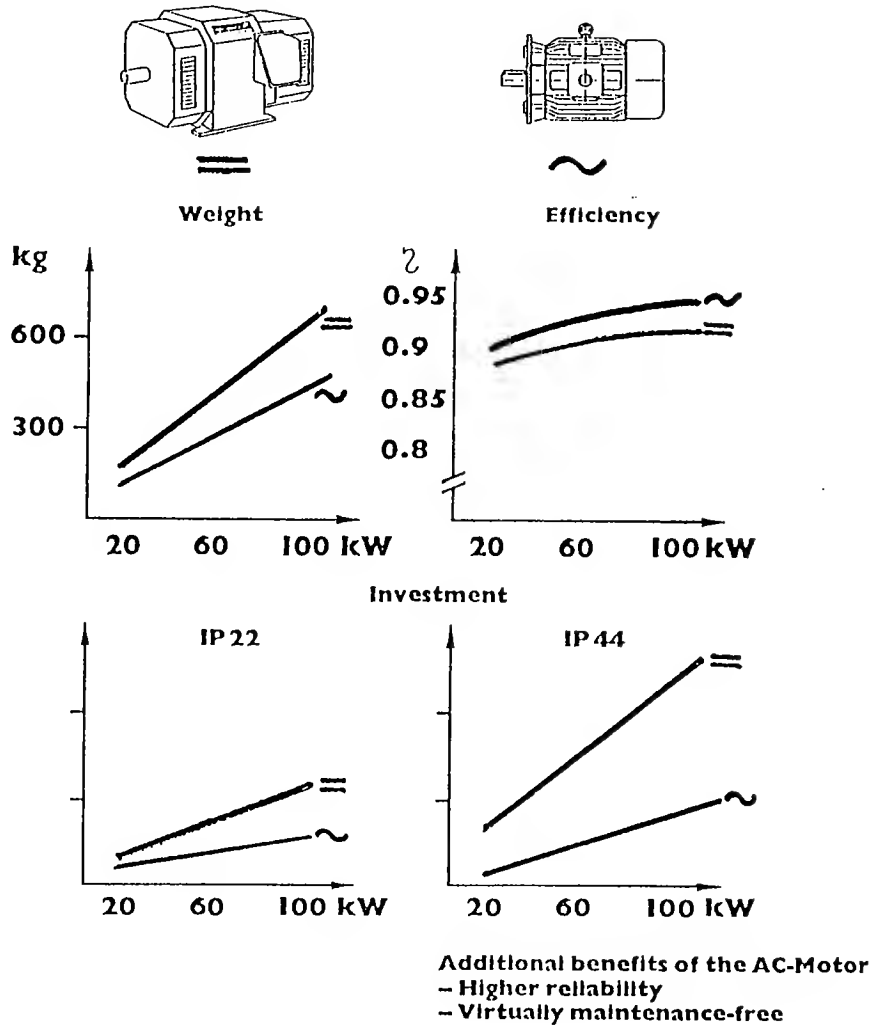
Motor protection:

- Today's standard are multiple microprocessor based protection relays Following faults / conditions are detected
  - *Thermal overload*
  - *Time / current characteristics*
  - *Start-up supervision*
  - *High-set over current*
  - *Incorrect phase sequence*
  - *Under current*
  - *Cumulative start-up time counter*
  - *Self-supervision*
- Motors > 100kW requires thermal sensing / protection device for:
  - *Stator*
  - *Winding (resistance thermometers, thermistors: semiconductor temperature sensor) → cannot be replaced*
  - *Sleeve / roller bearings (contact or resistance thermometers)*

PROTECTIVE RELAYS



Comparison DC/AC Motor

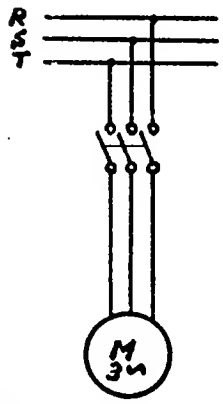


Additional benefits of the AC-Motor
- Higher reliability
- Virtually maintenance-free

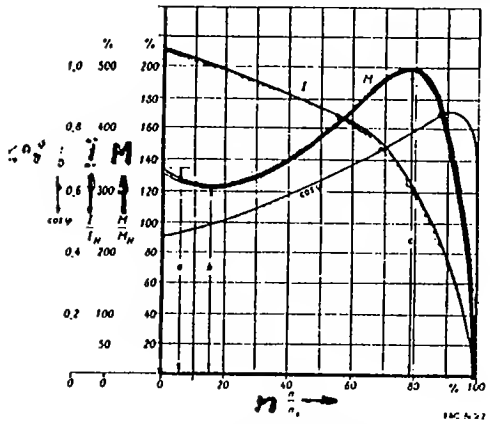
b) SQUIRREL CAGE MOTOR (INDUCTION MOTOR)

Construction

- The squirrel cage motor is in its construction the simplest motor used in the cement industrie
- The main feature is a rotor without external connections (no slip rings, no brushes)
- Its two bearings are the only parts exposed to wear and tear.
- It is economical in price

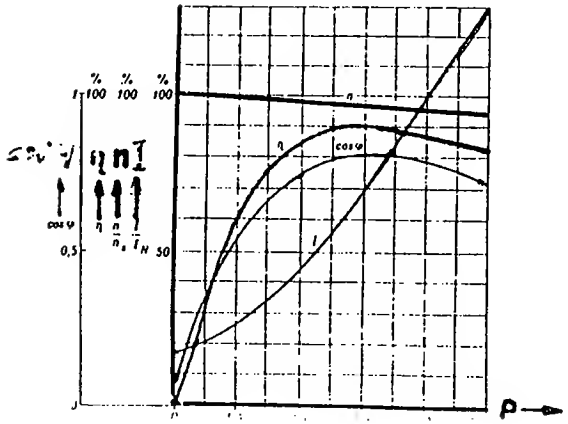


TYPICAL STARTING CHARACTERISTIC



a Starting torque
b Sattle "
c Break down "
i Current
m Torque
n Speed
cosφ Power factor
η Efficiency
p Load

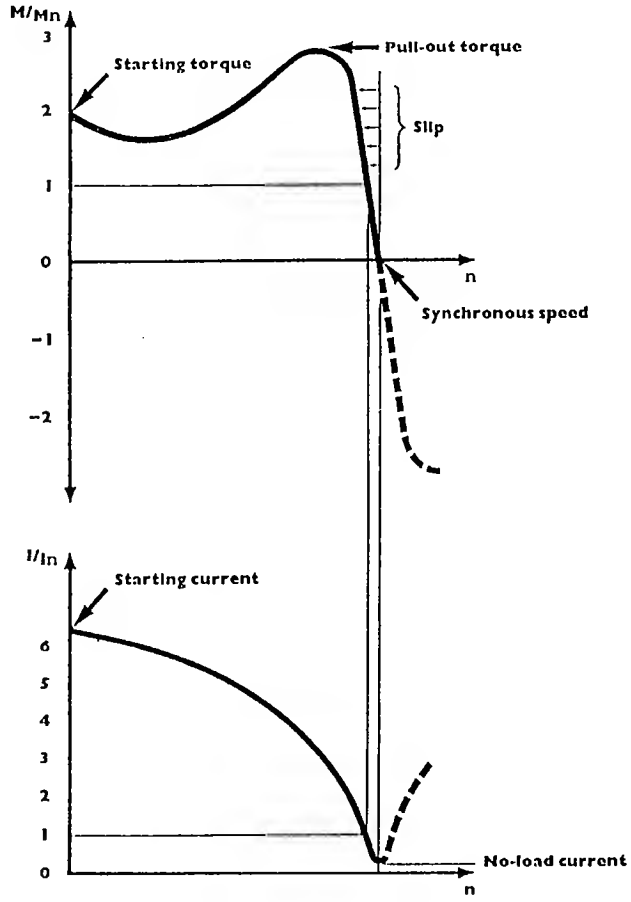
OPERATING CHARACTERISTIC

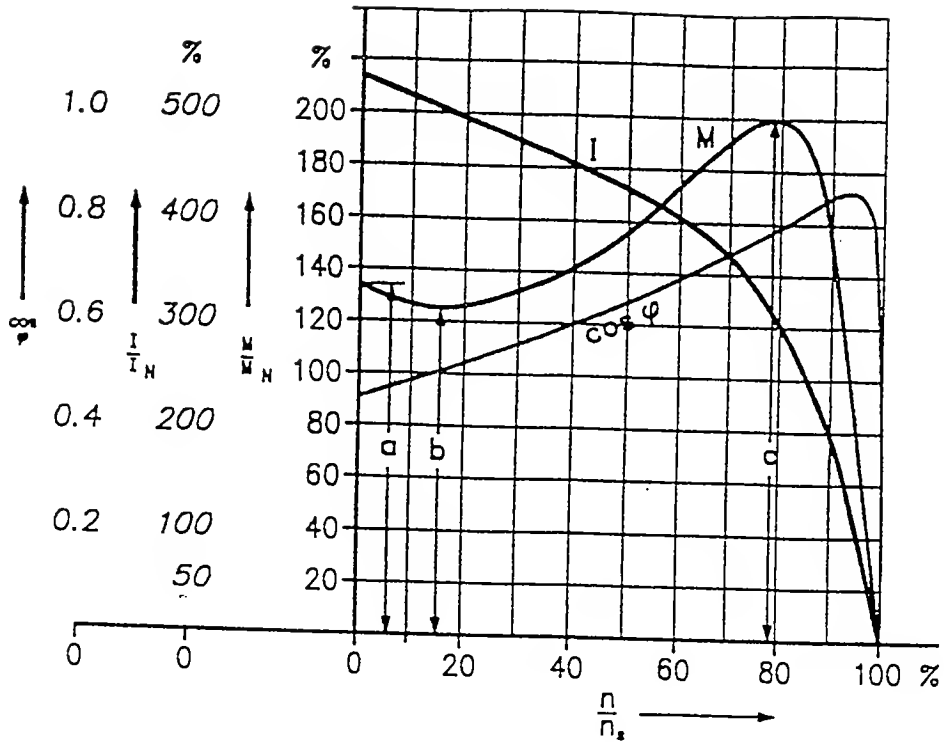


PROPERITIES :

- High starting current 3,5 to 7 times full load
- Relatively constant speed
- Torque changes with the square of the voltage

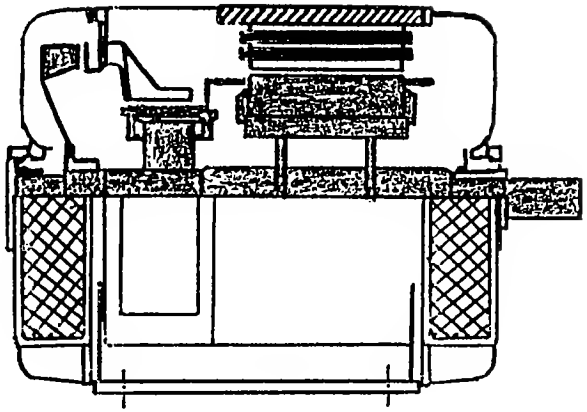
Torque Curves

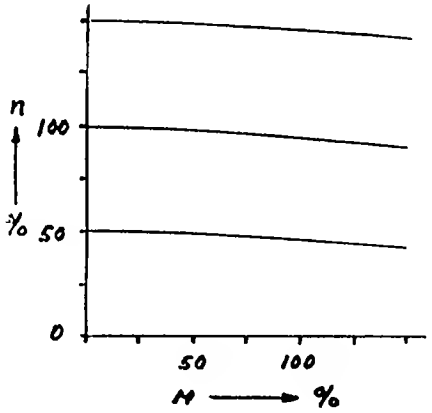


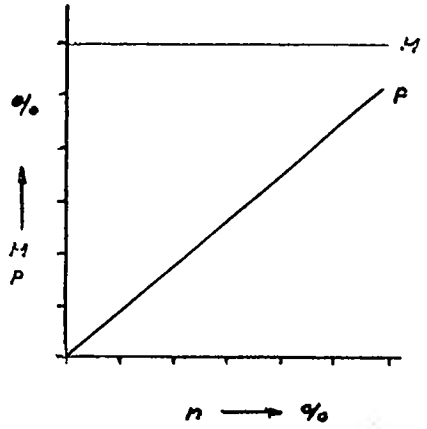


a :	starting torque	b :	saddle torque
c :	break-down torque	I :	current
M :	torque	n :	speed
cosφ :	power factor	η :	efficiency
P :	load		

Synchronous Motor





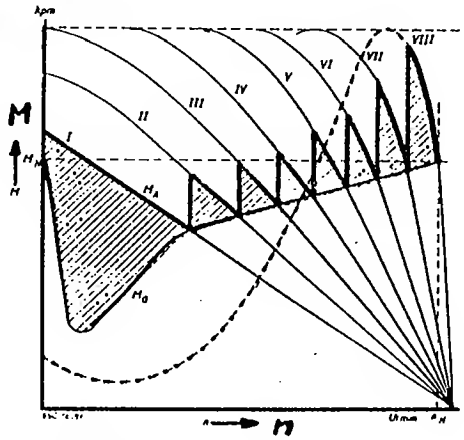


c) SLIP RING MOTOR

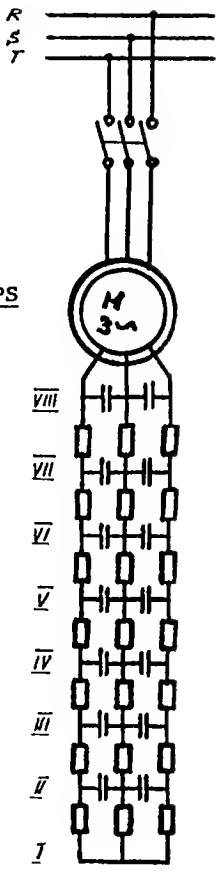
Construction

- The slip ring motor is like the squirrel cage motor an induction motor
- Its rotor windings are brought out to slip rings which allows to control the starting torque and current within a wide range.

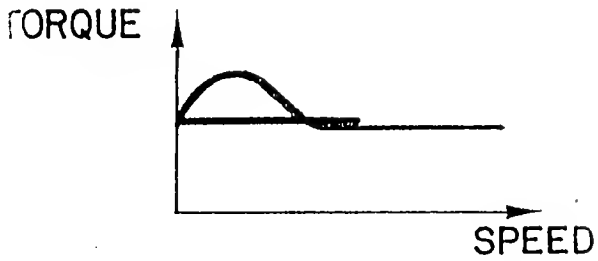
START OF A SLIP RING MOTOR WITHIN EIGHT STEPS



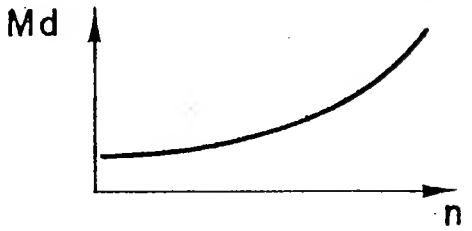
MN Nominal torque
MA Torque characteristics during start up
MG Torque of the load



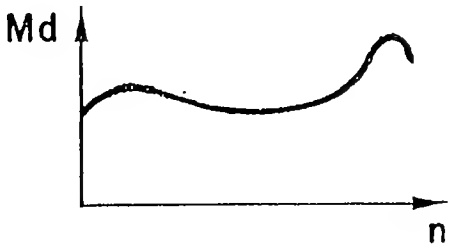
TYPICAL TORQUE / SPEED CURVES



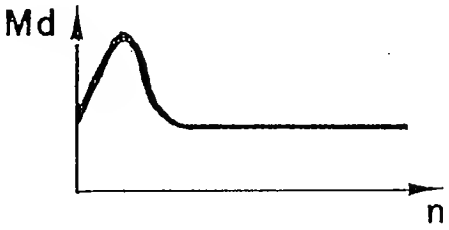
CONVEYOR BELT



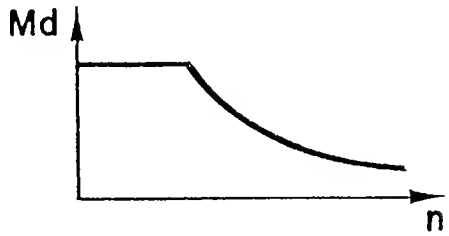
FAN



TUBE MILL

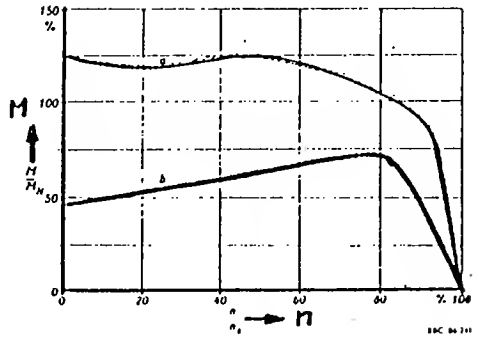


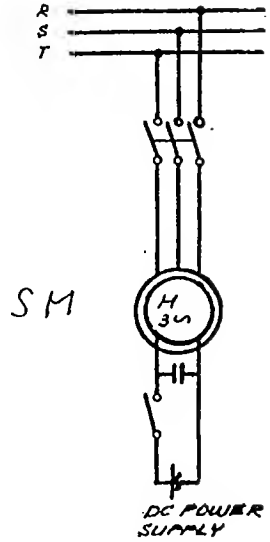
KILN



WINDER

STARTING CHARACTERISTIC



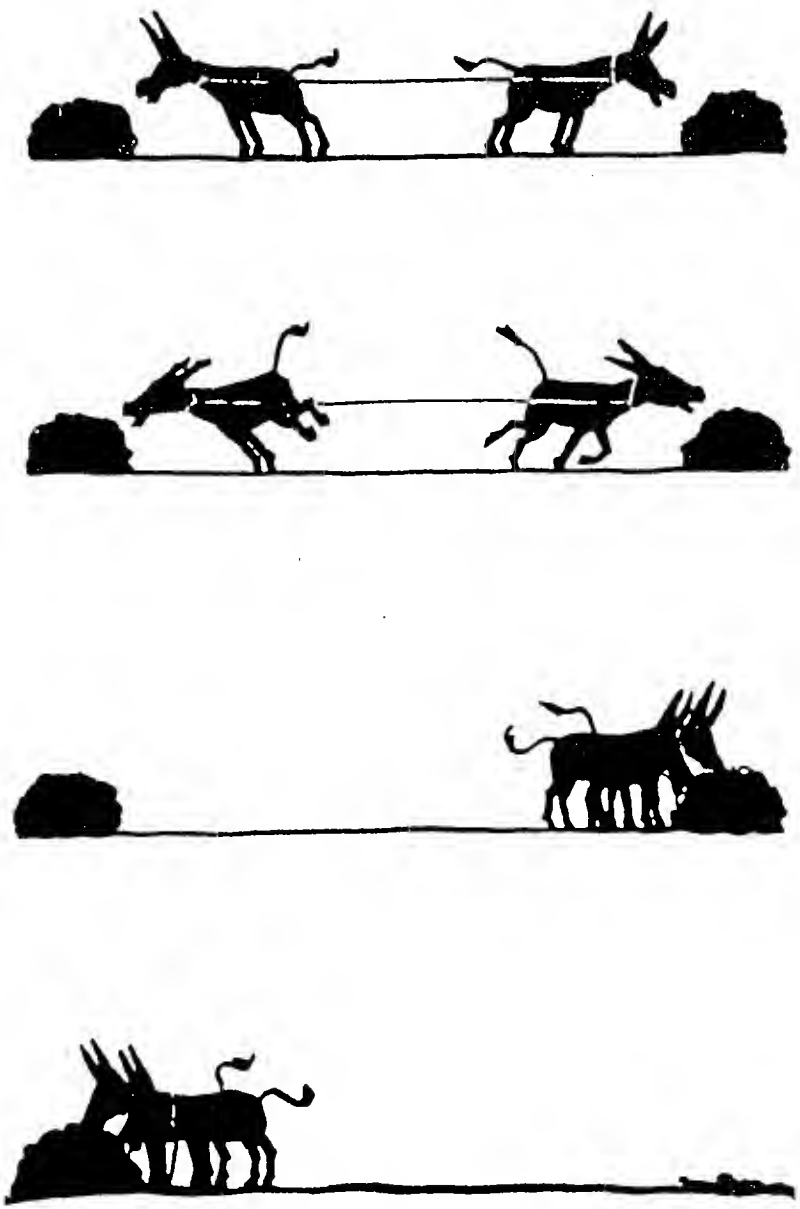


M Torque

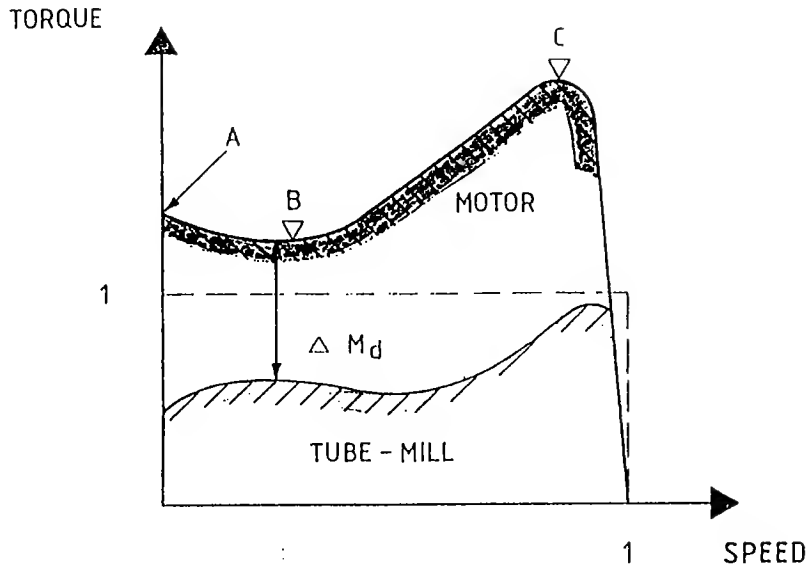
n Speed

a) Motor with massiv poles and starting winding

b) Motor with lami - nated poles and starting winding



TORQUE / SPEED CHARACTERISTIC



A : STARTING TORQUE

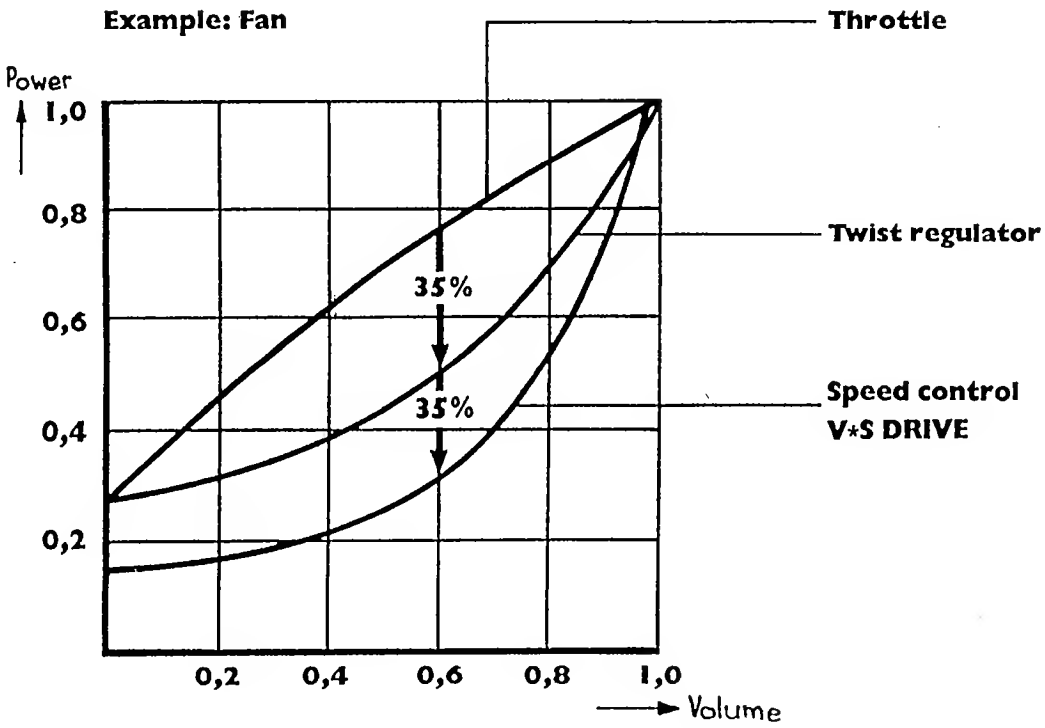
B : SADDLE TORQUE

C : BREAK DOWN TORQUE

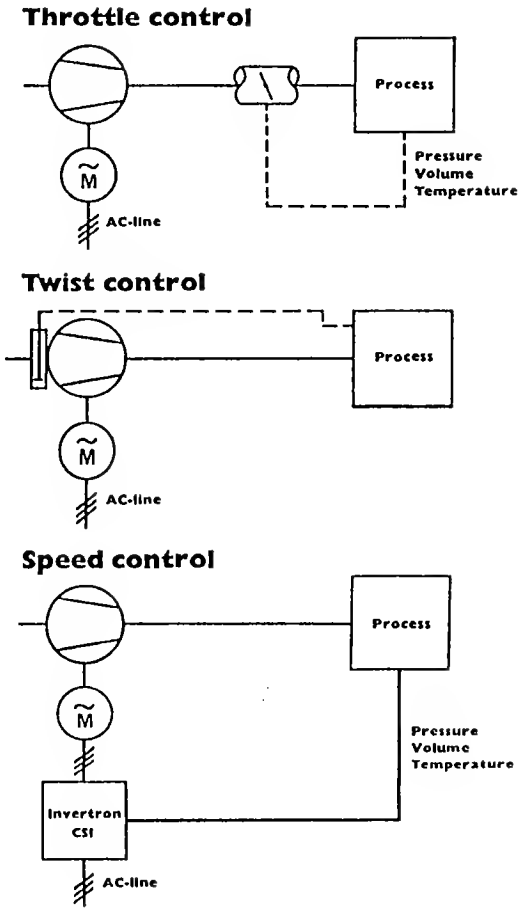
△ M_d : TORQUE AVAILABLE FOR ACCELERATION

Application Examples

Helps to save energy

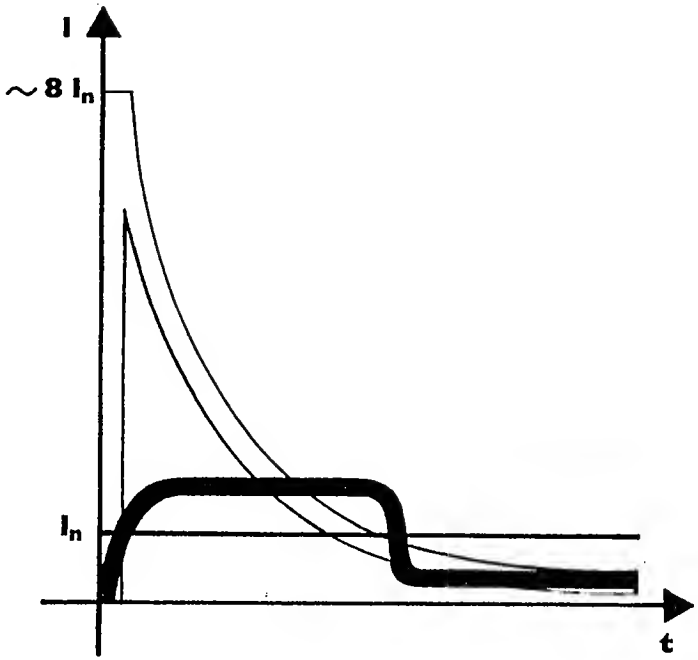


Ventilator



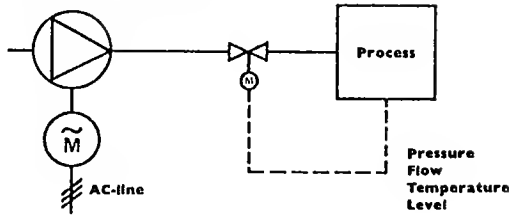
Application Examples

Reduces Starting Current
Reduces Mechanical Wear

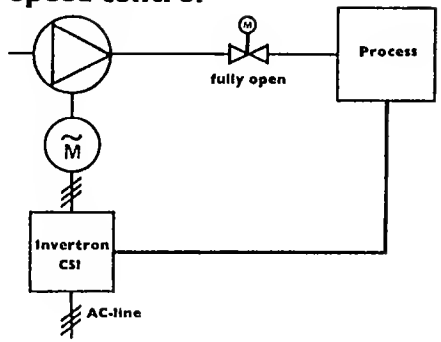


Pumps

Throttle control

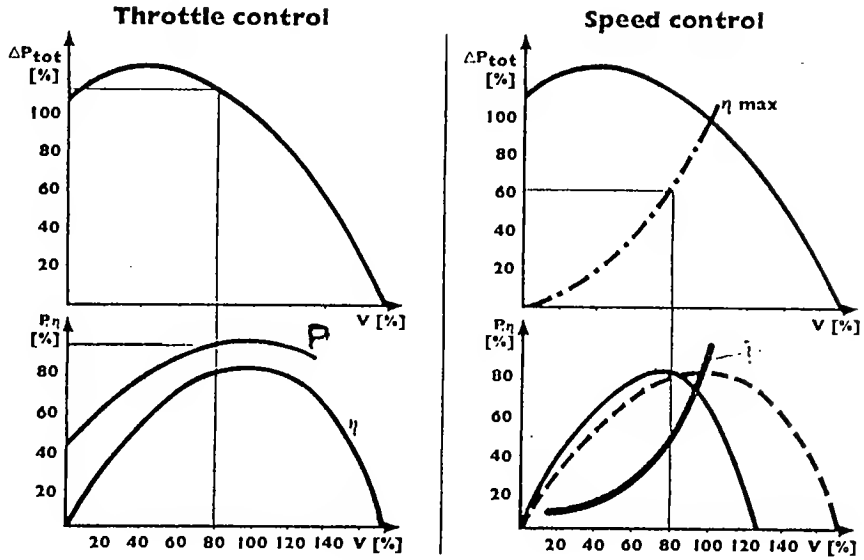


Speed control



Application Examples

Ventilator



$$P = \frac{V \cdot \Delta P_{tot}}{\eta}$$

Example: Volume of flow: 80 %, Drive power: 100 kW, Running hours: 8000, kWh-cost: £ 0,042

$$P_{80} = \frac{P_{100} \cdot 0{,}8 \cdot 1{,}12}{0{,}94} = P_{100} \cdot 0{,}95$$

$$B_K = 95 \cdot 8000 \cdot 0{,}042 = £\ 31\,920$$

$$P_{80} = \frac{P_{100} \cdot 0{,}8 \cdot 0{,}6}{0{,}94 \cdot 0{,}95} = P_{100} \cdot 0{,}53$$

$$B_K = 53 \cdot 8000 \cdot 0{,}042 = £\ 17808$$

Savings on energy costs:
14 112 £/year

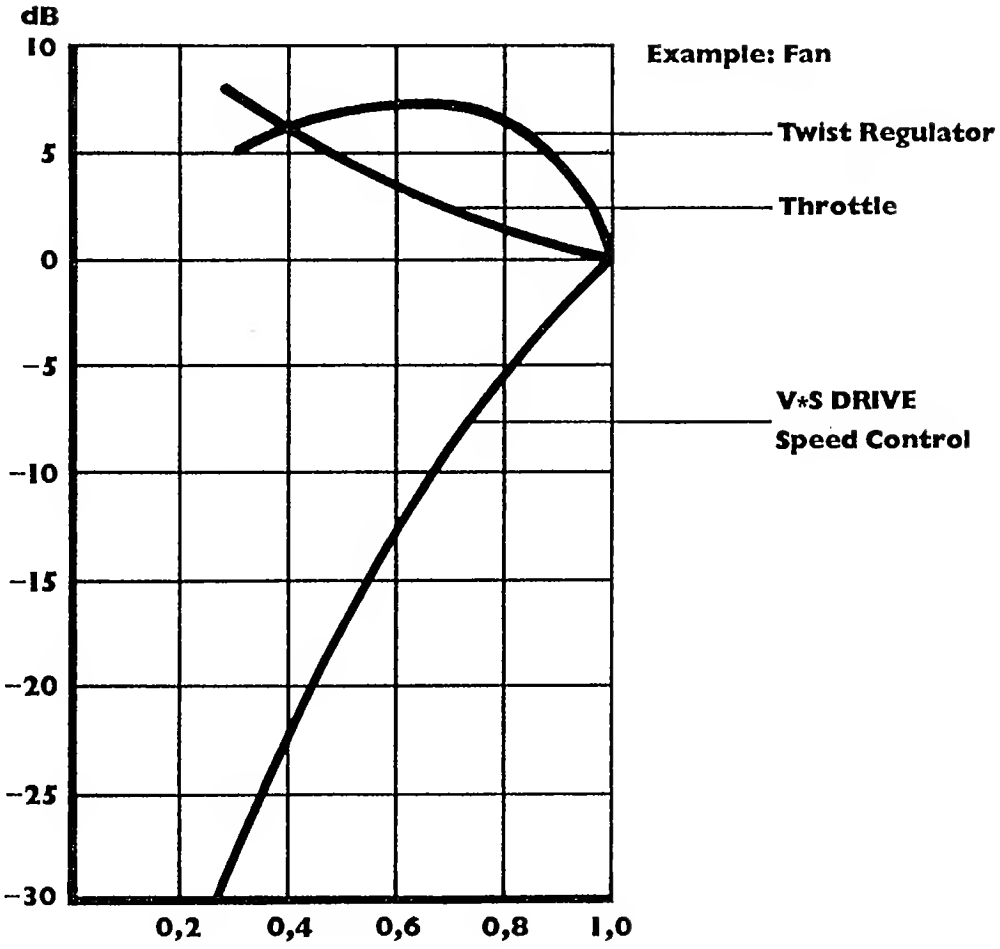
Outstanding Advantages of Variable Speed Drives

- Optimal process control
- Reduced stress on machines and supply system during starting
- Better utilization of the primary energy owing to the higher total drive system efficiency

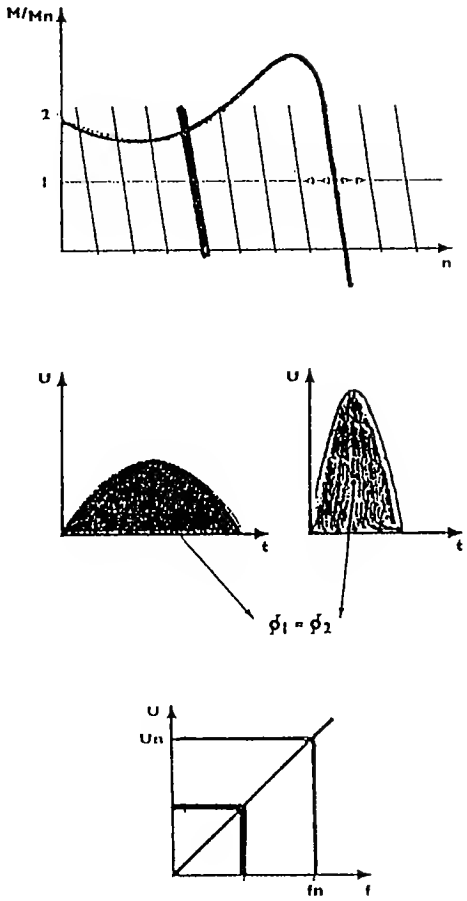
Drive Specification

- Robust
- Ease of maintenance
- Reliability
- Efficiency
- Investment cost

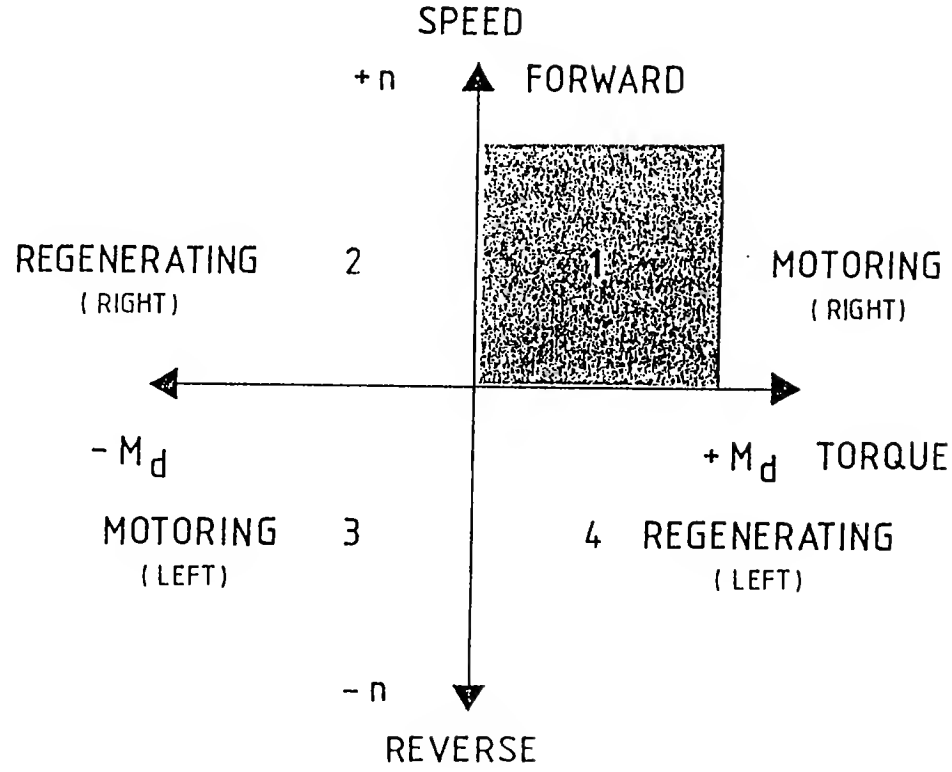
Reduces Noise



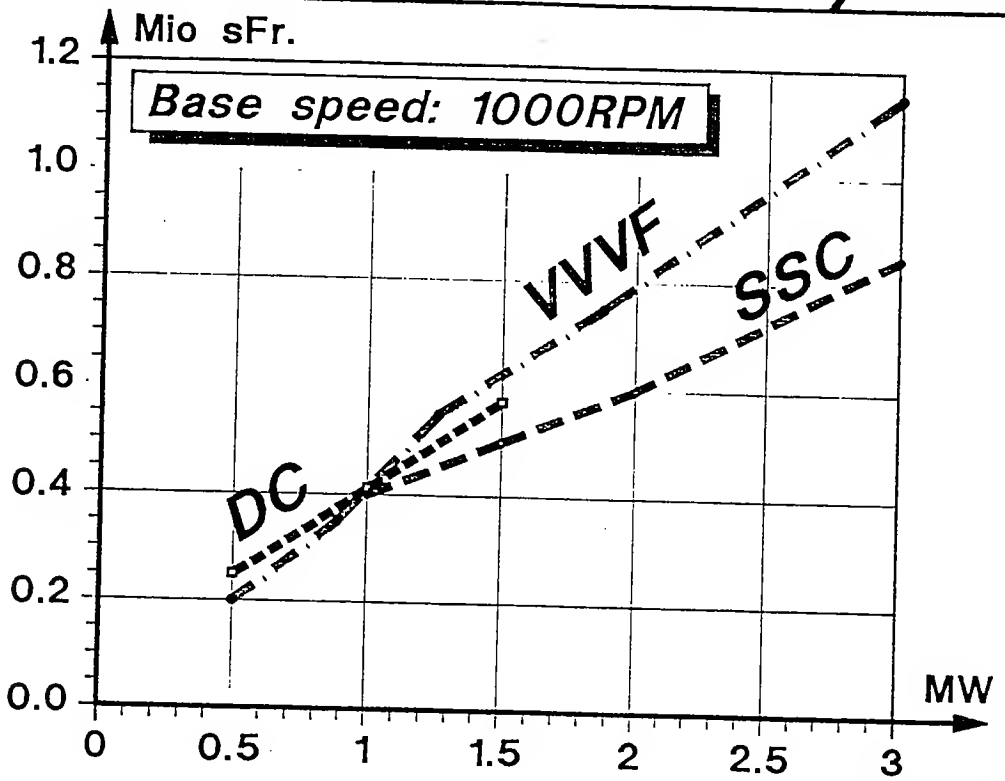
Torque curves of an AC-Motor at variable frequencies



SPEED / TORQUE
(4 QUADRANTS)

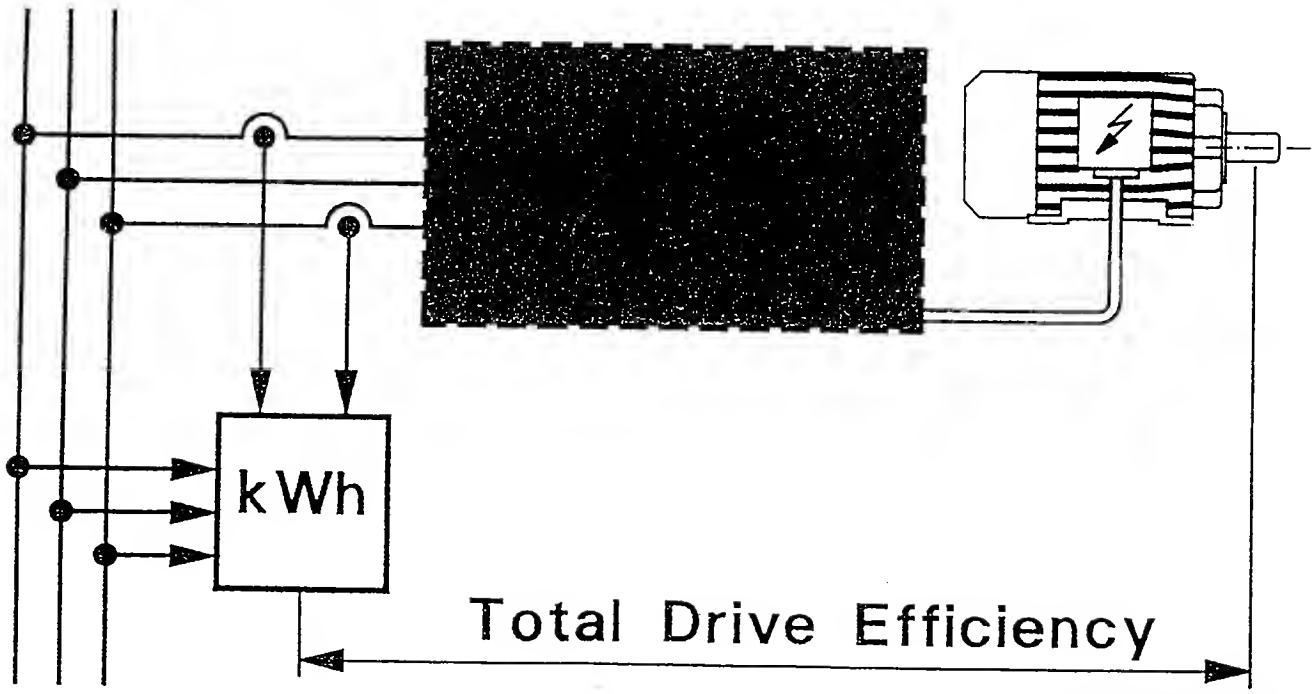


Budget Prices for Variable Speed Drive Systems



E410007 "HOLDERBANK"

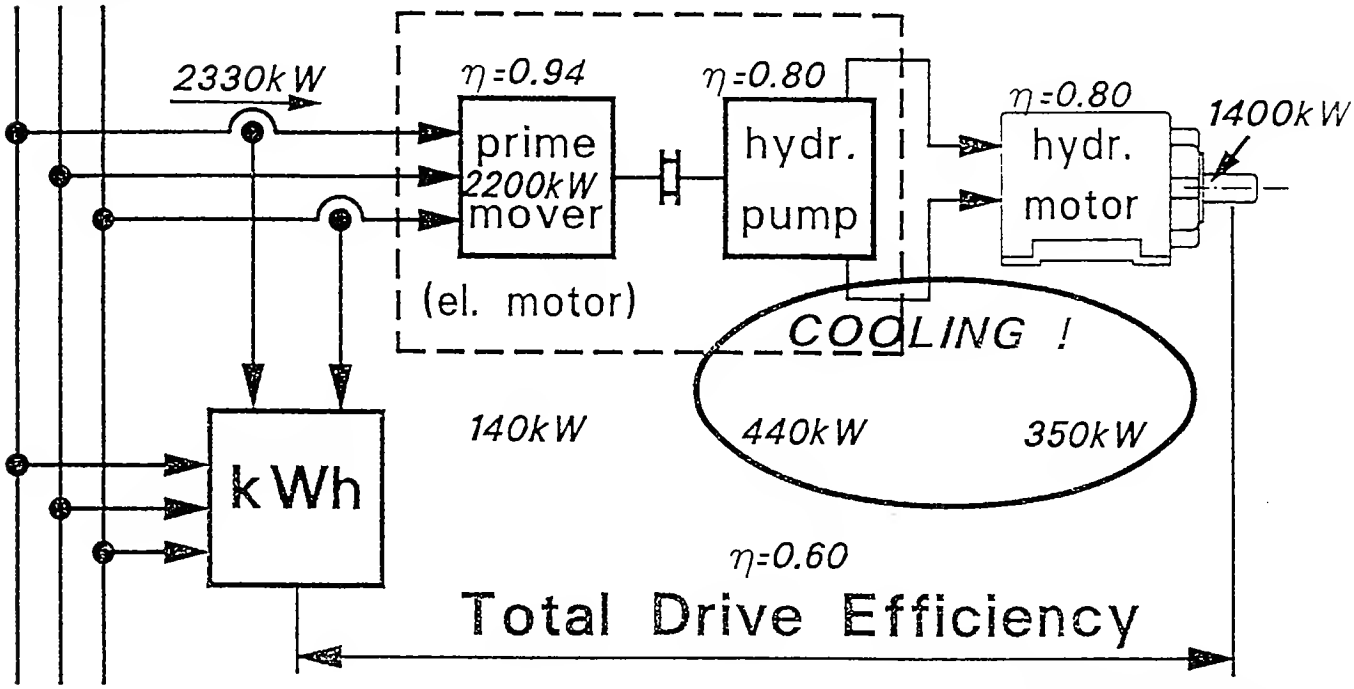
DEFINITION OF TOTAL DRIVE SYSTEM EFFICIENCY



"HOLDERBANK"

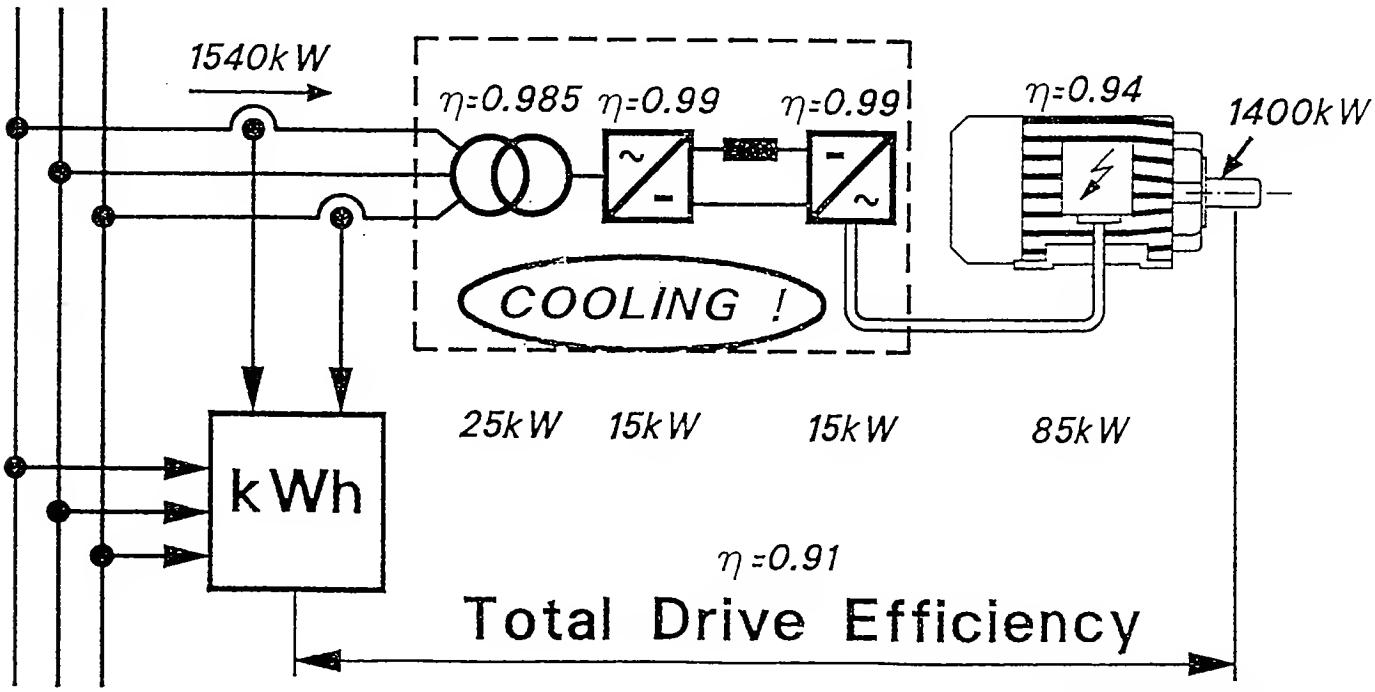
E49593-1 21.1.92 ip/Rü

MAIN COMPONENTS OF A HYDROSTATIC VARIABLE SPEED DRIVE SYSTEM



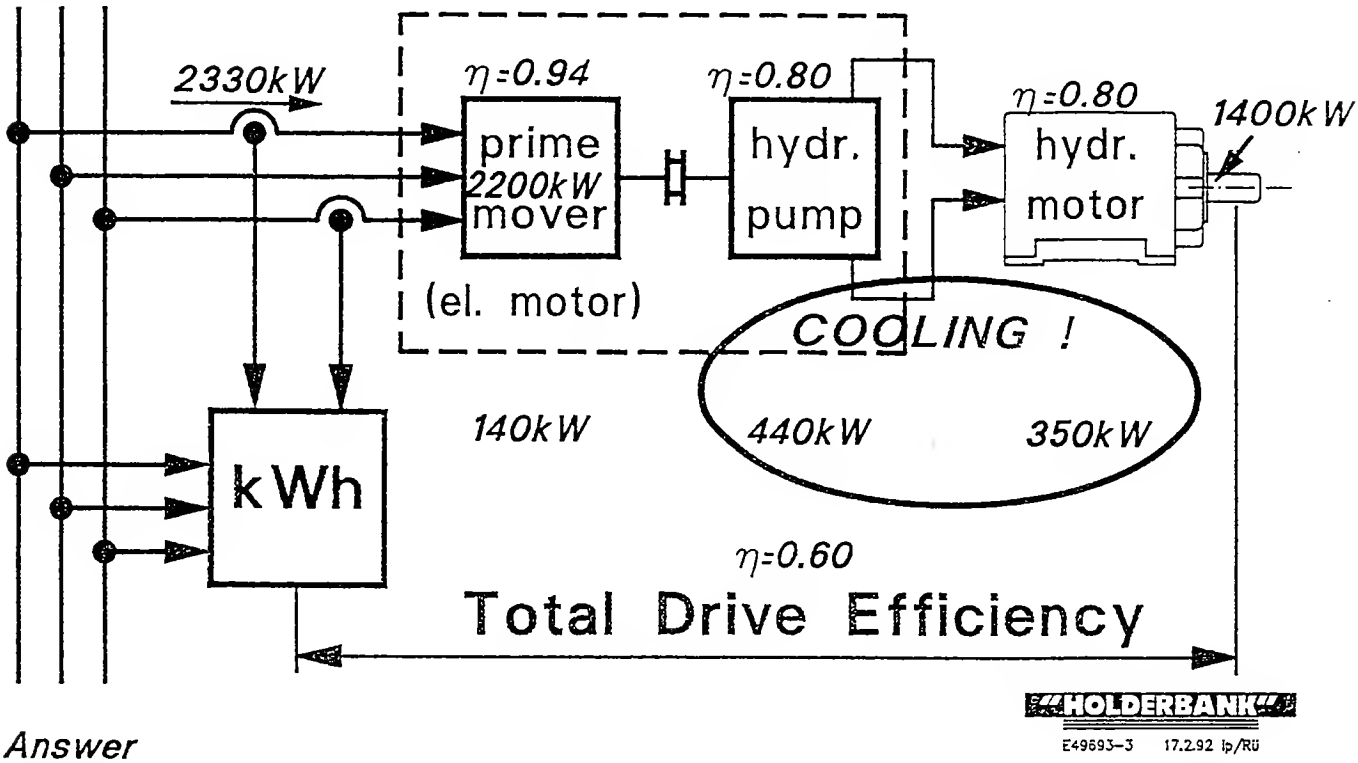
Answer

MAIN COMPONENTS OF A VVVF TYPE VARIABLE SPEED DRIVE SYSTEM



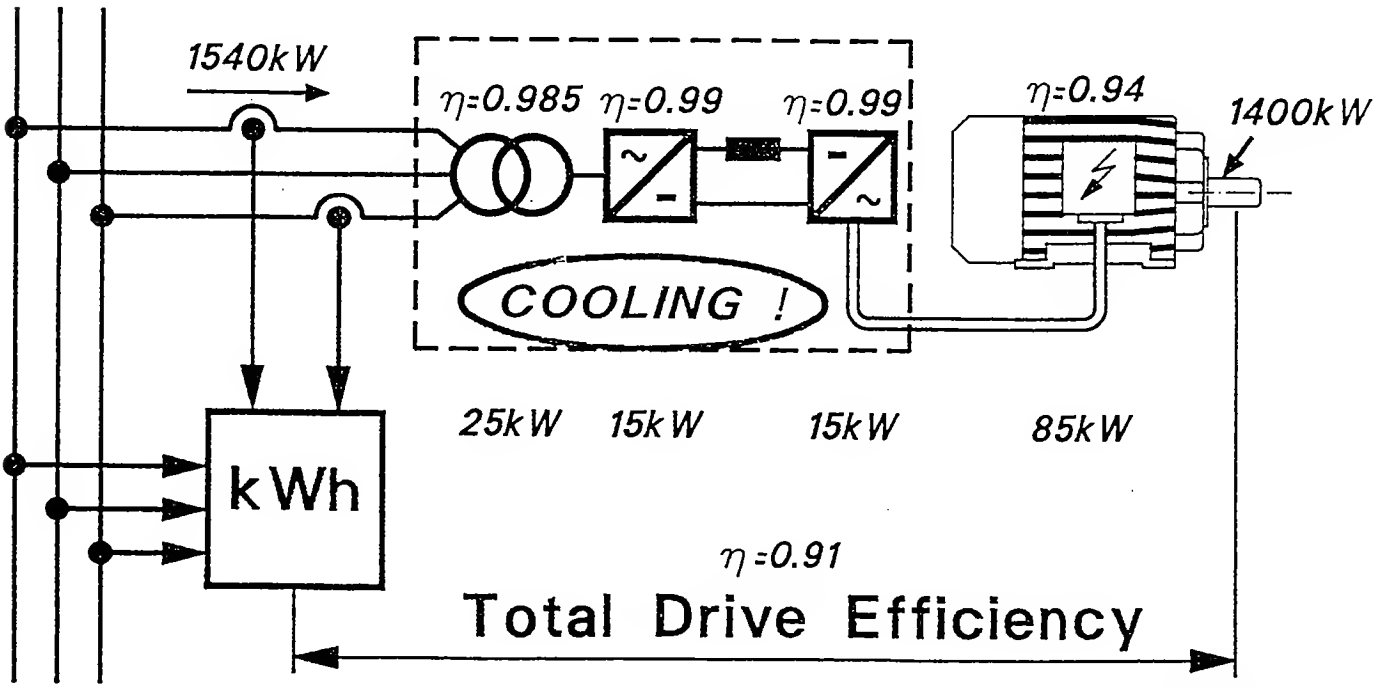
Answer

MAIN COMPONENTS OF A HYDROSTATIC VARIABLE SPEED DRIVE SYSTEM



Answer

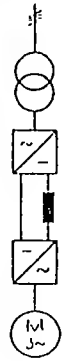
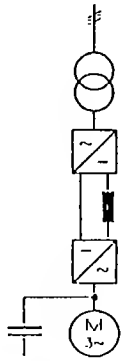
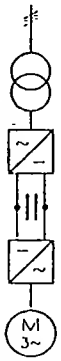
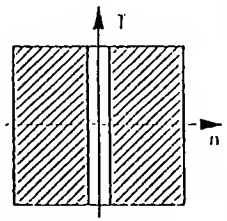
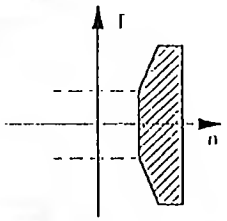
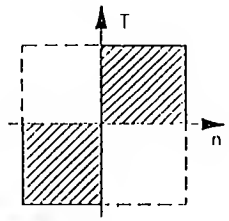
MAIN COMPONENTS OF A VVVF TYPE VARIABLE SPEED DRIVE SYSTEM



Answer

SUMMARY OF LARGE VARIABLE SPEED DRIVE SYSTEMS FOR THE CEMENT INDUSTRY (>1MW)

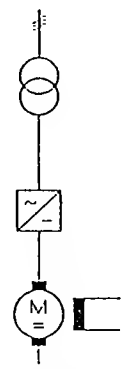
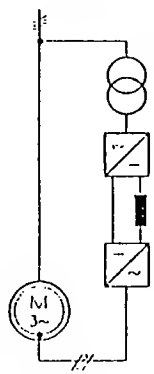
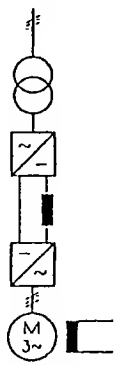
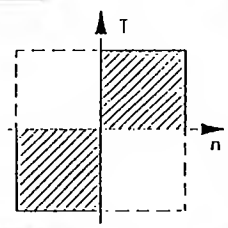
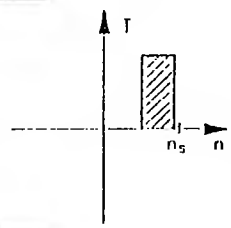
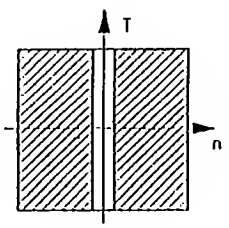
(based on cage induction motors)

Drive system	Current-source inverter-fed induction motor	Load-commutated inverter-fed induction motor	Voltage-source inverter-fed induction motor (PWM)
System diagram			
Type of machine	Cage induction motor		
Operating range T: torque n: speed			
Typical power range	up to 2000kW	500 ... 5000kW	800 ... 5000kW
Typical speed range	2 ... 100%	65 ... 100%	0 ... 100%
Significant properties	Single-motor drive of medium power Applicable normally for loads with squared torque/speed characteristic	Single-motor drive with nearly sinusoidal motor current and voltage Applicable normally for loads with squared torque/speed characteristic	Single- or multi-motor drive Good performance also at low speeds Power-factor to line nearly unity
Suitable for	Fan	Fan	Fan, (Kiln)
Price [%] Base: 1500kW/1000RPM	105%	110%	110%

2 Jun 94

SUMMARY OF LARGE VARIABLE SPEED DRIVE SYSTEMS FOR THE CEMENT INDUSTRY (>1MW)

(based on other than cage induction motors)

Drive system	Direct Current Drive	Wound-rotor induction motor with subsynchronous converter cascade	Converter-fed synchronous motor
System diagram			
Type of machine	Direct Current Motor	Wound-rotor induction motor	Synchronous machine
Operating range T: torque n: speed			
Typical power range	up to 2000kW	500 ... 5000kW	1000 ... 10000kW
Typical speed range	0 ... 100%	50 ... 98% of synchronous speed	(0) ... 10 ... 100%
Significant properties	Single-motor drive of medium power Good performance also at low speeds	Slip-power recovery Economic for small motor-speed-control-range High starting torque Suitable for retrofitting of existing slipring motors	Single-motor drive of high power High speed possible
Suitable for	Fan, Kiln	Fan	Fan
Price [%] Base: 1500kW/1000RPM	100%	95%	140%

2 Jun 94

Scenario of the example

A 1400 kW kiln fan drive operates at 80% of the nominal speed.
The required driving power at the fan shaft is, therefore, 720 kW.

	Drive 1	Drive 2
Type	hydraulic	electrical
Investment cost	240'000 US $	340'000 US $
Total losses	220 kW	100 kW
Total efficiency	(76%)	(87%)
Cost per kWh	4.2 US cents	4.2 US cents
Rate of kWh cost increase	4% p.a.	4% p.a.
Rate of interest	7% p.a.	7% p.a.

The systems are assumed to be in operation for 7000 hours per year.

PA/Rüe/gsr
March 92

ZÜRCHER KANTONALBANK

The Energy Saver

1 **Saves Electrical Energy on Pumps and Fans typically 20 to 40%!**

2 **Allows Optimization in Process Installations. Much is to be discussed with your specialists.**

3 **Reduces peak currents when AC-Motors are started. Your electricity supply company will be grateful!**

4 **Reduces mechanical wear. You save twice!**

5 **Reduces Noise. Your plant operators will be grateful for that, often also your neighbours!**

6 **Can be retrofitted to existing AC-Motors. This, of course, requires space and cabling**

Chapter 3

Drive Systems

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Drive Systems

Urs Rüesch

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1. INTRODUCTION

The power range of the drives in a cement work is very wide-spread. We normally find motors ranging from 0.2 kW up to 6 MW.

Low voltage drives are fed with 380 V to 580 V. For direct feed of drives exceeding 250 kW, the voltage range of 3.6 kV to 6 kV (11 kV) is used (high voltage motors).

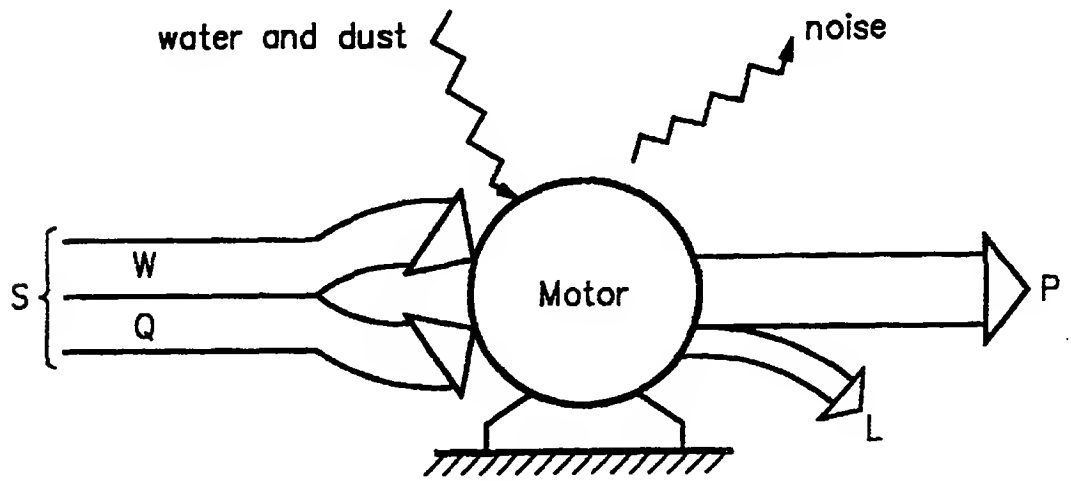
The evaluation criteria of high or low voltage drives are additionally dependent on:

- the distance between the motor and the substation (cable costs)
- investment costs of the drive
  (including costs of switching elements).

There is a certain tendency to raise high voltage to 10 KV and the low voltage to 660 V. With higher voltage, service currents and short-circuit currents are reduced, thus a number of advantages are attained, e.g.:

- smaller cable cross-sections (lower investment costs)
- full advantage can be taken of the 10 kV voltage by the switchgear
- low voltage motors can be used up to 500 kW.

Fig. 1: Input/output diagram



W : active power (kW)
Q : reactive power (kVar)
S : apparent power (kVA)

P : mechanical power (kW, J/s)
L : loss (~heat)

$$S : \sqrt{W^2 + Q^2} \longrightarrow \text{power factor} = \frac{W}{S}$$

$$W = P + L \longrightarrow \text{efficiency} = \frac{P}{W}$$

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The size and thus the investment cost of a motor is not only dependent on the power, but also on the speed of the motor:

- power = torque x speed

To understand the operation of a drive it is very important to know the characteristic "torque vs speed".

The influence of the torque or speed on the size of a motor can also be explained by means of a mechanical example:

- truck (30 t payload)
  motor: 15 l > power = 200 kW
- formula 1 racing car
  motor: 2 l > power = 400 kW

The truck motor generates a much higher torque at a much lower speed.

The very high degree of efficiency - up to 96% - of electrical drives should be mentioned. In spite of this high rate, the warming of large motors due to losses is remarkable.

Thus, ventilation should never be neglected in the planning of electrical drives. Depending on the prevailing conditions, machines can be cooled in different ways:

- natural cooling by convection, heat by itself produces an air current
- forced cooling by fans and filters
- forced cooling by air-to-air or air-to-water heat exchanger
  (closed-circuit ventilation).

The box-shaped casing makes the motor very versatile as regards its enclosure, so that it can be adapted to suit the wide range of environmental impacts encountered in cement works, most of which are far from favourable. For example, if a motor has to be installed outdoors, it can be fitted with a weather-proofing attachment. If the attachment is also lined with sound-absorbent material, it acts furthermore as an excellent silencer.

The type of protection of a motor is very important in the cement industry. The different types of protection are characterized by the so-called IP (interelement protection) class, followed by two numbers which indicate the degree of protection (according to IEC). e.g. IP 44 means:

- protection against foreign bodies with a diameter above 1 mm
- protection against spray water from all directions.

The cooling of a motor is closely related to its protection. The ideal solution of a completely closed, surface-cooled motor is problematic for big motors. Forced cooling with air filters or air/water heat exchangers are required. Figure 2 shows some examples.

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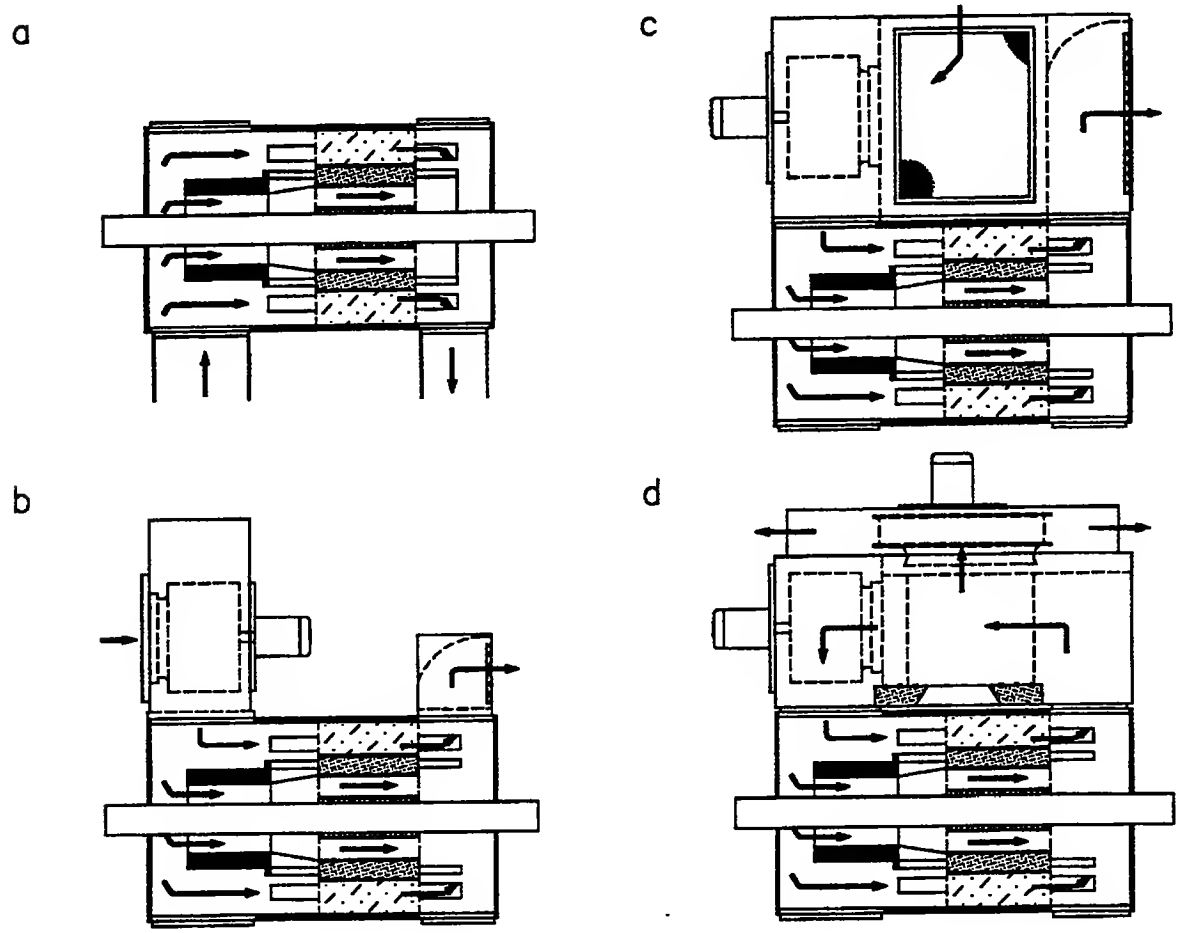
Fig. 2: Various alternatives for air-cooling and protection (for motors bigger than 1 MW)



a) separate ventilation, inlet and outlet through ducting

b) machine fitted with fan and exhaust shroud

c) machine fitted with through-draught ventilation unit with built-in filters

d) machine fitted with air-to-air cooling ventilation unit; the internal and external cooling air circuits are completely separate.

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2. MOTORS

2.1 Squirrel cage motor (induction motor)

2.1.1 Construction

The squirrel cage motor is in its construction the simplest motor used in the cement industry. The main feature is a rotor without external connections (no slip rings, no brushes). Its two bearings are the only parts exposed to wear and tear. It is furthermore economic in price.

Fig. 3: Typical connection diagram of a squirrel cage motor

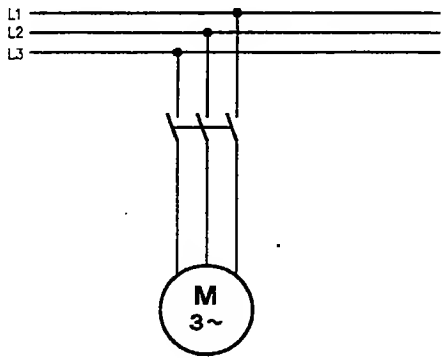
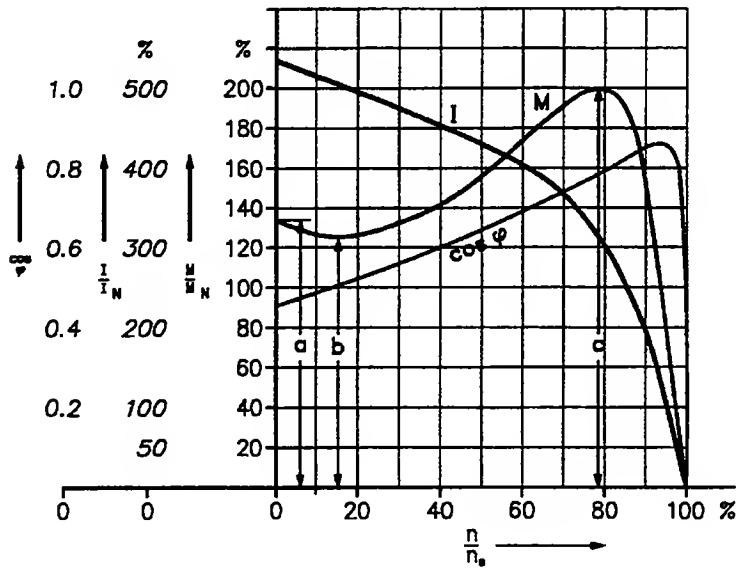


Fig. 4: Typical starting characteristics of a squirrel cage motor



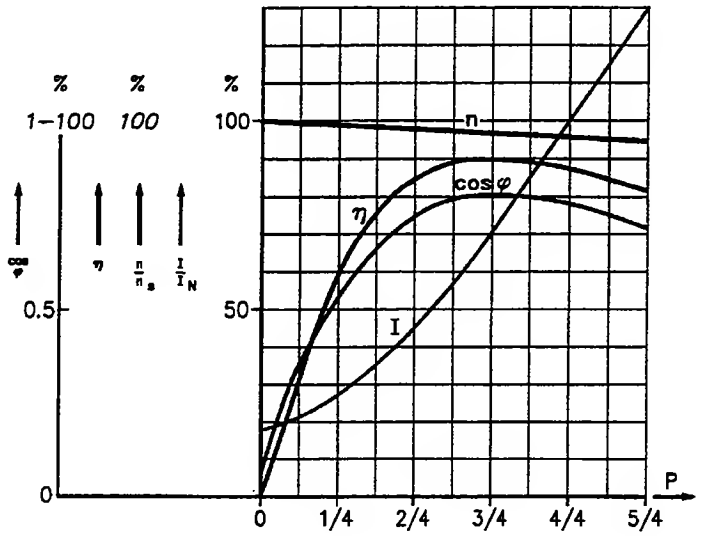
a :	starting torque	b :	saddle torque
c :	break-down torque	I :	current
M :	torque	n :	speed
cosφ :	power factor	η :	efficiency
P :	load		

2.1.2 Operating Characteristics

The squirrel cage motor has a high starting current of 3.5 to 7 times full load at relatively constant speed.

The torque changes with the square of the voltage.

Fig. 5: Operating characteristics of a squirrel cage motor (abbreviations see Figure 4)



2.1.3 Application

For almost any drive with a constant speed requirement and not too long a starting time. From a fraction of a kW to thousands of KW.

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2.2 Slip ring motor

2.2.1 Construction

The slip ring motor, like the squirrel cage motor, is an induction motor.

Its rotor windings are brought out to slip rings which allow to control the starting torque and current within a wide range.

Fig. 6: Typical connection diagram of a slip ring motor

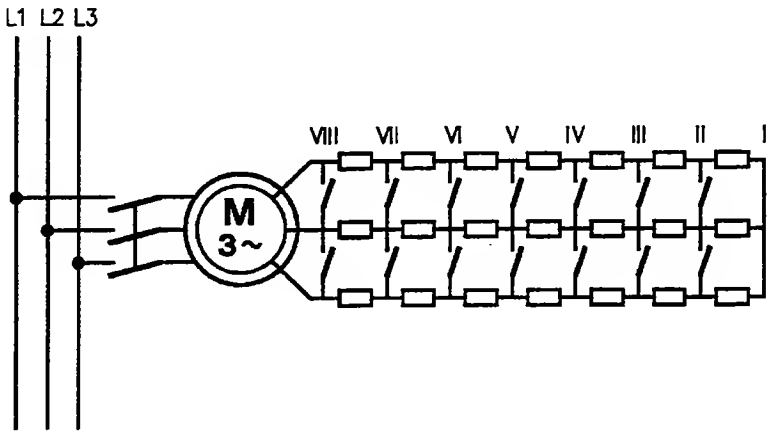
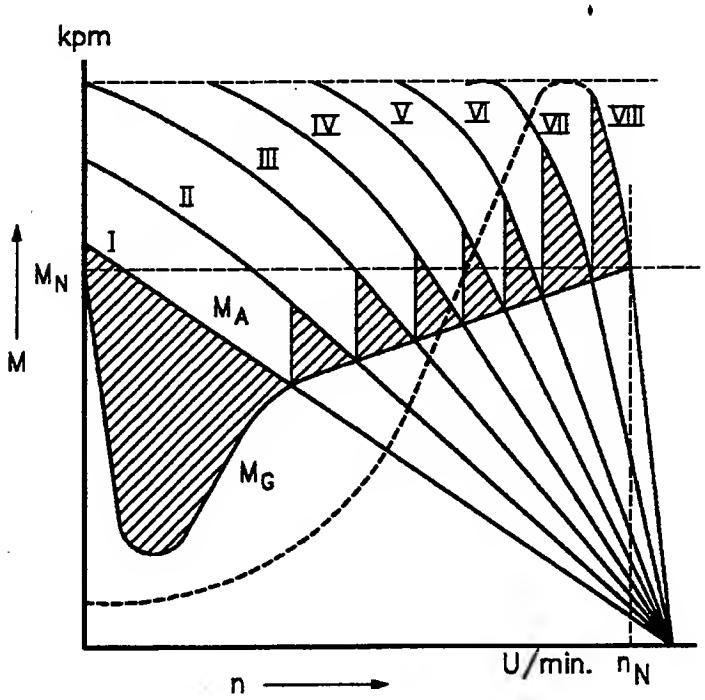


Fig. 7: Start of a slip ring motor by eight steps



M_N : nominal torque

M_A : torque characteristics during start-up

M_G : torque of the load

The introduction of an external resistance in the rotor circuit changes the torque characteristic of the motor and reduces the starting current. It allows changes of the torque of the motor and adaptation to the torque of the load (e.g. maximum torque at standstill).

The starting time of the motor can be extended since most of the heat is generated in the starting resistor away from the motor.

Metal starting resistors are built in different numbers of steps as required by the drive.

The last step of the resistor may be permanently connected to the rotor when the drive requires a softer torque characteristic.

Liquid starting resistors provide smooth and continuous acceleration.

2.2.2 Operating Characteristics

The slip ring motor is, once started and short-circuited with the resistor, not different from the squirrel cage motor.

2.2.3 Application

Where the starting torque and the starting current must be adjusted to the specific requirements of the drive. From one to thousands of kW.

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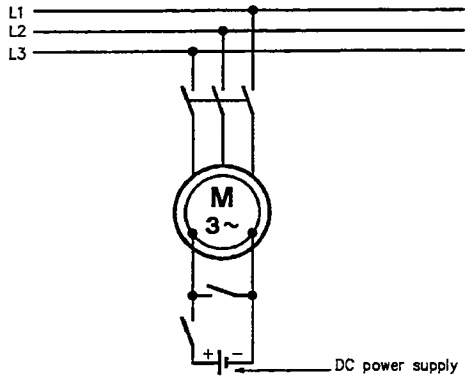
2.3 Synchronous motor

2.3.1 Construction

The synchronous motor has a rotor with salient poles. The rotor is connected by slip rings and brushes to a direct current power supply for its excitation.

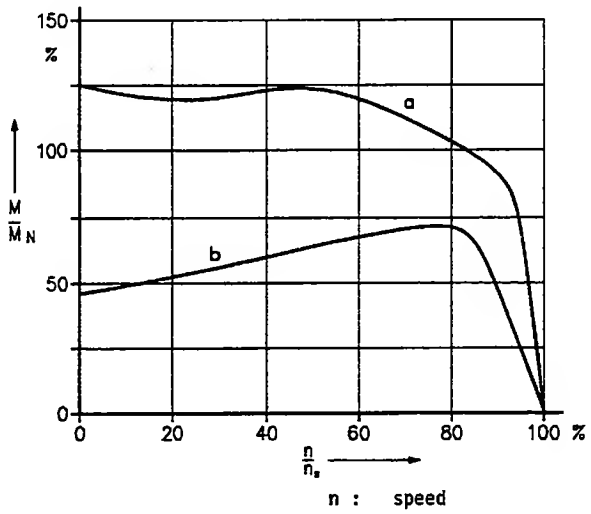
The AC windings are in the stator. The starting torque of an ideal synchronous motor is zero. To improve this situation, the rotors of synchronous motors are normally equipped with a squirrel cage type winding.

Fig. 8: Typical connection diagram of a synchronous motor



2.3.2 Starting Characteristics

Fig. 9: Typical starting characteristics of a synchronous motor



M : torque

a) Motor with massive poles and starting winding

b) Motor with laminated poles and starting winding

The synchronous motor accelerates, similar to a squirrel cage motor, up to near-synchronous speed. At his point the so far short-circuited DC winding is connected to the rotor. The torque now produced will accelerate the motor to synchronous speed.

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2.3.3 Operating Characteristics

The speed of the synchronous motor is proportional with the frequency of the supplying network and independent of the load on the motor shaft up to the break-down torque. The break-down torque is 1.5 to 1.9 times the nominal torque and depends on the excitation.

The great advantage of the synchronous motor is its capability of compensating reactive power and the very high efficiency of 96 to 98%.

2.3.4 Application

For rather steady loads with no speed control above 500 kW and where reactive power has to be compensated.

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2.4 Synchronous induction motor

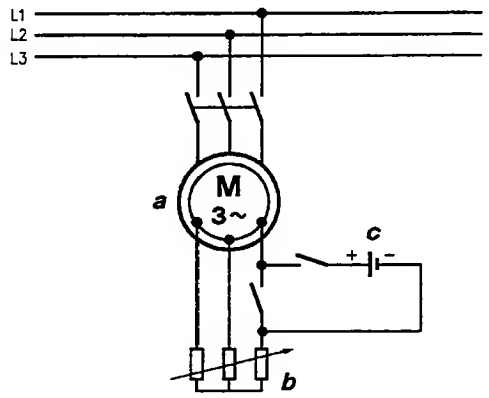
The synchronous induction motor combines the advantages of the slip ring motor and the synchronous motor.

It has the high starting torque at a low starting current of the slip ring motor and also the capability of compensating reactive power.

2.4.1 Construction

The synchronous induction motor is built like a slip ring motor; only the mode of operation differs.

Fig. 10: Typical connection diagram of a synchronous induction motor



a) induction motor

b) starting resistor

c) DC power supply

2.4.2 Starting Characteristics

The synchronous induction motor starts with a starting resistor like a slip ring motor. After the starter has short-circuited the rotor windings, the DC field is applied and the motor accelerates to full synchronous speed.

2.4.3 Operating Characteristics

In the synchronized operating mode the motor acts like a synchronous motor. It can operate with unity power factor or even compensate reactive power.

2.4.4 Application

For rather steady loads with no speed regulation, above 500 kW. Where reactive power has to be compensated and starting torque and current have to be adjusted to the requirements of the drive.

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2.5 DC motor (direct current motor)

Its name implies that the DC motor runs on a direct current power supply.

This power supply is not directly available in our cement plant. To connect a DC motor to our three-phase alternating current network, an AC-DC converter is required (see para 3. below).

2.5.1 DC Motor Design

Like all other electric machines, the DC motor consists of a stator and a rotor.

The excitation windings around the main stator poles are fed by a DC power supply and produce a constant magnetic field.

The DC current in the rotor conductors underneath the main poles produce a tangential force on the rotor which is identical to the torque on the motor shaft.

Fig. 11: Magnet flux of a DC motor

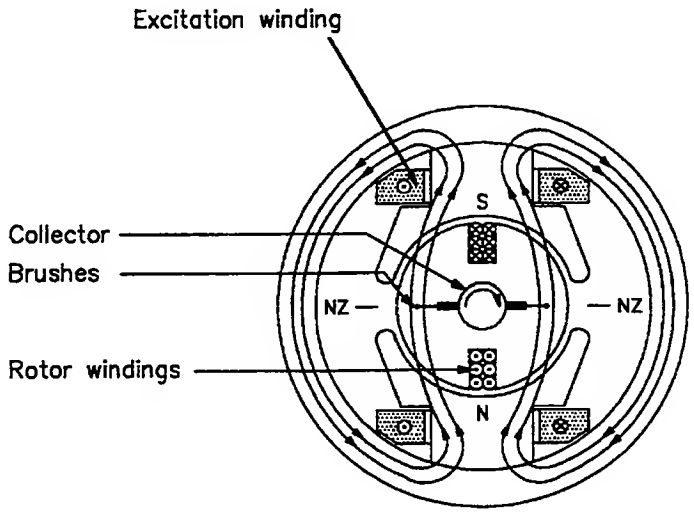
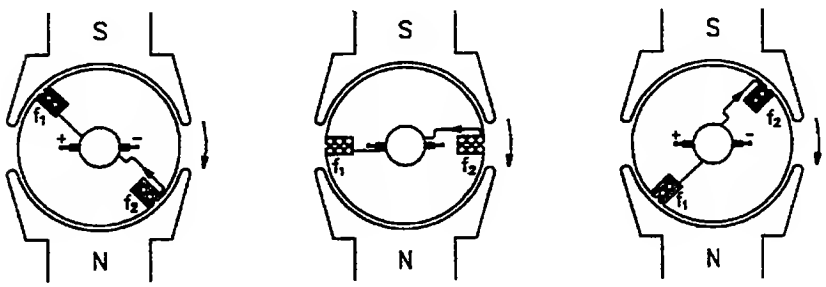


Fig. 12: Magnetic forces turning the rotor



The rotor now moves to the neutral position between the south and the north pole.

To keep the armature rotating, a polarity change in the rotor circuit is required. This is achieved with the collector.

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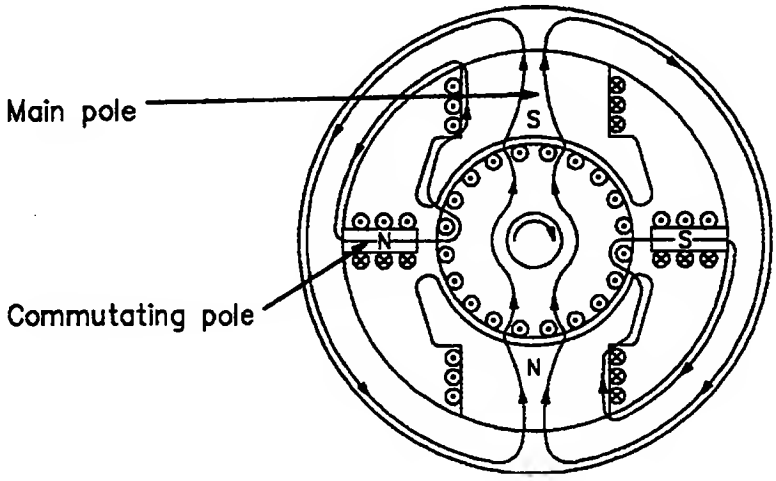
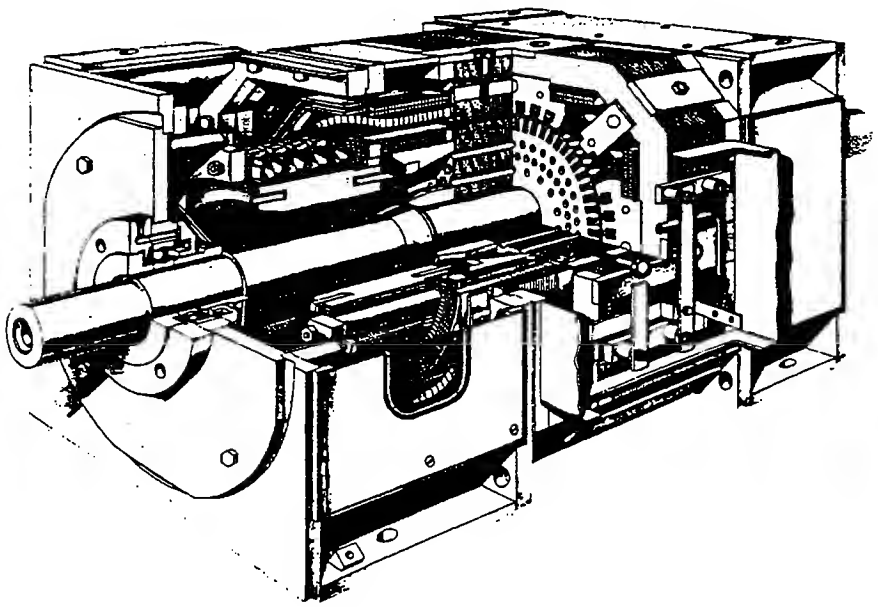
Commutation poles are furthermore installed in the neutral zones between north and south pole. These compensate the remaining magnetic field and thereby improve the commutation.

Fig. 13: DC machine with main and commutation poles



N: north pole S: south pole

Fig. 14: Cut-away view of a large DC motor



2.5.2 Characteristics of a DC Motor

The DC motor offers the great advantage of a simple torque and speed control over the full speed range, as well as high efficiency also for low speeds.

The torque can either be constant over the whole speed range, or any particular speed can be maintained independent of the torque.

These properties made the DC motor the most commonly used variable speed drive in the cement industry.

The main drawbacks of the DC drives are that they:

- are 2 - 4 times more expensive than squirrel cage motors
- are maintenance-intensive (collector, power electronic)
- are space-intensive (transformer, converter)
- require many spare parts

2.5.3 Application

DC motors are installed where variable speed is necessary and where the excellent characteristics of the drives outweigh the above mentioned drawbacks. Crusher feeders, weigh belt feeders, separators, kiln drives and kiln fans are such possible applications.

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2.6 Ring motor (gearless mill drive)

2.6.1 Construction

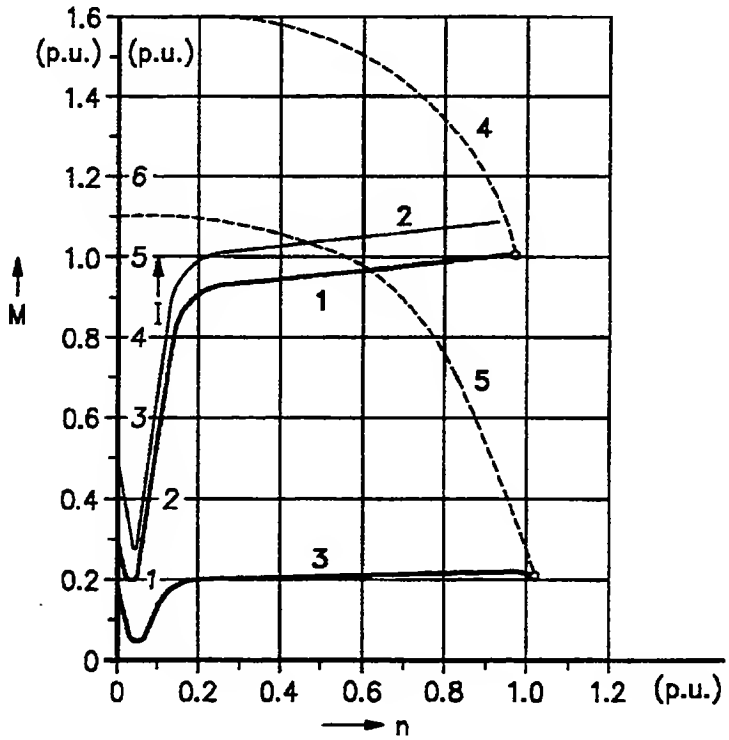
The ring motor is a synchronous motor. However, the stator has a variable frequency supply, generated in a frequency converter as described in para 3. below. The frequency ranges between zero and a few Hertz.

2.6.2 Operating Characteristics

The feature of variable frequency is used to start the motor. The torque can then be adjusted to the torque of the load resulting in a very smooth starting with low starting currents.

The power factor of the motor itself can be unity, the frequency converter, however, requires some reactive power.

Fig. 15: Starting characteristics of a synchronous motor with frequency and with asynchronous starting



M) torque

n) speed

I) motor current

1) torque of the load (mill)

2) torque with frequency starting

3) current with frequency starting

4) torque with asynchronous starting

5) current with asynchronous starting

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3. POWER ELECTRONICS

3.1 Introduction

The recent development of semiconductors has raised the application of power electronics to a level of considerable importance. The capacity/price ratio has become very interesting, and the reliability of the elements is in accordance with the industrial specifications.

To distinguish between electronics and power electronics, it may be said that electronics handle currents above 1 mA while power electronics handle currents above 1 A. The present maximum is about 9000 A (rectifier diodes) for a single element.

The principle elements of power electronics are:

- diodes
- transistors
- thyristors
- triacs
- GTO (gate turn-off thyristors)
- IGBT (insulated gate bipolar transistors)

Other electrical switching elements, such as the conventional thyratron, are not a subject of this exposé as they are vacuum-tube based and very rarely applied in the cement industry today; but functionally, these multi-grid tube elements possess similar operating characteristics.

The above mentioned electronic switches can be compared with other physical media, e.g. valves in water mains.

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3.2 Operating characteristics of power electronic elements

3.2.1 Diode

The symbol for a diode is shown in Figure 16 below. The operating characteristics of this element are very easy to understand. If a voltage "U_1" is applied across the diode from anode A to cathode C, the a current "i" will flow through the diode. If the voltage "U_2" is reversed from C to A, the diode blocks the current, i.e. no current will flow.

Figure 17 demonstrates the "rectifying effect" of a diode.

Fig. 16: Symbol for a diode

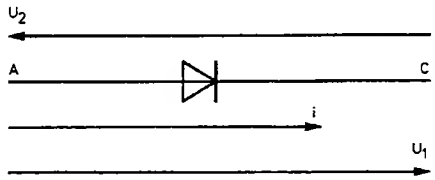
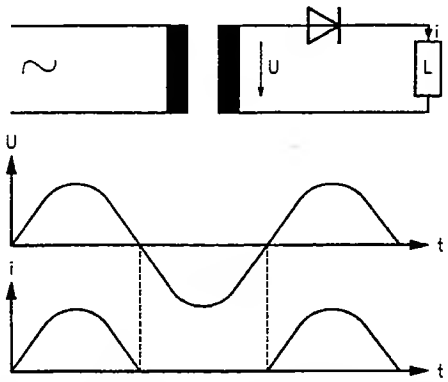


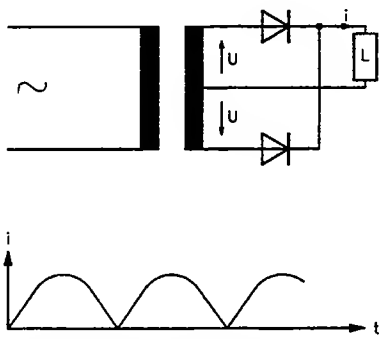
Fig. 17: "Rectifying effect" of a diode (half-ware rectifier)



The voltage "U" changes its polarity with every cycle. When the voltage across the diode is "positive" a current "i" flows through the load "L".

Figure 18 below shows a more efficient network.

Fig. 18: Full-wave rectifier



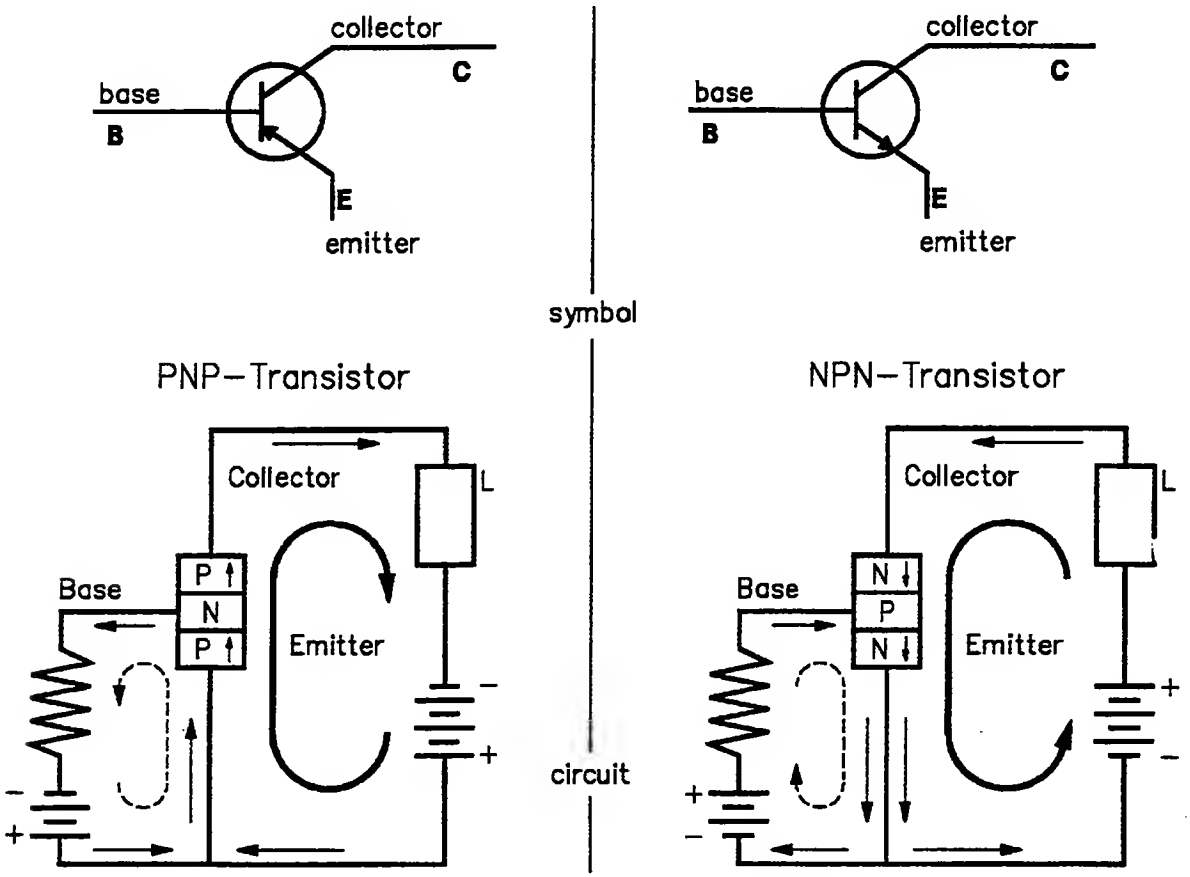
The current "i" is conducted alternatively by the two diodes. The time phase when the one "passes" the current to the other is called commutation.

3.2.2 Transistor

The transistor is built with three semi-conducting materials similar to a diode, which has two. Depending on the physical arrangement of these three semi-conducting materials, the literature speaks of PNP or NPN transistors. Also, the symbol and electrical circuitry is different, but today generally the PNP concept is manufactured.

The transistor can be used in many different circuit-configurations, but is basically a current amplifier, whereas the old electron tubes where voltage amplifiers.

Fig. 19: Symbols and typical transistor circuits



- small current controlling the transistor (closed or conducting)
- large current through the load

(note: arrows do not indicate the electron-flow)

The transistor input current (base-emitter) controls the transistor output current (emitter-collector) in a proportional manner over a certain range. Of course, the same transistor may also be used as switching element only, i.e. fully closed and fully conducting. Above is identically valid for PNP and NPN types.

Elements with output currents above 1 A are called power transistors, used as last stage in amplifiers and variable frequency converters for smaller drives up to approx. 400 kW.

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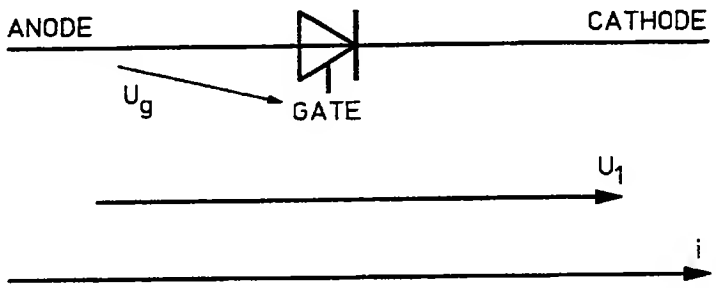
3.2.3 Thyristor, Triac

The symbol for a thyristor is shown in Figure 20. The operating characteristics of this element are similar to those of a diode except for the additional ignition voltage "U_g" from point anode A to point gate G.

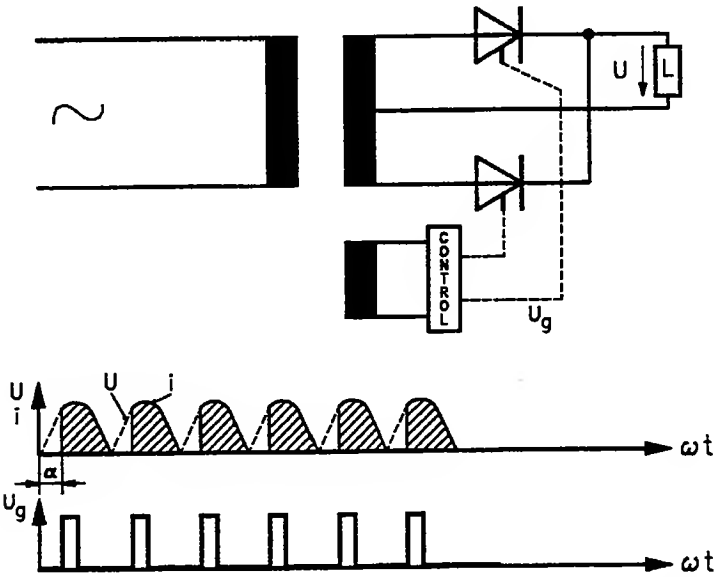
Fig. 20: Thyristor symbol



If a voltage "U_1" is applied across the thyristor, current "i" will only flow through the element when a voltage "U_g" (ignition voltage) is applied between A and G. The current "i" will flow only as long as the voltage "U_1" does not change its polarity.

The network in Figure 21 demonstrates the use of thyristors as a voltage regulator for DC.

Fig. 21: Network for a variable DC supply



By altering the angle of ignition, α, one can alter their voltage "U" across the load "L", theoretically within the range 0 - 100%.

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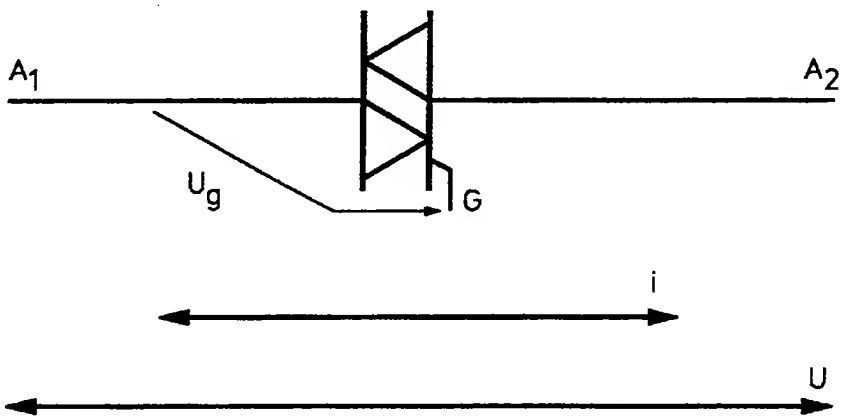
The symbol for a triac is shown in Figure 22 below. A triac consists of two antiparallel thyristors. This element represents an electronic switch for alternating current.

Fig. 22: Symbol for a triac.



A_1 = anode 1

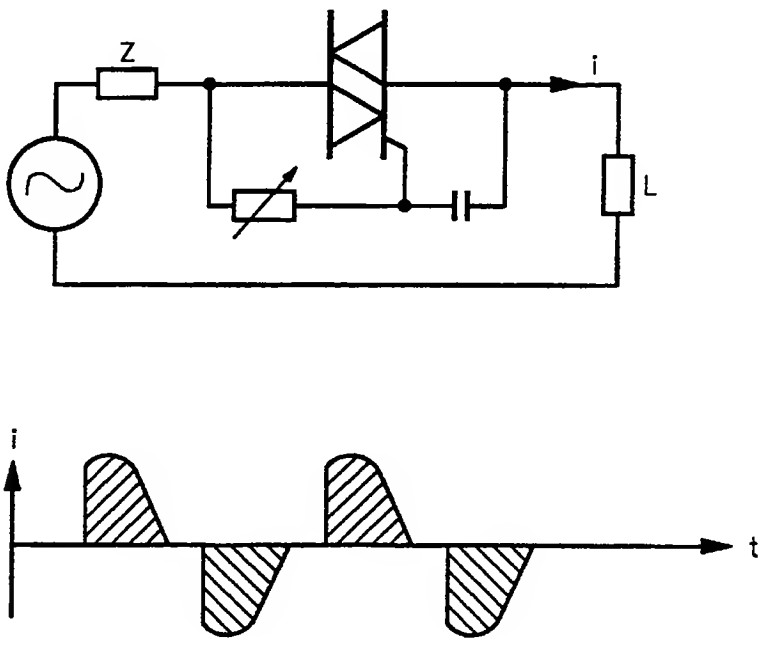
A_2 = anode 2

U_g = ignition voltage

Together with the ignition voltage "U_g", the triac is able to conduct the current in both directions.

Figure 23 below shows its application as a "voltage" regulator" for an AC-load.

Fig. 23: The triac as "voltage regulator" for an AC-load



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3.2.4 GTO (Gate turn-off-thyristor)

The symbol for a GTO is shown in Figure 24 below. The special feature of the GTO's is that they are not only turned on through their gate, but also off. Naturally, this would be the ideal switch, because no forced commutation equipment would then be needed to turn off the current when the thyristor is operated on direct voltage. In order to turn off the thyristor through the gate, negative triggering pulses which are large enough to reduce the load current of the thyristor below the holding current for a brief time are required. The turn-off pulse must be at least 10% of the forward current.

Fig. 24: Symbol for a GTO

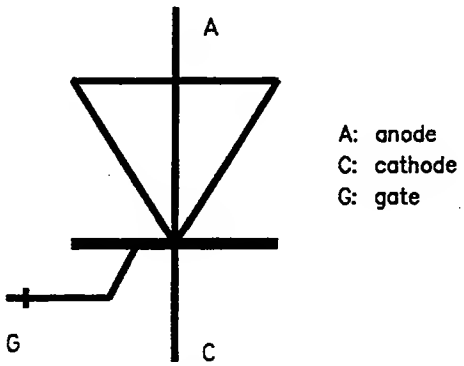
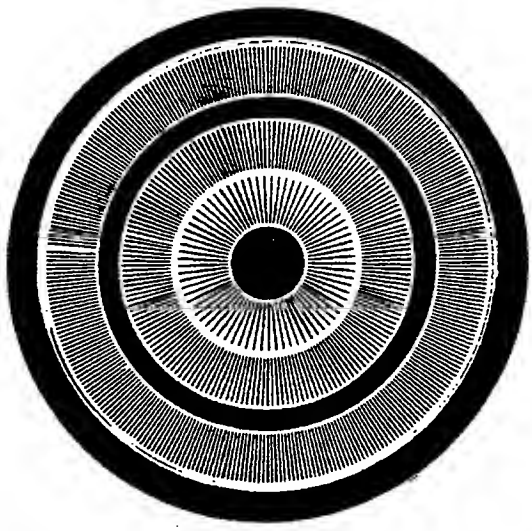


Figure 25 shows a pill of silicon with multitude of small, circular arranged "cathode-fingers" which are looking out of a connected gate-surface. Each "finger" is an independent small GTO. All these small GTO's are connected in parallel to a large GTO.

Fig. 25: Pill of silicon



Gate turn-off thyristors are used for forced-commutated current converters and static inverters.

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3.2.5 IGBT (Insulated gate bipolar transistor)

The IGBT is a combination of the advantageous characteristics of a bipolar transistor and a self-blocking field effect transistor (MOSFET). Its characteristics are a powerless drive like a MOSFET, a low forward resistance and a high inverse voltage like a bipolar transistor. Figure 26 shows the equivalent network and figure 27 the symbol for an IGBT.

Fig. 26: Equivalent network for an IGBT

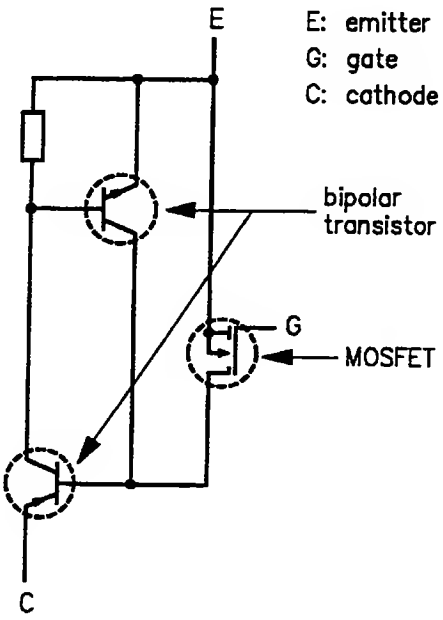
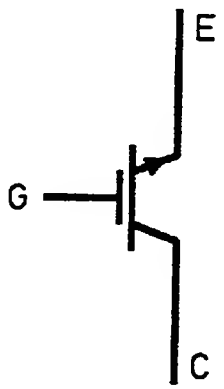


Fig. 27: Symbol for an IGBT



The IGBT is suitable for numerous applications in power electronics, especially in pulse width modulated servo and three-phase drives requiring high dynamic range control and low noise. They also can be used for power-supplies and other power circuits requiring high switch repetition rates.

IGBT's will replace the bipolar darlington-transistor in many applications because the control circuit is less sophisticated and thus cost-efficient.

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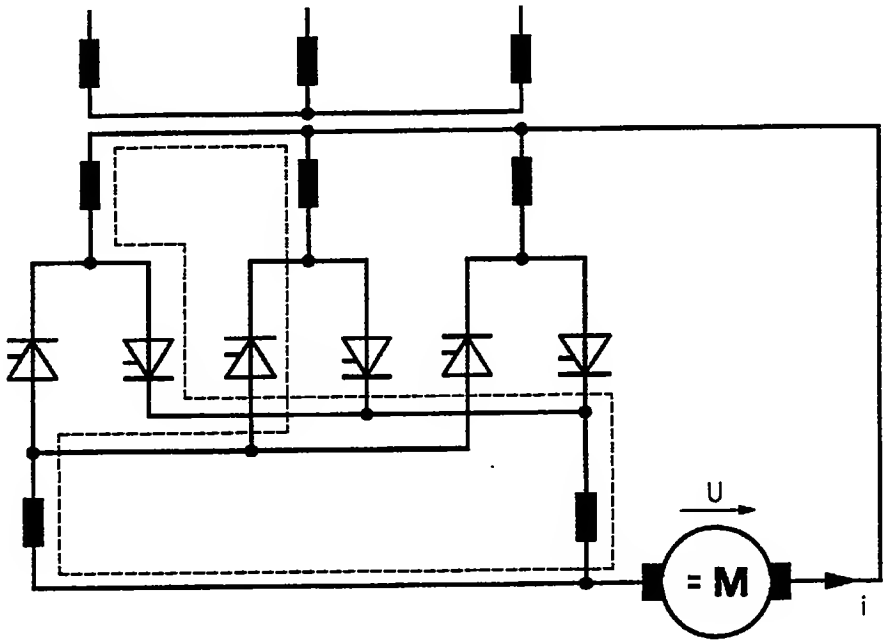
3.2.6 Application for power electronic elements

Figures 28, 29 and 30 illustrate how a frequency converter can be built with the aid of power electronic components.

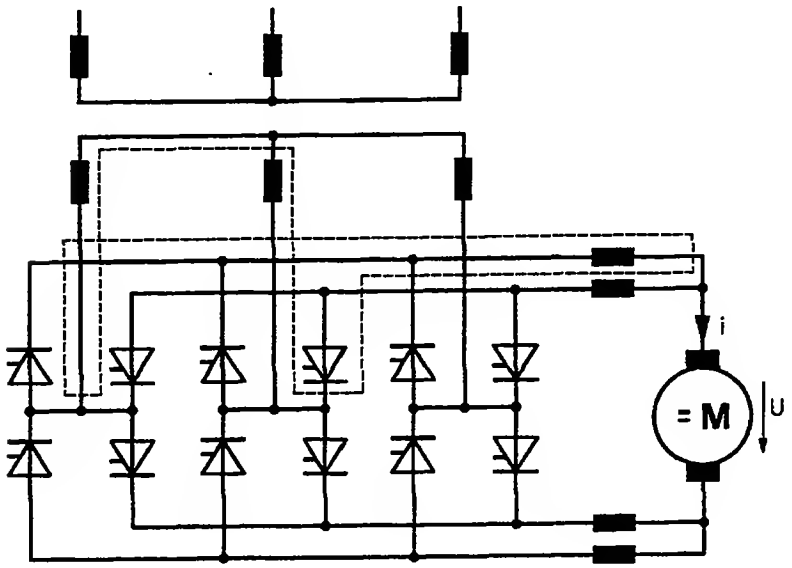
Fig. 28: 3-pulse anti-parallel circuit



— load current

----- circulating current (during transfer motoring to regenerating)

Fig. 29: 6-pulse anti-parallel circuit



— load current

----- circulating current (during transfer motoring to regenerating)

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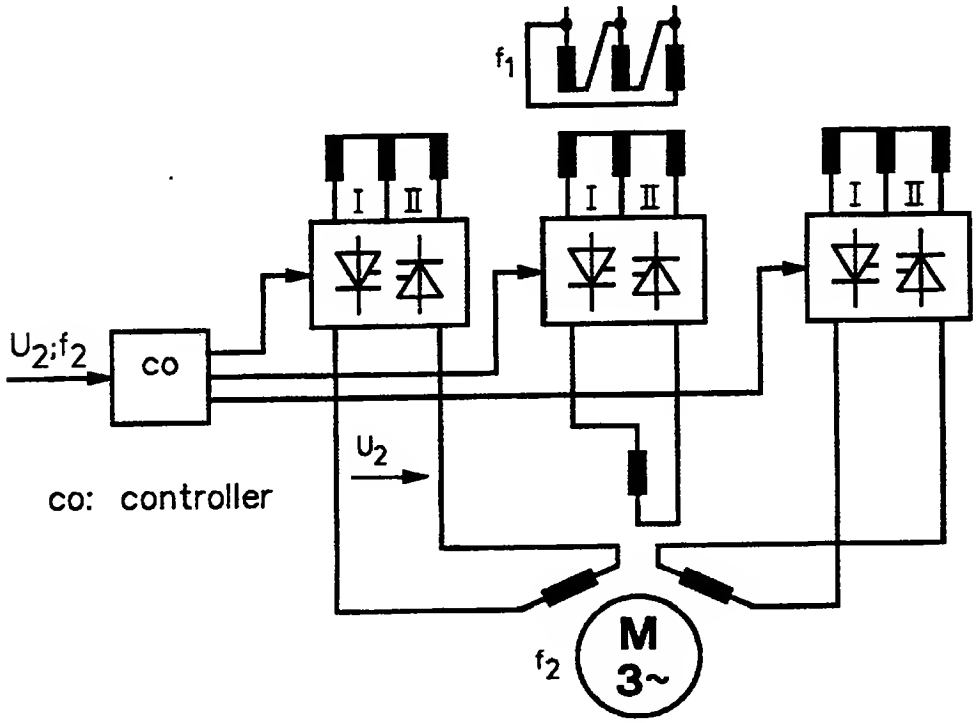
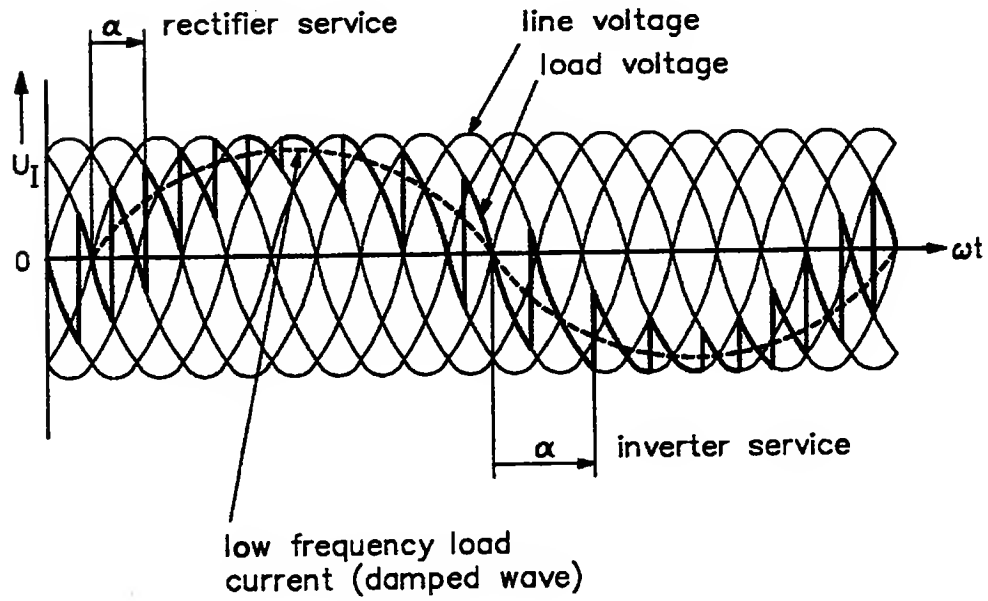
Figure 30 shows a mounting diagram and voltage from an output phase of a frequency converter. Each output phase is formed by a 6-pulse anti-parallel partial-current converter. Totally at least, 3x2-6 (3-phase; 2-anti-parallel; 6-pulse) = 36 current converters are needed. The two partial-current converters change between rectifier - and inverter service; so that the output-voltage will be sine shaped.

Fig. 30: Example for a frequency conversion with a time-dependent ignition control





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3.3 Application for power electronics in the cement industry

In modern cement works power electronics are used as:

(a) rectifiers for	- electrostatic filters
	- magnetic separators
	- DC power sources
(b) voltage regulators for	- speed (torque) control of DC drives
	- voltage control of electrostatic filters
	- electronic contactors
(c) frequency converters for	- speed control of synchronous motors, e.g. ring motor, squirrel cage motors
	- stabilized power sources for supply of control equipment, e.g. computers

3.3.1 Advantages of electronic elements

Electronic elements do not wear out. Their modular design permits quick trouble-shooting and short repair times. Furthermore, they offer ideal characteristics for motor controlling, e.g. speed variation.

3.3.2 Disadvantages of electronic elements

a) Cooling
Energy losses occur in electronic elements as they are not ideal switches. These losses produce heat and since the switching components have small dimensions, cooling is a considerable problem. The cooling media is usually air, but water may also be used (e.g. water-cooled thyristors of the ring motor at Rekingen). The ambient temperature of electronic boards is often specified up to a maximum of 45°C, thus in most cases air-conditioned rooms are necessary.

b) Distortion of the sine wave form
The above diagrams (e.g. Fig. 30) show that the power electronics "cut" the source sine-wave in such a way that "angles" are formed. The degree of distortion of the initial wave form represents a quantity of reactive power of higher frequency produced by the power electronics (harmonics).
This high frequency power causes the following inconveniences:

- emission of strong magnetic fields which can disturb control signals (control cables must be protected and separated)
- malfunction of other electrical equipment inside and outside the plant due to distortion of the voltage wave form.

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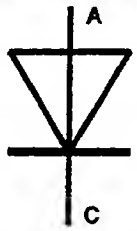
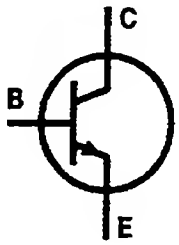
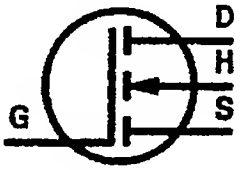
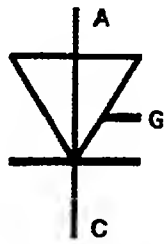
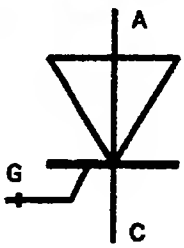
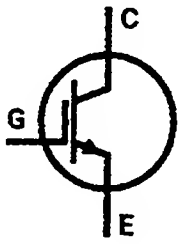
Summary of semiconductor elements for power electronics

Symbole	Description	max. voltage & current
	**DIODES** Diodes ore semi-conductor devices which allow current to flow in one preferred direction. If o positive voltage is applied, the diode operates in the forward direction; when the voltage is negative, it blocks. The properties of a good diode are high reverse resistance, low forward resistance and high allowable temperature. Standard and avalanche diodes are used mainly for rectification in circuits operating ot mains frequency Fost recovery diodes ore used for static frequency changers or in pulsed power supply units.	Revers voltage: 100 V ... 30 kV max. permissible RMS on-state current: 1A ...9000 A
	**POWER-TRANSISTOR** (Power field effect tranistor) The silicon pellet of the bipolar power transistor consists of three layers of altemate p and n type silicon material with two pen ¡unctions. On principle the succession of the layers npn or pnp is possible but today only the pnp concept is manufactured to achieve optimum electrical characteristics for power electronics. Power transistors are almost exclusively fumed on and off in "Switching operation". To allow the output current to flow, a forward base current must be maintained for the desired duration of the conducting state. When removing the control signal the power transistor reverts to the blocking state.	max. collector - emitter voltage: 100 V ... 1400 V max. DC collector current: 20 A... 400 A
	**POWER-MOSFET** The Power-MOSFET is a controllable switch, if it is running in forward directions. In this case the MOSFET can block up high invers voltage and can switch high power. In backward running it has similar qualities like a diode, but it is possible to influence the characteristic curve with the tension on the gate. The MOSFET has on advantage over the bipolar transistor because no control current is necessary. The Power-MOSFET are used for rectifier for DC-motors, frequency converter for AC-motors and power supply units.	Break-through voltage 50 V... 1000 V Output current: 2 A... 100 A
	**THYRISTOR** The silicon pellet of the thyristor consists of four or more foyers of alternate p and n type materials. It has two different conditions, one is high-resistance and the other one is law-resistance. The difference between a diode and a thyristor is, that a thyristor can switch between the two conditions with a current at the gate. To switch off the current, the thyristor needs a quenching capacitor. Thyristors ore used for contactless switches and controlled rectifiers.	Reverse voltage: 100 V... 5000 V max. permissible RMS on-state current: 10 A ... 5500 A
	**GTO** (Gate tum-off Thyristor) The special feature of the GTO is that they are not only turned on through their gate, but also off. Naturally, this would be the ideal switch, because no forced commutation equipment would then be needed to turn off the current when the thyristor is operated on direct voltage. The tum-off pulse has to be about 10% of the forward current. Gate tum-off thyristors are used for current converters and static inverters.	Reverse voltage: 100 V ... 4500 V max. current: 10 A ... 3000 A
	**IGBT** (Insulated Gate Bipolar Transistor) The IGBT is o technologically combined device having the advantageous characteristics of a bipolar power transistor and a self-blocking field effect transistor. This characteristic is therefore similar to the MOSFET in the input and to the bipolar power transistor in the output. The IGBT is suitable for numerous applications in power electronics, especially in Pulse Width Modulated frequency converters and three-phase drives.	max. collector - emitter voltage: 100 V ... 1200 V max. DC collector current: 15 A ... 400 A

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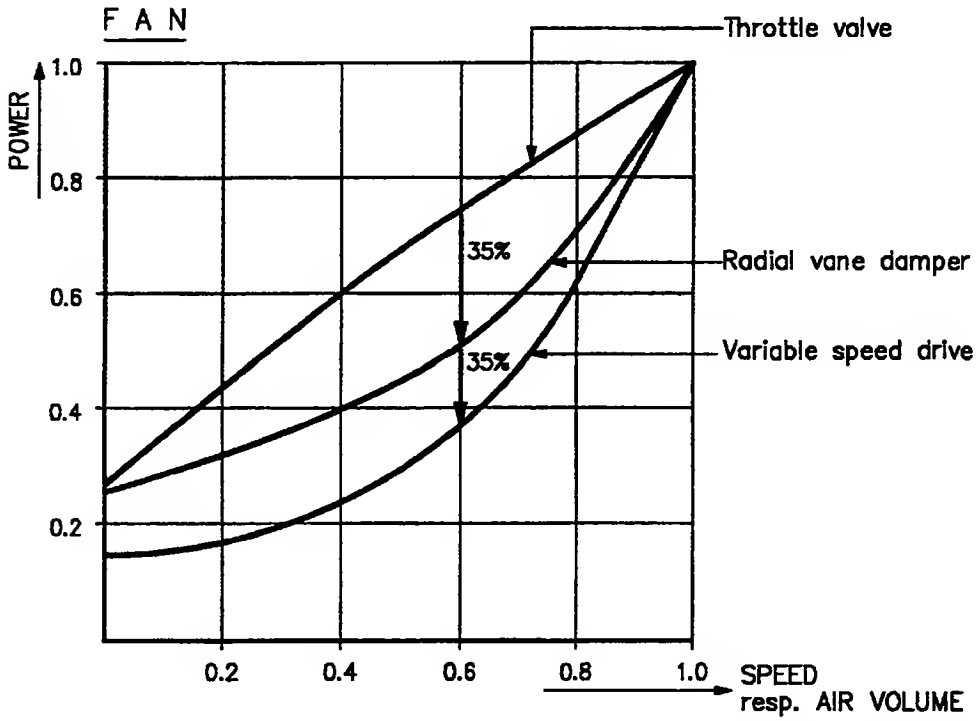
4. VARIABLE SPEED DRIVE SYSTEM

4.1 Introduction

The correct air volume for the process is often achieved by damper control or radial vanes in conjunction with constant speed drives. Considering the "BCM" (Better Cost Management) concept, constant speed and damper control for large fans (1-4 MW) is forbidden today.

Figure 31 compares the reduced energy consumption of using variable speed equipment for fans and pumps with radial vane damper and throttle valve. Where air or water quantities have to be adjusted according to process parameters, a variable flow is needed. Very often, this variable flow is created with more losses than necessary. Compared with other means of flow-adjusting devices, the variable speed drive can save a considerable amount of energy, especially at 50 to 90% of the rated speed.

Fig. 31: Power requirement at different speed



Traditionally, requirements for variable speed in the cement industry were covered with the application of direct current (DC) drives or occasionally by hydraulic drive systems.

As a result of new semiconductor developments in the field of power electronics, many static converter circuits have become reality in recent years. In addition to the traditional DC drive, these static converters have opened up new applications for variable-speed AC drives of high ratings.

This paper presents a number of systems, showing where they can be applied and quoting the criteria which simplify a choice from the wide variety offered.

Some outstanding advantages of a variable-speed drive as follows:

a) Optimal process control

b) Reduced stress on machines and supply system during starting

c) Better utilization of the primary energy owing to the higher efficiency

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Familiarity with the entire spectrum of electrical and mechanical variable speed drive systems is not only useful when ordering a new plant, but is equally necessary when carrying out partial modernizations or conversions, e.g. replacing an installation involving undue maintenance or with a poor efficiency.

The following catchwords applies for drive specification:

Robust	The drive system must be designed to cope with the typical cement plant environment and the type and quantity of dust prevailing at the location of installation, e.g. clinker dust for cooler fan drives.
Ease of maintenance	The necessary amount of man-hours required by the equipment must be minimal. Diagnostic systems must help to identify failures and indicate steps to correct the fault/failure. Modular design and access must allow for a fast replacement of the defective component in order to restore normal operation.
Reliability	High reliability shall be achieved with adequate sizing of a well-proven drive system. The system shall not be overengineered with additional redundant equipment, which increases the initial installation cost.
Efficiency	Total drive system efficiency is of utmost importance, since it will substantially influence the operating cost for many years to come at an always increasing cost of electrical energy
Investment cost	Last but not least, also the investment cost shall be considered. However, an evaluation oft the investment cost is only meaningful if complete systems are compared including auxiliary installations (e.g. differences in cooling systems, civil works etc.) as well as the operating cost over the next ten to fifteen years.

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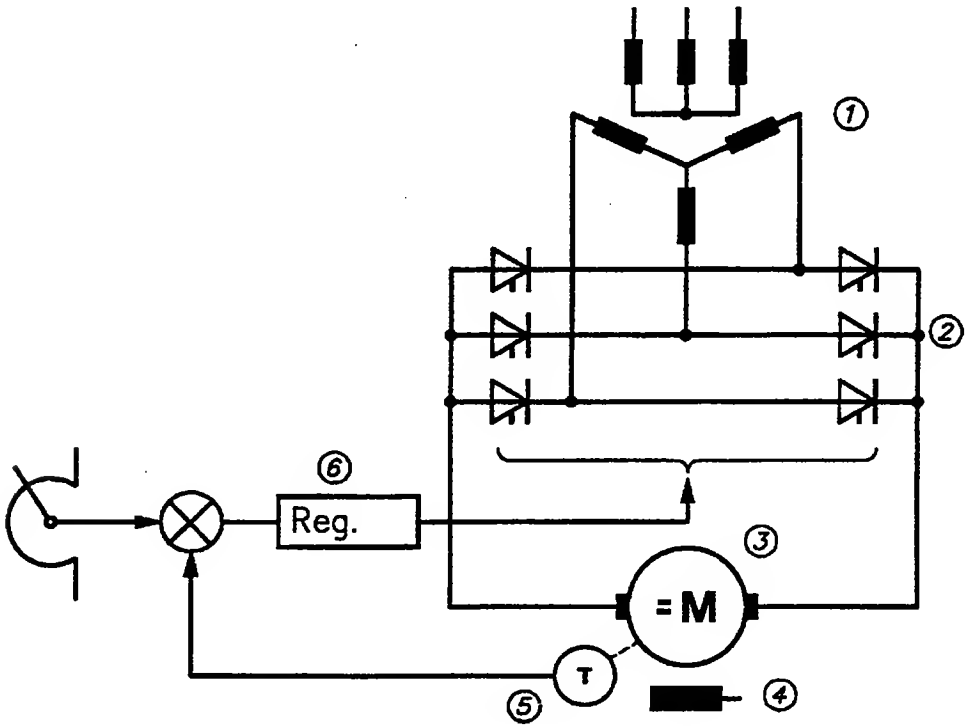
4.2 Electrical variable speed drive system

4.2.1 DC drive

Fig. 32: DC drive system



A DC drive system normally includes:

- 3-phase isolation transformer (1)
- 3-phase full wave rectifier (2)
- DC motor (3) with shuntfield (4) and tachometer (5)
- electronic speed regulator (6)

The speed of the DC machine varies proportionally to the applied armature voltage. Motor field weakening can increase the speed even more, but the result is reduced torque. DC drives can be built from less than 1 kW to approx. 1000 kW, considering motor speeds of 3000 min^{-1} for the smaller type and 700 min^{-1} for the larger sizes. At steel mills, large DC drive systems are built up to 8 MW with approx. 100 min^{-1}. The motor size is the limiting factor due to the centrifugal forces of the commutator. The usable speed range is almost infinite, since the DC drive can start and run close to zero speed even under severe overload conditions. Due to the wide speed range, DC drives in most cases require external forced-cooling systems.

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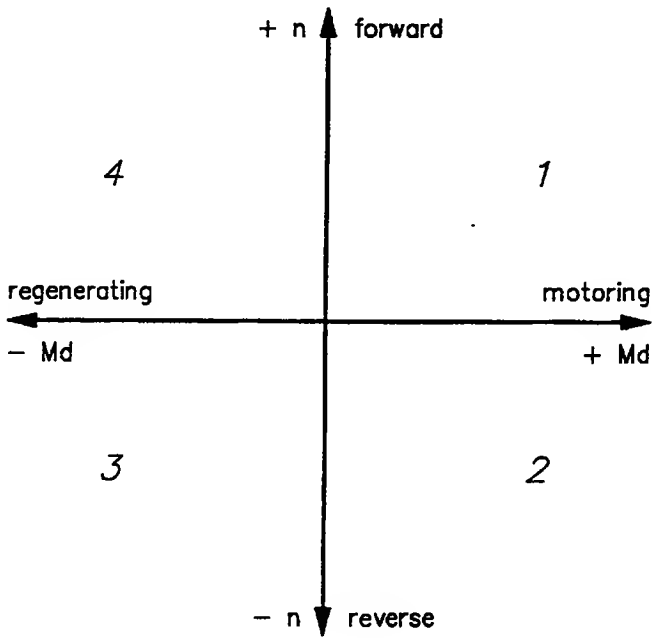
4.2.1.1 Operating Characteristics

The scheme as shown in Figure 33 can operate only in quadrant 1, i.e. motoring in forward direction. For full 4-quadrant operation, a double anti-parallel (back-to-back) thyristor bridge arrangement is necessary.

Once the DC motor is equipped with a forced-cooling system, no torque limitations exist. Totally closed DC machines are available too, but they are oversized and cover a limited speed range only. Therefore, a totally enclosed fan-cooled machine is usually very uneconomical. The field weakening range is not used in the cement industry, since it serves mostly for winder applications.

Fig. 33: 4-quadrant operation



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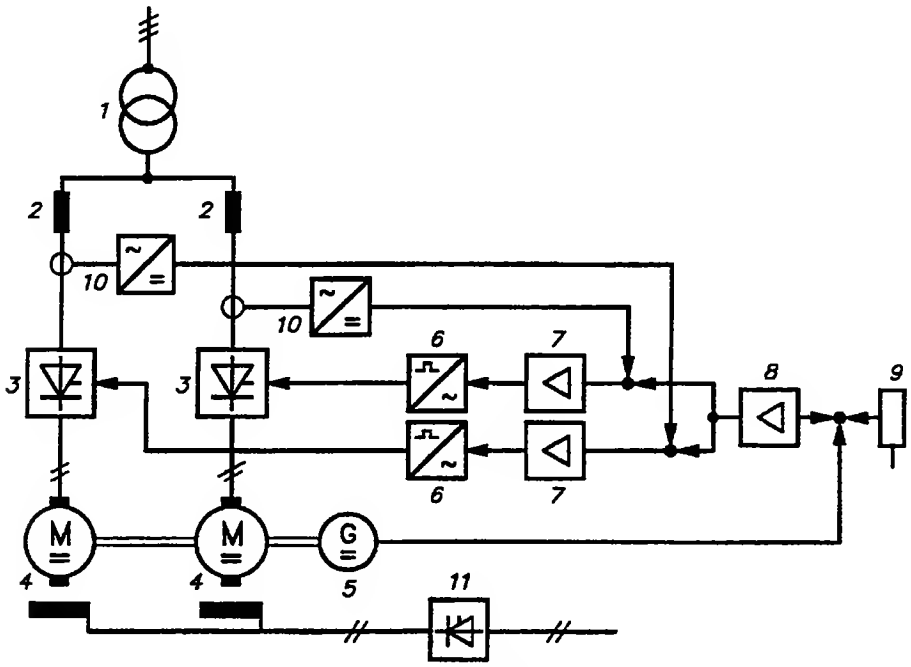
4.2.1.2 Application

The DC drive system is widely used in the cement industry for the following machines:

- kiln main drives 200 - 500 kW and twin drive
- large fans (e.g. kiln, raw mill) 800 - 2000 kW
- weigh feeders 5 - 15 kW
- apron feeders and special belt conveyors 20 - 100 kW

Due to the rapid developments in power electronics, the variable speed drive technology has gone through various stages in the last decades, but the DC drive represents still an efficient, approved and economic solution today. The commutator, the most delicate part of the whole system, and the cooling system require special and permanent maintenance attention. These two aspects explain the desire for other variable speed drive systems without commutator.

Fig. 34: Typical schematic diagram of a twin drive for a rotary kiln employing thyristor-controlled DC motor



1	Rectifier transformer	7	Current controller
2	Reactor	8	Speed controller
3	Thyristor rectifier, controllable	9	Speed reference potentiometer
4	DC motor	10	Current transformer and rectifier for the current actual value
5	Tacho generator	11	Thyristor rectifier, uncontrollable
6	Gate control unit		

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4.2.2 AC drive with squirrel cage motor

The variable speed drive system, using a squirrel cage motor, consists of the following main components (see Fig. 35):

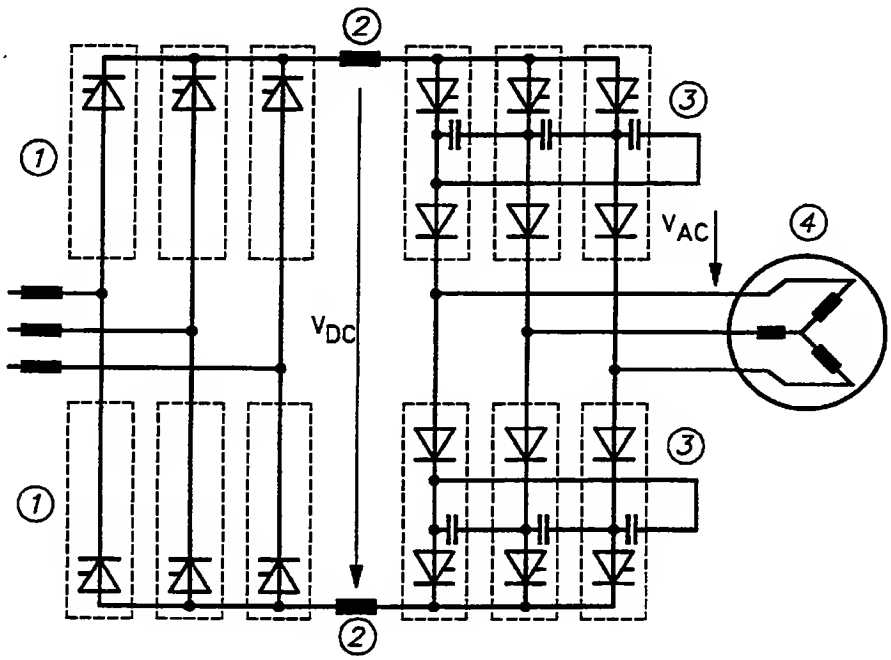
- 3-phase full-wave rectifier (1)
- DC-intermediate link with reactors (2)
- forced commutated inverter (3)
- normal 3-phase squirrel cage motor (4)

An input rectifier creates the intermediate link DC voltage. The reactors inserted in the DC link uncouple the AC power supply side from the inverter side driving the asynchronous motor. The forced commutated inverter is using the principle of phase sequence turn-off. Each of these switching circuits consists of a thyristor, a diode and a commutating capacitor. The input rectifier is current-regulated and supplies its power into the DC intermediate link. The output inverter is voltage regulated, maintaining the correct V/Hz relationship over the speed range. The speed of the motor is adjusted by variable frequency. No tachometer is needed, since the frequency feed-back signal is taken from inside the panel. The normal speed range is from 5 to 50/60 Hz and up to approx. 90 Hz.

The power of the available units presently ranges from 5 kW to approx. 1800 kW of several typical AC input voltage levels like 380, 415, 500, 660 V AC. In special cases, units of 3 MW have been built for all motor speeds up to 4000 min^{-1}.

4.2.2.1 Current-source inverter-fed induction motor

Fig. 35: Frequency converter with phase-sequence turn-off



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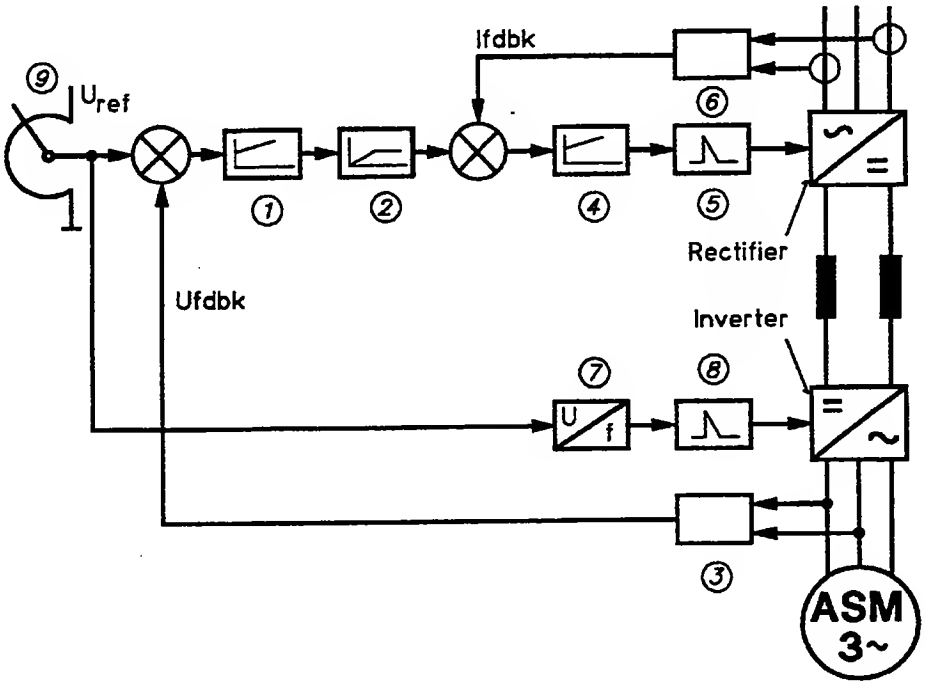
Operating Characteristics

The converter as described above does not need any additional semiconductors in order to perform a full 4-quadrant operation. The flow of energy is reversed by reversing the polarity of the DC link voltage, with the current direction remaining unaltered. At speeds below 5 Hz, torque pulsation may be noted as a result of low frequency motor-current harmonics. This effect is damped by the mass of the mechanical system.

The prevailing use for this type of variable speed drive is to be found with fans and pumps in many different configurations.

Usually a totally enclosed, fan-cooled standard motor can be chosen with no extra forced cooling system because the torque curve of these mechanical devises follows a square function versus speed.

Fig. 36: Typical block diagram of the voltage-controlled variable frequency converter



1	voltage controller	5	trigger unit of the line-commutated converter
2	value generator and limiter	6	actual current measurement
3	actual voltage measurement	7	voltage/frequency converter
4	current controller (secondary)	8	trigger unit of the self-commutated converter
		9	voltage reference potentiometer

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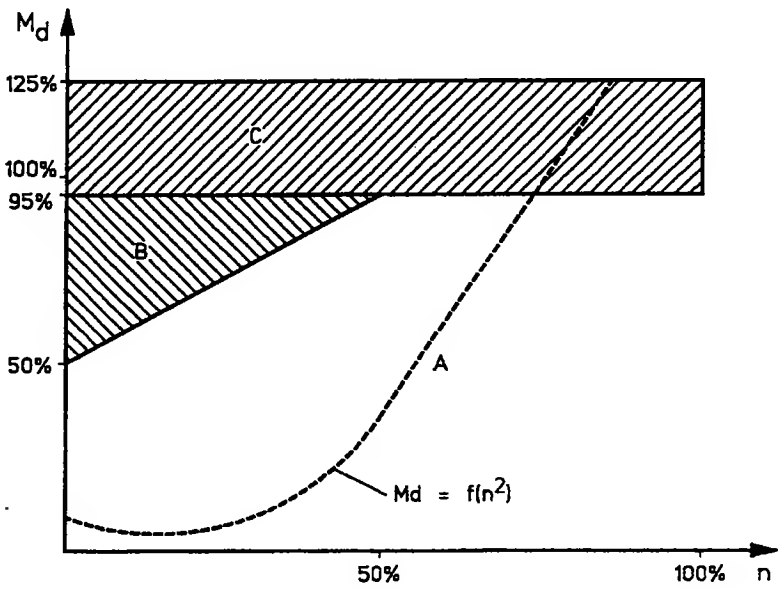
Fig. 37: Torque/speed diagram



A : continuous duty self-ventilated

B : continuous duty with forced ventilation

C : intermittent duty

Application

Cooling equipment manufacturers and suppliers of water pumping stations have used this type of variable speed drive since 1975. Several converter manufacturers have application references for more than 1000 units of a wide power range within the past years.

The initial equipment cost is slightly higher than for a comparable DC drive system due to more semiconductor elements in the power path. On the other hand, the squirrel cage motor is much cheaper than the DC motor. Harmonic content and power factor aspects are identical with those of a DC drive since the input rectifier represents the same type of load to the supply side network.

This system offers interesting aspects for modifications of existing equipment. When introducing the variable frequency converter to an installation which was so far connected to a constant 50 or 60 Hz supply, the motor can be speed-controlled. It might be of interest that even a speed-increase is possible by applying more than line frequency (e.g. 70 - 90 Hz). In other words, a belt conveyor can run faster and develop more power without motor - or gear change!

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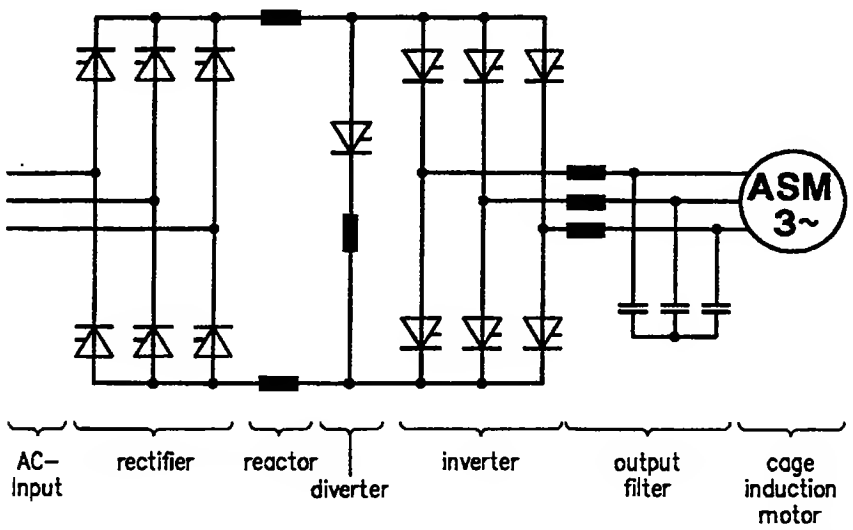
4.2.2.2 Load commutated inverter-fed induction motor

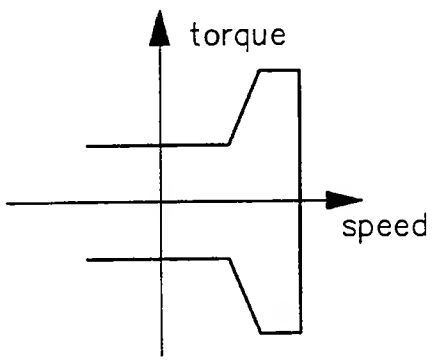
The main components of this drive type consist also of a line-side converter, DC link circuit reactor and load-side converter. Additionally, a so called diverter is added in order to force commutate (switch-off) the inverter at low frequencies, while an output filter is added to smooth output waveforms and provide excitations for the induction motor.

Fig. 38: Cage induction motor with load-commutated inverter (output Filter)



As with the conventional frequency converters, for normal operation the converter is commutated by line-voltage and the inverter is commutated by the load. Unlike the conventional type, the diverter circuit on the DC link is used to commutate the inverter bridge for low frequency operation. The entire inverter is commutated by the diverter, then appropriate thyristors are gated (switched on) to produce the three-phase output. Above about 60% of rated frequency, depending on the motor, the diverter circuit turns off and the inverter is load commutated by the combined effects of the output filter and the induced motor-voltage of the induction motor itself. The filter is sized to provide motor excitation over a wide frequency range. The voltage and current waveshapes are nearly sinusoidal, typically containing less than 5% harmonic distortion at rated output. No motor derating is necessary.

Fig. 39: Torque/speed diagram



Application

The load-commutated inverter-fed induction motor is best suitable for loads with squared torque/speed characteristic (Fig. 39) and reduced speed range, i.e. for fans.

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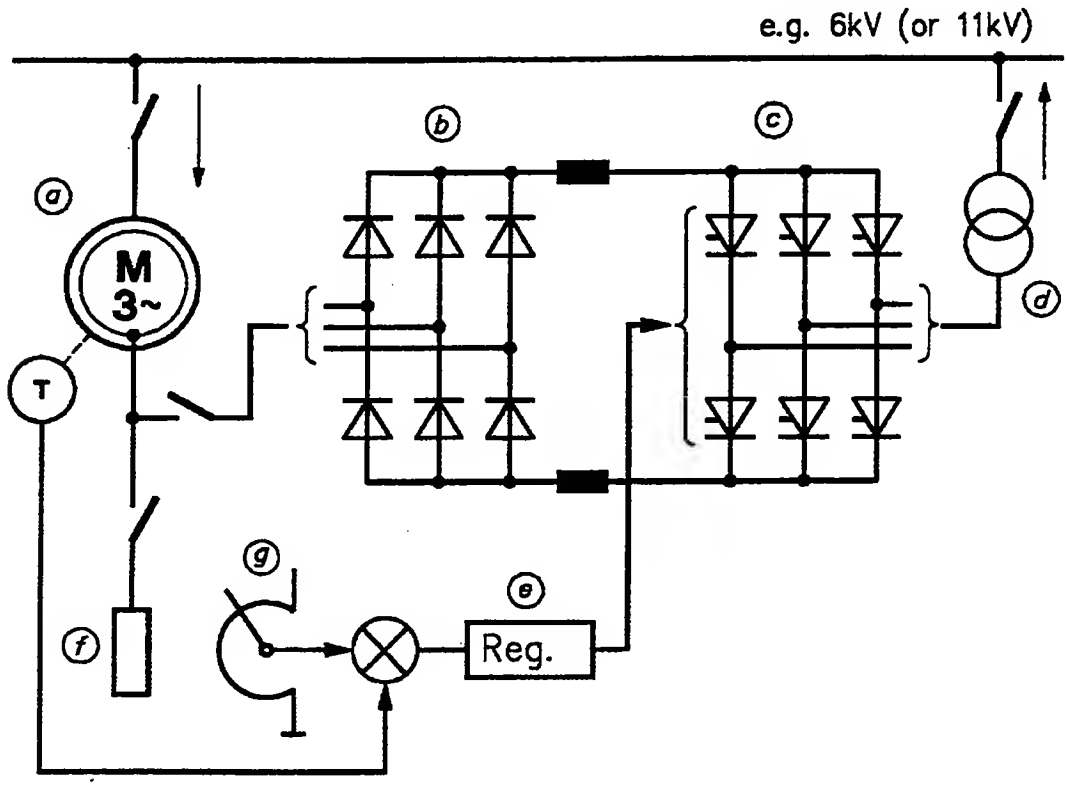
4.2.3 AC drive with slip ring motor

Fig. 40: Schematic of sub-synchronous cascade



a) HV slip ring motor with tachometer (T)
b) 3-phase full-wave rectifier (diodes)
c) 3-phase full-wave inverter (thyristors)
d) matching transformer
e) electronic speed regulator
f) starting resistor
g) speed reference potentiometer

The stator of the slip ring motor is connected directly to the power system. The rotor slip power, which is proportional to the slip frequency, is fed back into the power system via a diode rectifier, a smoothing reactor, an inverter and a matching transformer. A starting resistor is normally used to drive the motor up to approx. half speed, then the rotor is connected to the converter and the electronic regulator takes over the speed control. The static converter section has to be sized only for the rotor slip power. The sub-synchronous cascade system is mostly used to drive large pumps, fans and compressors, where the torque increases with the square of the speed. A considerable change in capacity is obtained by only a slight adjustment in speed, therefore, a large speed range is normally not required. A range of 2:1 or 3:1 is more than sufficient.

The normal power range for sub-synchronous cascade systems used in industry is from about 500 kW to 10 MW with motor nominal speeds of 1500 min^{-1} or below. For special applications, similar converter systems have been built up to 60 MW. Motor cooling systems are identical to those of normal slip ring motors running at a constant speed.

Operating Characteristics

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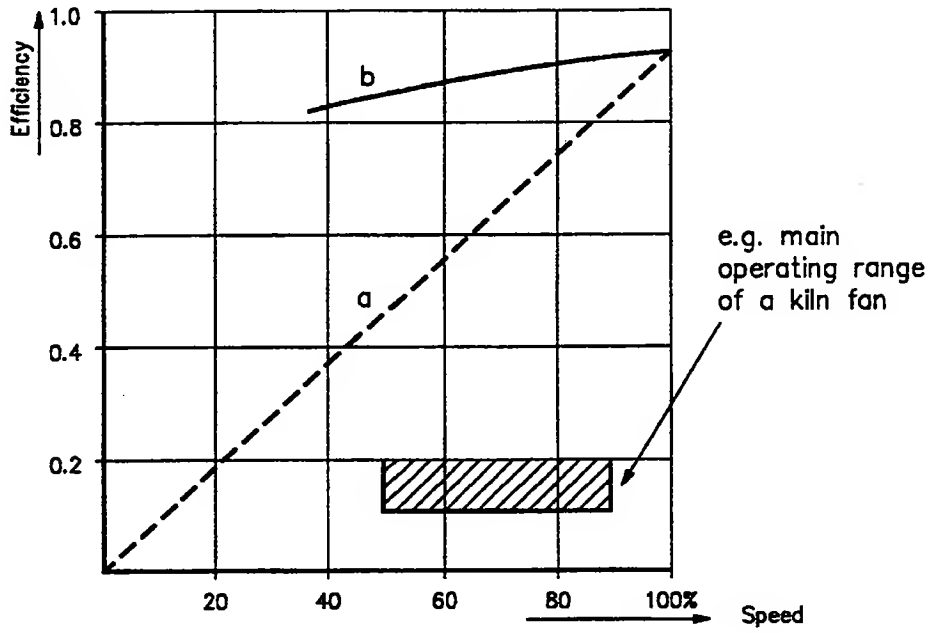
A sub-synchronous cascade drive needs a starting resistor. The variable speed range is very much reduced compared to a DC drive. No oversynchronous speed can be reached and only 1-quadrant operation is possible, i.e. motoring in one direction only.

Every converter requires reactive power. A larger drive systems has a higher demand for reactive power, which has to be considered and compensated. With the sub-synchronous cascade drive system, the reactive power demand increases with increasing speed range. Therefore, the variable speed range should be kept as small as possible. The compensation system has to be designed on an individual basis and should be optimized for the normal running speeds of the motor. Furthermore, the harmonic currents, created by the static converter, have to be considered during the design of the power factor compensation system. In a modern installation, the filter-circuits cover both aspects, resulting in a combination network of reactors and capacitors instead of capacitors only.

The efficiency of the total variable speed drive system is not as high as that of a slip ring motor alone due to more power components being involved in the former. The overall efficiency over the speed range is, however, much better than for example controlling the air-flow with a radial vane damper at constant motor speed or at variable slip ring motor speed using permanently connected resistances in the rotor circuit.

Fig. 41: Typical speed/efficiency curve of a sub-synchronous cascade drive



a : variable speed by rotor resistance
b : variable speed by sub-synchronous cascade

Application

Especially large plants require large fans where DC drives are not feasible as the power/speed ratio exceeds the typical DC motor frame size. Here, the sub-synchronous cascade system offers an interesting alternative.

A 2000 t/d plant, for example, needs a kiln fan of 1700 kW at 1500 min^{-1}. Large fans in the cement industry can have a range of up to 5 MW. Therefore, this type of drive will be seen in our industry more often since it meets all requirements in terms of controlability, operating behaviour and economy.

Furthermore, any existing slip ring motor can be converted into a variable speed drive by adding a sub-synchronous cascade converter system. On the other hand, every cascade system can run at rated motor speed without the static converter, e.g. during a fault in the electronic regulation part. Leaving the mechanical flow control device installed will be of advantage!

The sub-synchronous cascade drive is, therefore, a technically and economically favourable system for large fans requiring variable speed du to process parameters.

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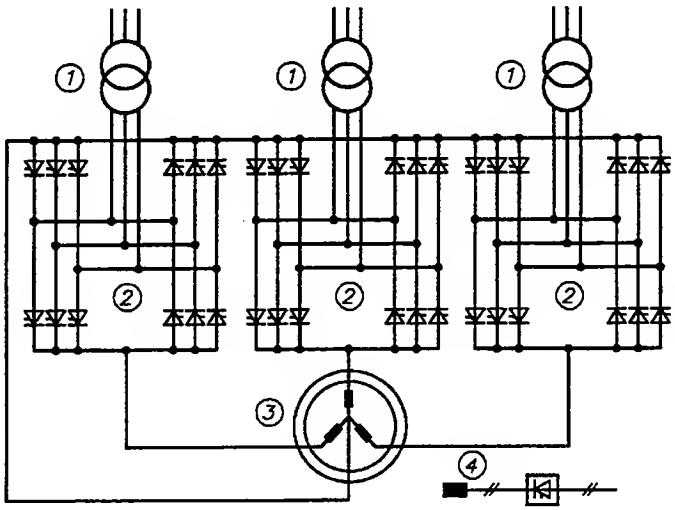
4.2.4 AC drive with synchronous motor

4.2.4.1 Synchronous motor with cyclo-converter

Fig. 42: Converter schematic used in conjunction with the ring motor (gearless mill drive)



1) converter transformer
2) two converters in anti-parallel three-phase bridge connection
3) synchronous motor
4) exciter winding

Each motor phase is connected to the feeding power system via two static converters arranged in an anti-parallel three-phase bridge network. A low frequency output voltage is delivered by the converters by means of phase angle control. At a system frequency of 50 Hz, the maximum attainable output frequency is approx. 20 Hz. With this drive system, four-quadrant operation, i.e. reversal of the direction of rotation and regenerative braking, is possible without any modification. This system corresponds fully to a four-quadrant DC drive. A high starting torque and almost sinusoidal current results in particularly favourable characteristics at low speeds.

This system is well-suited for the substitution of large DC drives, e.g. for conveying machinery, in rolling mills or as propeller drives for ice breakers and mine winders, especially when the DC motor can no longer be employed because of ambient conditions, maintenance costs or power limits.

The system covers a range from 1 to 20 MW.

4.2.4.1.1 Application

The cement industry uses this system only for large (cement) mills, avoiding the gear and, therefore, saving space and building cost. Of course, this is not of equal importance all over the world and its technical complexity can be a drawback in many third world countries. Therefore, this drive system is not very often selected. But in other industries, it will replace in the near future more often the large DC machines in the MW-range.

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4.2.4.2 Synchronous motor with intermediate circuit converter

Fig. 43: Basic circuit, 6-pulse

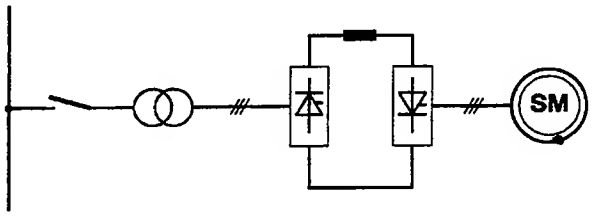
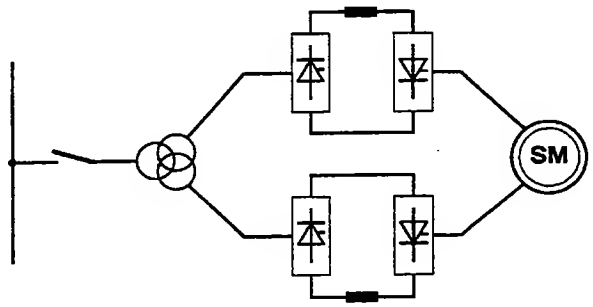


Fig. 44: Basic circuit "12-pulse"



Rectifier and inverter, "12-pulse" with two motor windings displaced by 30° el.

Static converter in parallel connection

The circuit is generally called a converter-fed synchronous motor and consists of a controllable rectifier, a smoothing reactor and an inverter. In these designs (Figures 43 and 44 above), rectifier and inverter have to be sized for the full motor power, compared to the sub-synchronous cascade, where the converter has to cope with the rotor slip power only. The commutation from one phase to another of the inverter is dictated by the terminal voltage of the synchronous machine. This natural commutation does not need any additional circuit like e.g. the forced commutation with the converter type for squirrel cage motors.

This type of converter is suitable for 4-quadrant operation and can cover a full speed range like a DC drive variable speed system.

The 6-pulse scheme is normally used for power of 1 to 5 MW. For larger systems, the harmonic currents lead towards 12-pulse configurations due to motor and line side problems. Modern synchronous motor drives have a brushless excitation system. An auxiliary asynchronous machine, integrated into the synchronous motor, supplies its power through a rotating diode rectifier to the DC field winding thus avoiding trouble causing slip rings.

Converter frequencies of up to 120 Hz can be realized driving a two pole synchronous motor up to 6000 min^{-1} at almost any power. Systems of 30 MW have been built and projects of 50 MW are being studied.

4.2.4.2.1 Application

The main applications of the converter-fed synchronous motor for pumps, extruders and compressors, where a precise speed control over a wide speed range is important.

These drives are not installed in the cement industry, as the existing type of machinery does not specifically require a converter-fed synchronous motor system.

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4.2.5 Electronic smooth-start for three-phase motors (soft starters)

The simplest and cheapest way to start a three-phase motor is full-voltage, across the line starting, and that method should be used whenever feasible. But there has always been a need in some applications to limit the locked-rotor inrush current to the motor, control motor starting torque, or both.

Control of starting torque and acceleration is often required to protect the dirven-load. For example, it might be necessary to control acceleration and starting torque of a conveyor motor to prevent shock damage to system elements and damage to products on the conveyor.

Fig. 45: Electronic soft-start for a three-phase motor

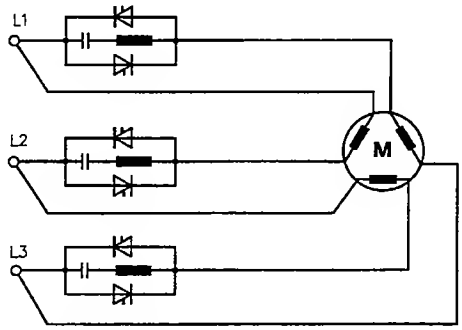
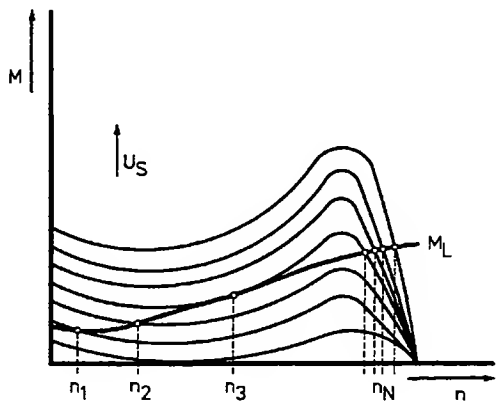


Figure 46 shows the course of torque with a smooth-start for a three-phase motor. The starting procedure begins by 20 to 40% of the nominal voltage. During the adjusted starting time, the stator voltage will be increased to 100% through the control of the firing-angle of the thyristor-controllers.

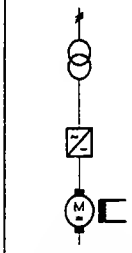
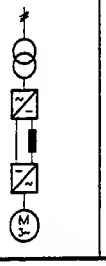
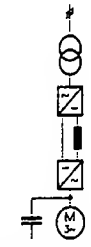
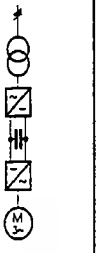
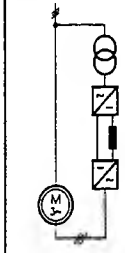
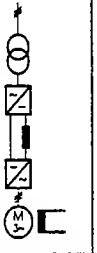
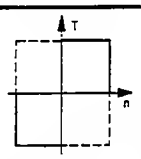
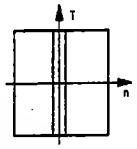
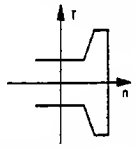
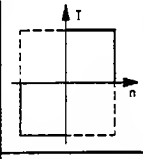
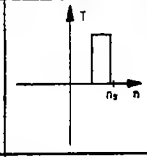
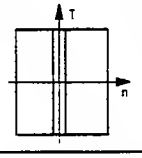
The motor runs up along the load characteristic Ml, whereby torque-shocks will be avoided. The speed increases linear during the starting time from 0 to the nominal speed of the motor. After the starting procedure, when the motor runs with nominal load, the thyristor will be fully conducted.

The electronic smooth-start works similar to the hydrodynamic coupling (see chapter 4.3.3), but it has the decisive advantage that the starting time and the starting torque can be easier adjusted to the individual operating conditions.

Fig. 46: Torque/speed diagram



Summary of large variable speed drive systems for the cement industry (>1 MW)

Drive system	Direct Current Drive	Current-source inverter-fed induction motor	Load-commutated inverter-fed induction motor	Voltage-source inverter-fed induction motor (PWM)	Wound-rotor induction motor with subsynchronous converter cascade	Converter-fed synchronous motor	Drive system
System diagram							System diagram
Type of machine	*Direct Current Motor*	*Cage induction motor*			*Wound-rotor induction motor*	*Synchronous machine*	Type of machine
Operating range T: torque n: speed							Operating range T: torque n: speed
Typical power range	up to 2000kW	up to 2000kW	500 ... 5000kW	800 ... 5000kW	500 ... 5000kW	1000 ... 10000kW	Typical power range
Typical speed range	0 ... 100%	2 ... 100%	65 ... 100%	0 ... 100%	50 ... 98% of synchronous speed	(0) ... 10 ... 100%	Typical speed range
Significant properties	Single-motor drive of medium power Good performance also at low speeds	Single-motor drive of medium power Applicable normally for loads with squared torque/speed characteristic	Single-motor drive with nearly sinusoidal motor current and voltage Applicable normally for loads with squared torque/speed characteristic	Single- or multi-motor drive Good performance also at low speeds Power-factor to line nearly unity	Slip-power recovery Economic for small motor-speed-control-range High starting torque Suitable for retrofitting of existing slipring motors	Single-motor drive of high power High speed possible	Significant properties
Suitable for	Fan, (Kiln)	Fan	Fan	Fan, (Kiln)	Fan	Fan	Suitable for
Price [%] Base: 1500kW/1000RPM	100%	105%	110%	110%	85% (without filter)	140%	Price [%] Base: 1500kW/1000RPM

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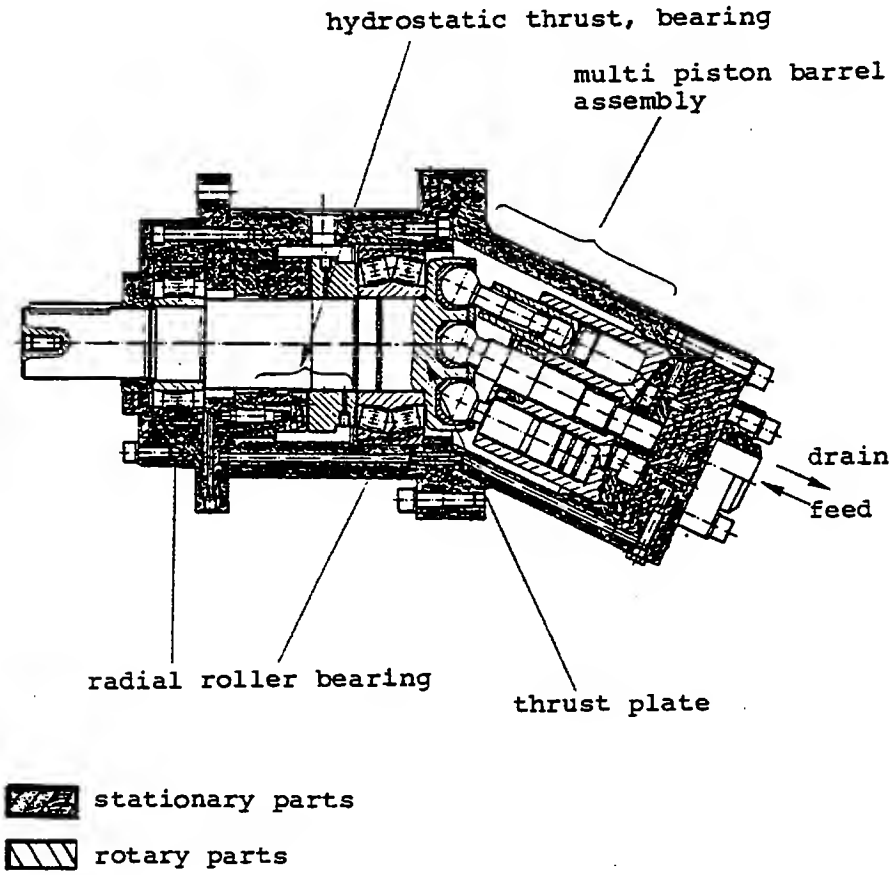
4.3 Hydraulic variable speed drive system

4.3.1 Hydrostatic drives

Fig. 47: Scheme of a hydrostatic motor



The hydrostatic motor is connected to a hydraulic high pressure pump system. The oil feed and drain pipes are located on the opposite side of the drive shaft.

The unit can be subdivided into two parts i.e. the drive shaft bearing part and the torque creating multi-piston barrel assembly. Due to the inclined mounting of the piston barrel assembly, continuously varying cylinder volumes exist during one revolution. The pistons, therefore, perform strokes similar to those of an automobile-engine. The high-pressured oil enters and leaves through slots acting as valves. The piston forces react on the thrust-plate, causing the cylinder barrel and attached shaft to rotate with a torque proportional to the supplied oil pressure. The rotation speed of the motor shaft changes proportionally to the supplied oil-flow.

The hydrostatic motor parameters are:

- oil pressure → torque
- oil flow (quantity) → speed

These two variables are supplied to the motor by a hydrostatic pump driven by a prime mover and the associated speed control regulation devices.

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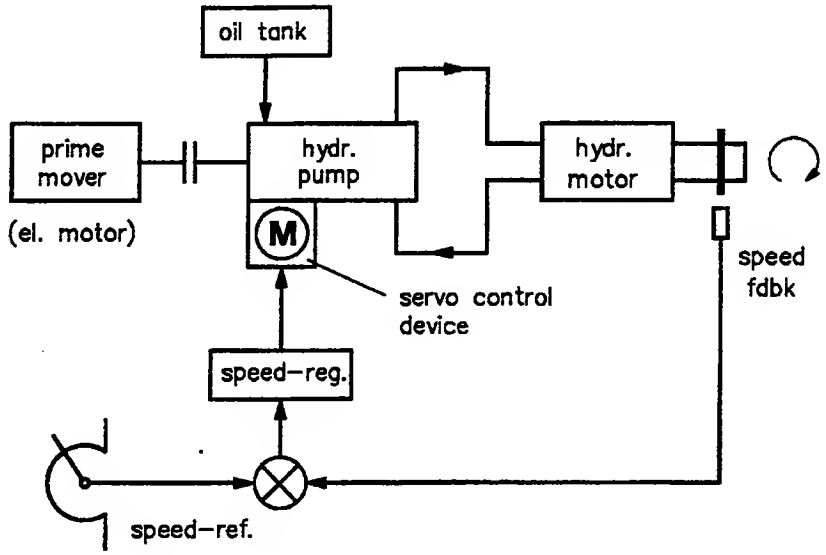
Fig. 48: Main components of a hydrostatic variable speed drive system



The hydrostatic pump basically consists of the same elements, only the multi-piston barrel assembly does not have a fixed inclined angel. The complete piston unit is designed to swivel about a transverse trunnion axis. At zero degree deflection, all pistons remain axially at the same position, i.e. do not perform any stroke and, therefore, no oil-flow is created. Moving the piston unit out of the straight centre line, the pistons start to execute a stroke proportional to the deflection angle. An oil flow is established and the motor starts to turn at a speed proportional to the deflection angle.

Fig. 49: Plan view of a hydrostatic pump with piston unit at 0 degree deflection

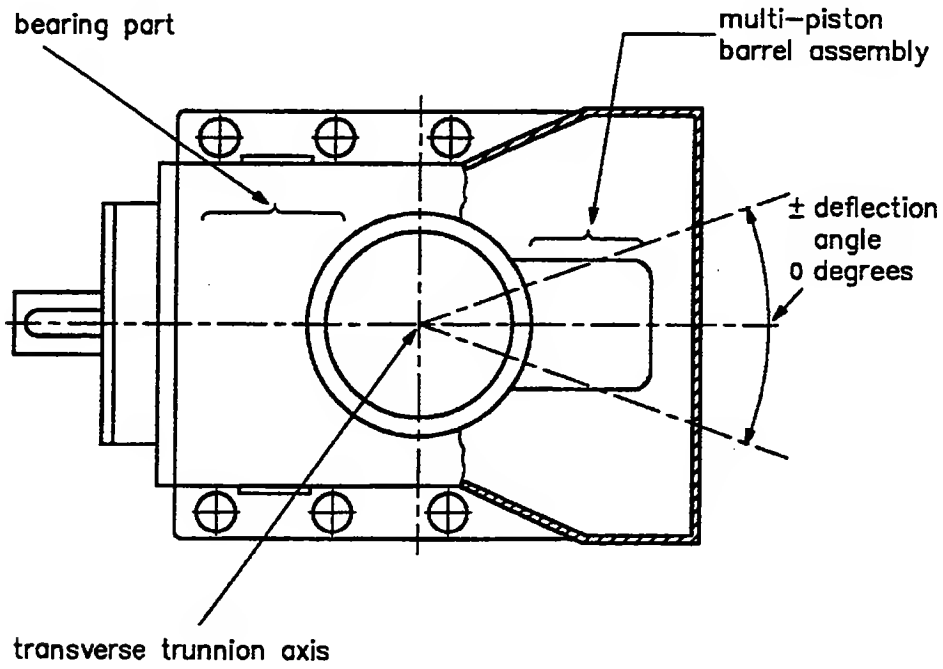


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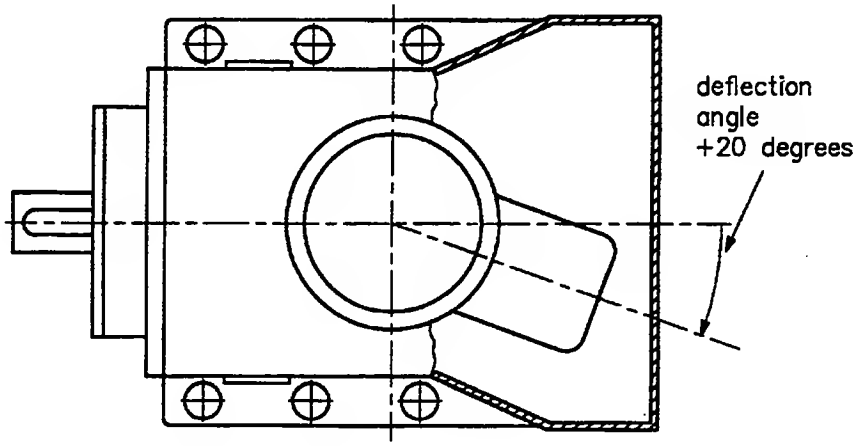
Fig. 50: Plan view of a hydrostatic pump with piston unit at +20 degrees deflection



This is a short introduction to the hydrostatic operating principle. Many additional accessories like valves, oil cooler, operating protection, torque limiters, emergency shut-down, etc. are not explained, but are available and together with pump and motor form a complete drive system. Good operating behaviour and controllability therefore make it truly comparable to electrical variable speed drive systems.

4.3.1.1 Application

The hydrostatic drive system is widely known and used in the cement industry since approx. 1965. Drives requiring variable speed and high starting torque have been equipped with the above system, e.g. for crusher-feeders, grate coolers, etc. from a few kW-approx.-200 kW. Good reliability and low maintenance of this hydromechanical system make it an alternative to electrical drives.

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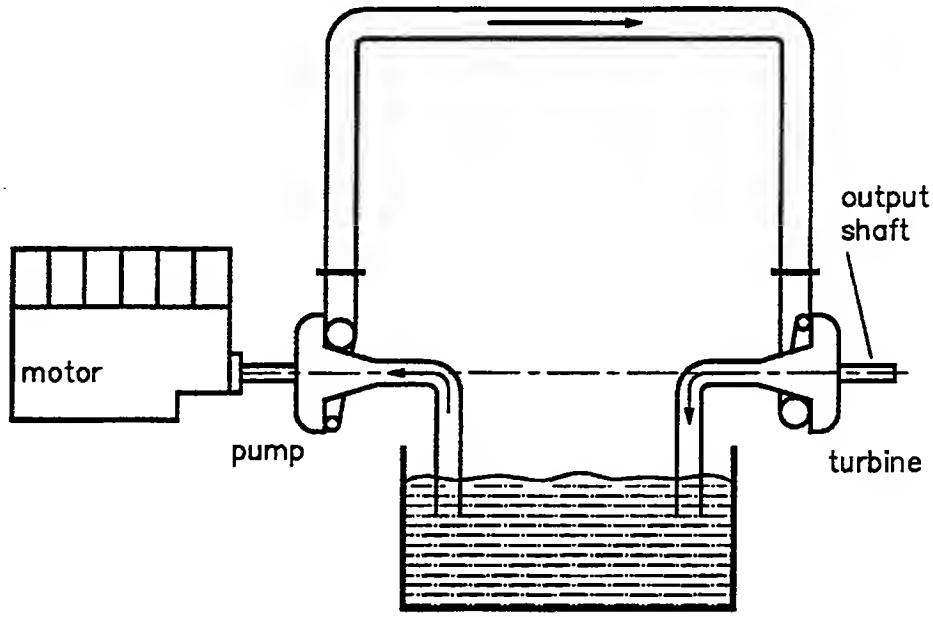
4.3.2 Hydrodynamic drives

The construction of a hydrodynamic drive is similar to that of a turbine, where rotor and stator are not in direct mechanical contact, but are coupled through a liquid or gaseous medium. A prime mover (e.g. electric motor) drives a hydraulic pump. The medium set in motion by the pump is feeding a hydraulic turbine which at its output shaft drives the coupled machine requiring smooth-start or variable speed.

Fig. 51: Operating principle of a hydrodynamic coupling



Pump and turbine are brought together and built into one common casing. This combined unit is then called hydrodynamic coupling or turbo coupling.

In most commercially used types of couplings, the medium which transports the kinetic energy from the pump to the turbine is oil. The quantity of oil represents a very important parameter since the transmitted torque and speed depend on the filling degree of the coupling.

Therefore

- constant oil volume → coupling
- variable oil volume → variable speed drive

In order to achieve a variable oil volume, technical means of adding to and subtracting from the oil volume have to be established during operation at any speed. This oil quantity e.g. can be varied with an adjustable sliding scoop tube. In this way, the power transmitted by the hydrodynamic coupling can be adjusted and stepless speed regulation of the driven equipment in accordance with load demands is provided.

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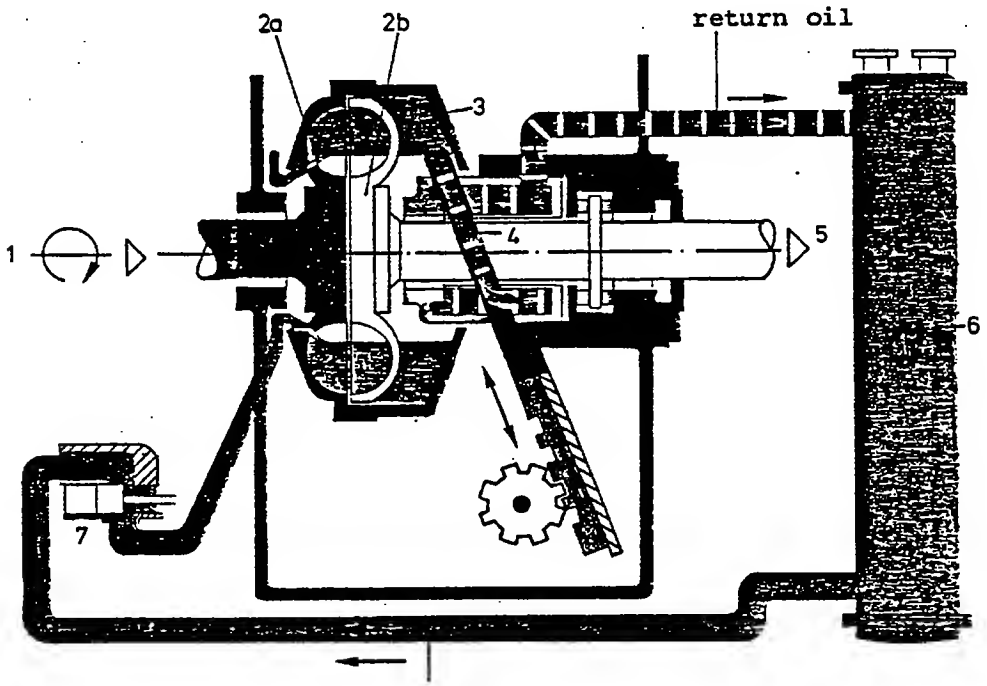
Fig. 52: Schematic diagram of a variable speed hydrodynamic coupling



1) prime mover (e.g. electric motor)
2) hydrodynamic coupling:
   a : pump, b: turbine
3) oil level in the casing during operation
4) adjustable sliding scoop tube (up and down)
5) output shaft with variable speed depending on scoop tube position
6) heat exchanger
7) oil flow control valve

4.3.2.1 Application

The hydrodynamic coupling itself behaves according to the propeller law. The output torque increases with the square of the input speed. The coupling, therefore, is well-suited to drive machines with parabolic torque load characteristics such as centrifugal pumps and fans with a regulating range of not more tan 4:1. Machines with a constant torque load characteristic can be used only with a speed range of not more than 3:1 and have to be oversized in most cases. Dynamic response of the variable speed drive system is much slower than e.g. with a DC drive since this depends on the position regulator of the scoop tube.

On the other hand, the hydrodynamic offers very interesting benefits since very large units at very high speed are quite normal. The size ranges from a few kW (approx. 20 kW) up to 8 MW at 12,000 min^{-1} or 60 MW at 5,000 min^{-1}. Especially the units with extreme speed requirements (very high or low) operate either at the input or output with multiple gear stages.

Many of those very large and high speed units are installed in nuclear and thermal power plants as boiler feed pumps. Others, including the cement industry, use some of the wide variety of hydrodynamic variable speed drive systems too.

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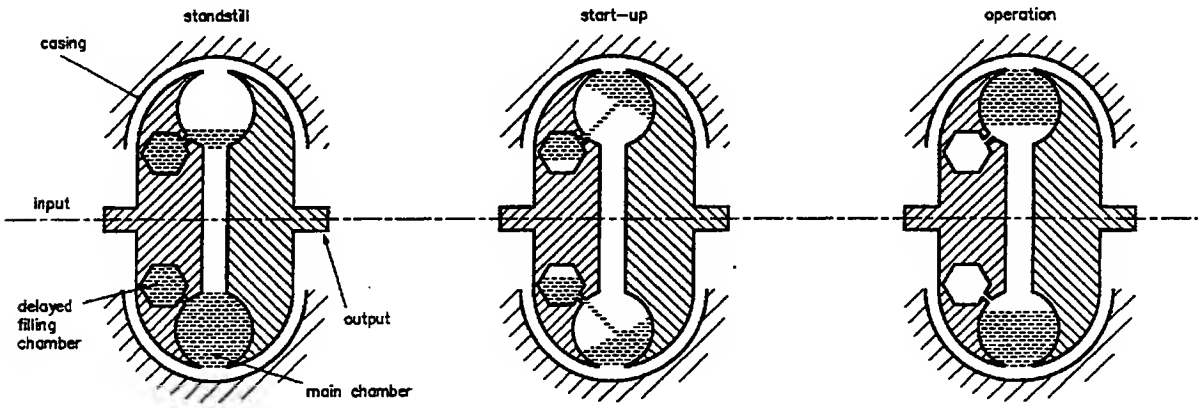
4.3.3 Smooth-start by turbo couplings

The operating principle of the hydrodynamic or turbo coupling is described in para 4.3.2 above. The main feature of such a device is not the speed regulation, but the soft start and shock absorbing characteristic. The final output speed at the end of the start-up sequence is, therefore, always similar to the input speed. The plain coupling has no scoop tube. One of the special features is the retarded filling of the oil chamber after stand-sill. During start-up, the integral delayed filling chamber retains part of the operating fluid from the coupling working chamber, resulting in a reduced torque transmission until all the oil has reached the main chamber. This allows the electric motor to start-up under virtually no load.

Fig. 53: Principle of operating of the delayed filling chamber



The total oil volume is also a measure to control the maximum transmittable torque. In a multi-motor belt system, e.g. load balance can be adjusted by the individual oil filling.

4.3.3.1 Application

The family of hydrodynamic couplings is well-known in the cement industry. Soft or controlled start-up can be achieved by a slip ring motor and the corresponding size of the rotor resistor. A squirrel cage motor and a hydrodynamic coupling perform the same task more elegantly. Therefore, heavy starting machines like long belts, crushers, fans with a large external mass are often equipped with hydrodynamic couplings as well as mechanical items which do not permit excessive starting torque stresses like chains on bucket elevators. This type of coupling is available from 1 kW up to approx. 1,500 kW at nominal input speeds of 3,000 min_{-1} for the small units and 1,000 min_{-1} for the larger ones.

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5. CRITERIA FOR ASSESSMENT

5.1 Specifications

In the specifications the operational requirements, standard of manufacture and the stipulated reliability have to be summarized by the user. Apart from the technical details generally given and the ambient conditions, a number of other factors are important for variable-speed drives:

- Starting and slow-running characteristics
- Speed/torque characteristic of the driven machine and of the selected drive system
- Range of operating speed and accuracy
- Suitable means of protecting the installation, which does not lead to unnecessary stops in the event of short interruption of the supply
- Definition of the maximum admissible harmonic current content on the network and of the filter equipment
- Extent to which the power electronics is proof against short circuits
- Cooling for the motor and converter
- Redundancy requirements

5.2 Reliability

The main objective when using any drive system, be it mechanical or electrical, is to ensure high availability and reliability for the installation as a whole, with minimum maintenance. The choice of system can to a large extent be influenced by the qualifications of the local staff. This does not only apply to electric drive systems; hydromechanical systems today use components and technologies which can no longer be regarded as common knowledge for the average mechanic. On the other hand, the electrical industry, by utilizing high-power thyristors and by simplifying the control electronics, is making an attempt to keep the complexity of the systems within reasonable limits.

5.3 Efficiency

As far as running costs are concerned, the efficiency at the most frequently used operating point is a factor of decisive importance. In the foreseeable future energy costs will continue to increase at a faster rate than investment costs. Therefore, when planning installations, it is necessary to make a comparison of the investment cost with the operating costs of the potential drive systems. This trend should be taken into account in the appropriate manner during the evaluation.

The efficiency figures quotated by the manufacturers of drive systems have to be examined with great care, as in most cases they only provide an efficiency curve for full load of the most significant drive component, e.g. the motor. Information on partial load is difficult to obtain, but in most cases the values are below those quoted.

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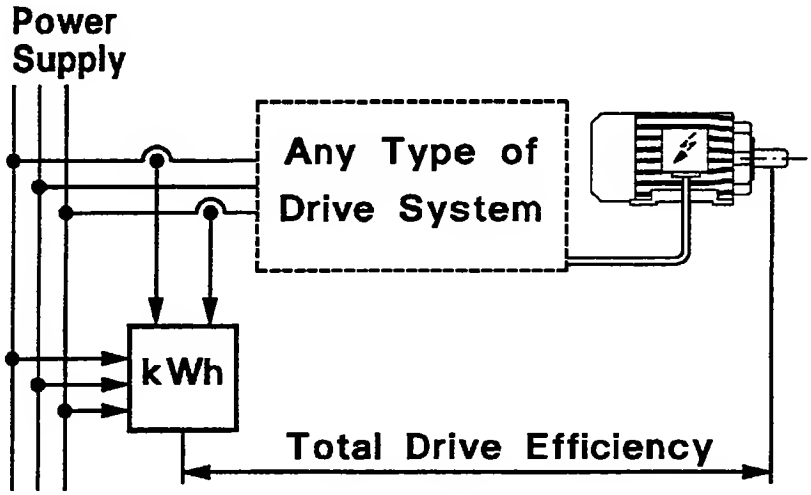
5.3.1 Definition of total drive system efficiency

Efficiencies of individual drive components do not define the total system behaviour. For comparison, it is therefore essential to establish meaningful and measurable limits, which define the border lines of efficiency for one total drive system. On the one hand, the power drawn from the network is measured and, on the other, the mechanical power imparted at the variable speed shaft. All components located between these two interfaces are appropriately to be taken into account for any system including auxiliary power consumption e.g. for cooling or ventilating purposes.

Fig. 54: Definition of total drive system efficiency



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6. CONCLUSIONS

Especially when high powers are involved, existing and newly planned installations should be closely examined to determine whether they are not equipped with drives or control systems involving unduly heavy losses. Using up-to-date techniques, this is a field where it is possible to achieve substantial savings in running costs.

It may be taken for granted that the present trend towards variable-speed drive systems fed by static converters will continue in the future. The development of power and control electronics also allows one to expect that the outlay for variable-speed a.c. drives will decrease further. The high effieciency of electric drives will therefore make their utilization increasingly interesting. The trendency to seek an alternative to d.c. motors and thus to get away from their commutator problems, is unmistakeable. Opportunities for this are provided by hydromechanics and three-phase a.c. systems. But even these systems require a certain amount of maintenance. It is therefore advisable to analyse all alternatives very closely. No matter how high the efficiency may be, it loses all its significance if the system fails only a few times! It will therefore be necessary to weigh reliability and efficiency very thoroughly, one against the other.

Some typical applications of variable-speed drive systems were dealt with in this session. Unfortunately there are not generally valid solutions for the various applications in all countries.

7. MESSAGES

- Be energy conscious when selecting variable speed drives
- Consider alternatives and new technologies
- Analyse new technologies very thoroughly especially with respect to reliability and efficiency

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Plant Automation

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My Profile:

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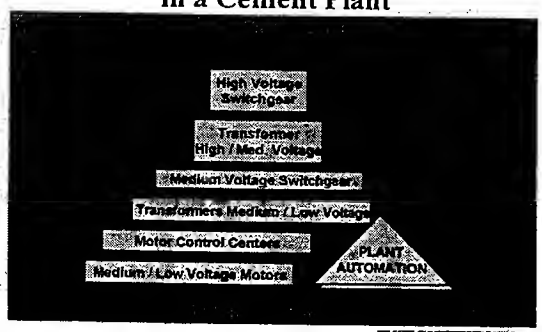
Structure of Electrical Systems in a Cement Plant



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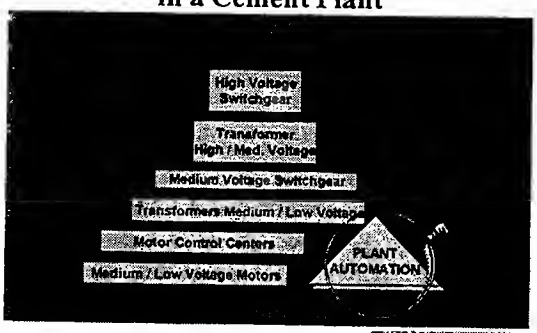
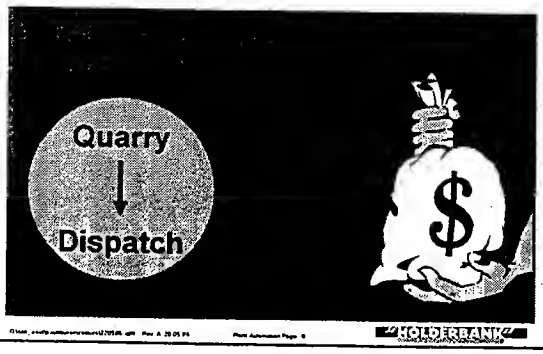
Structure of Electrical Systems in a Cement Plant

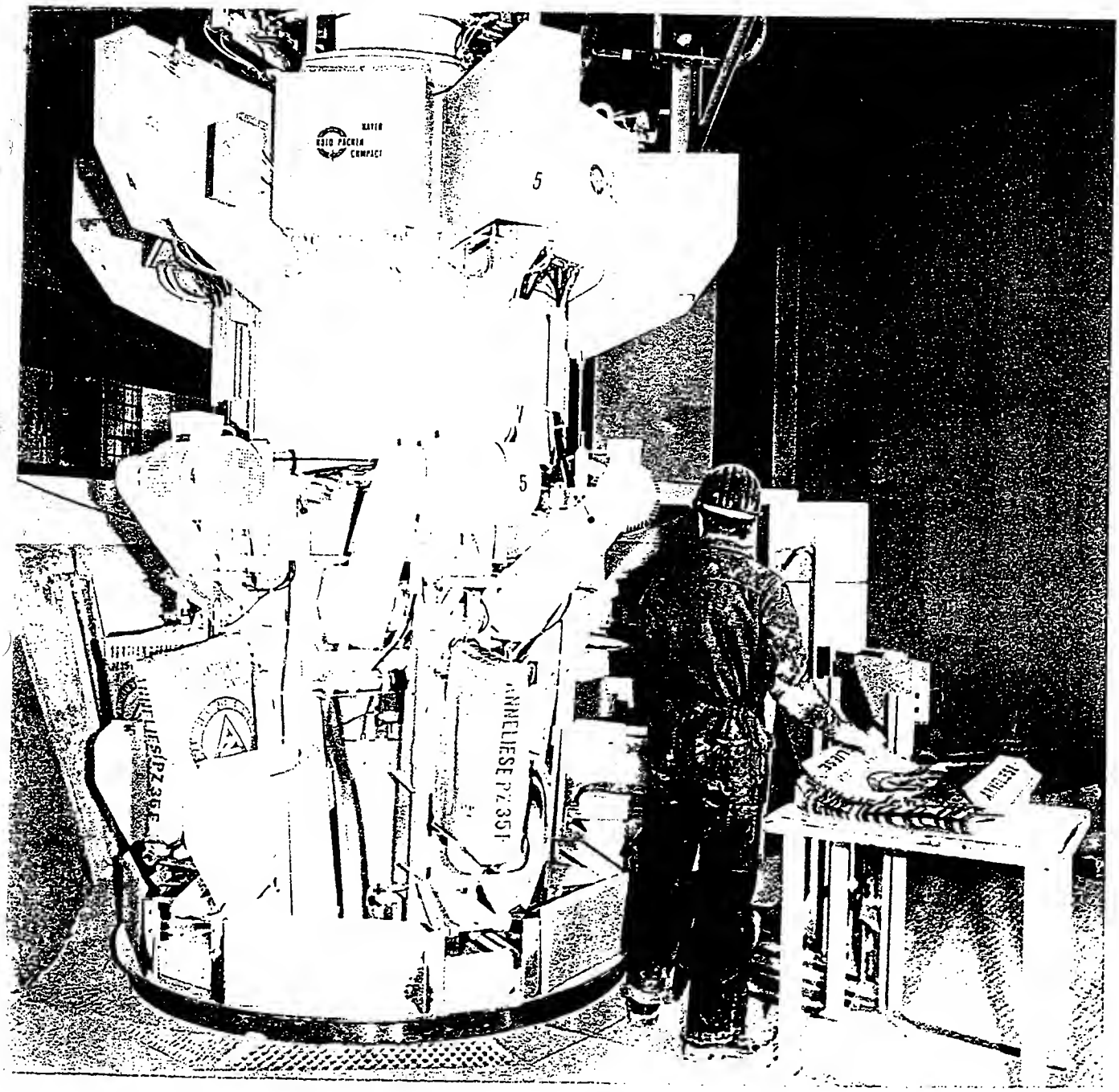


Plant Automation Subjects:

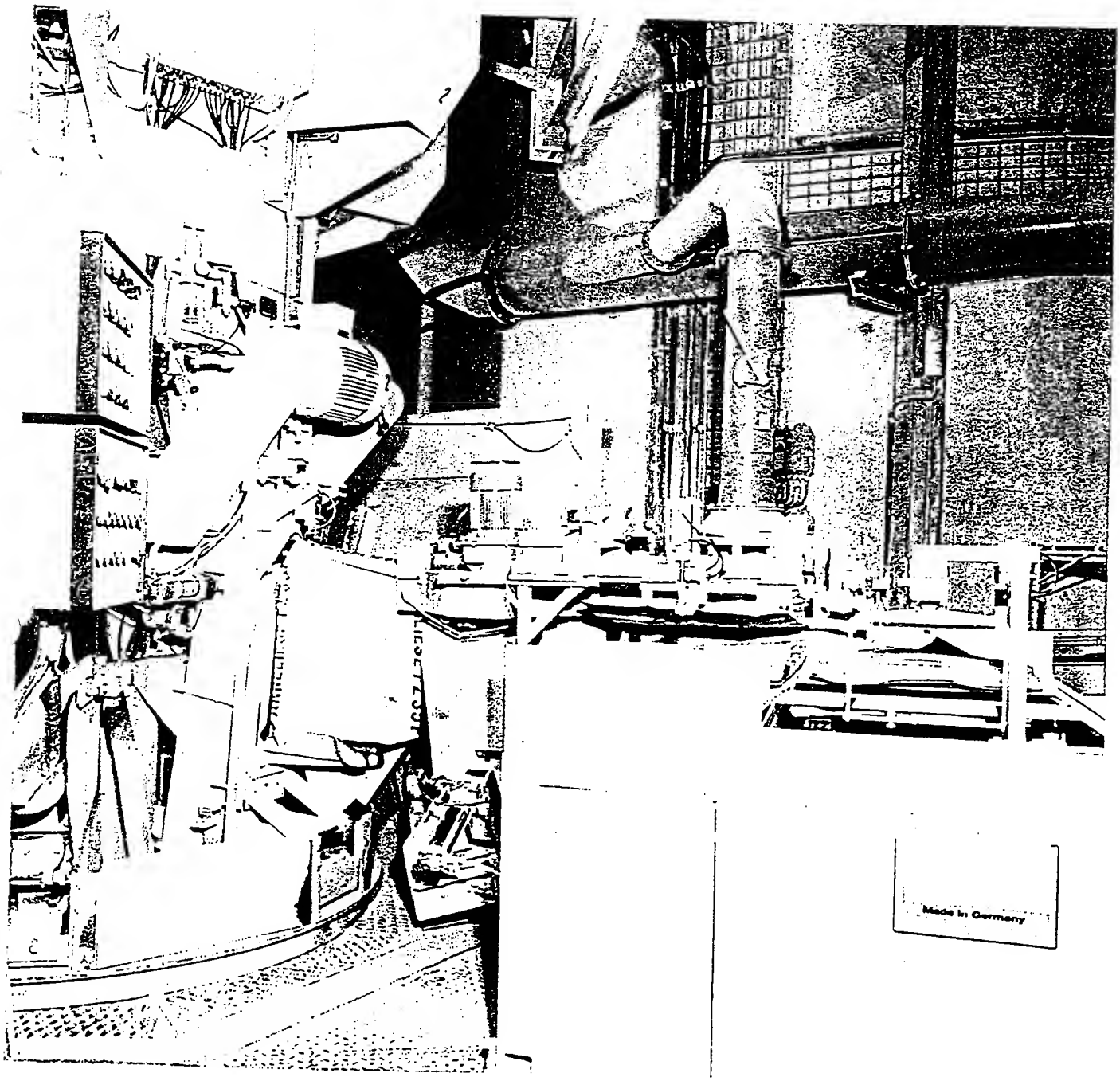
Definition of Automation



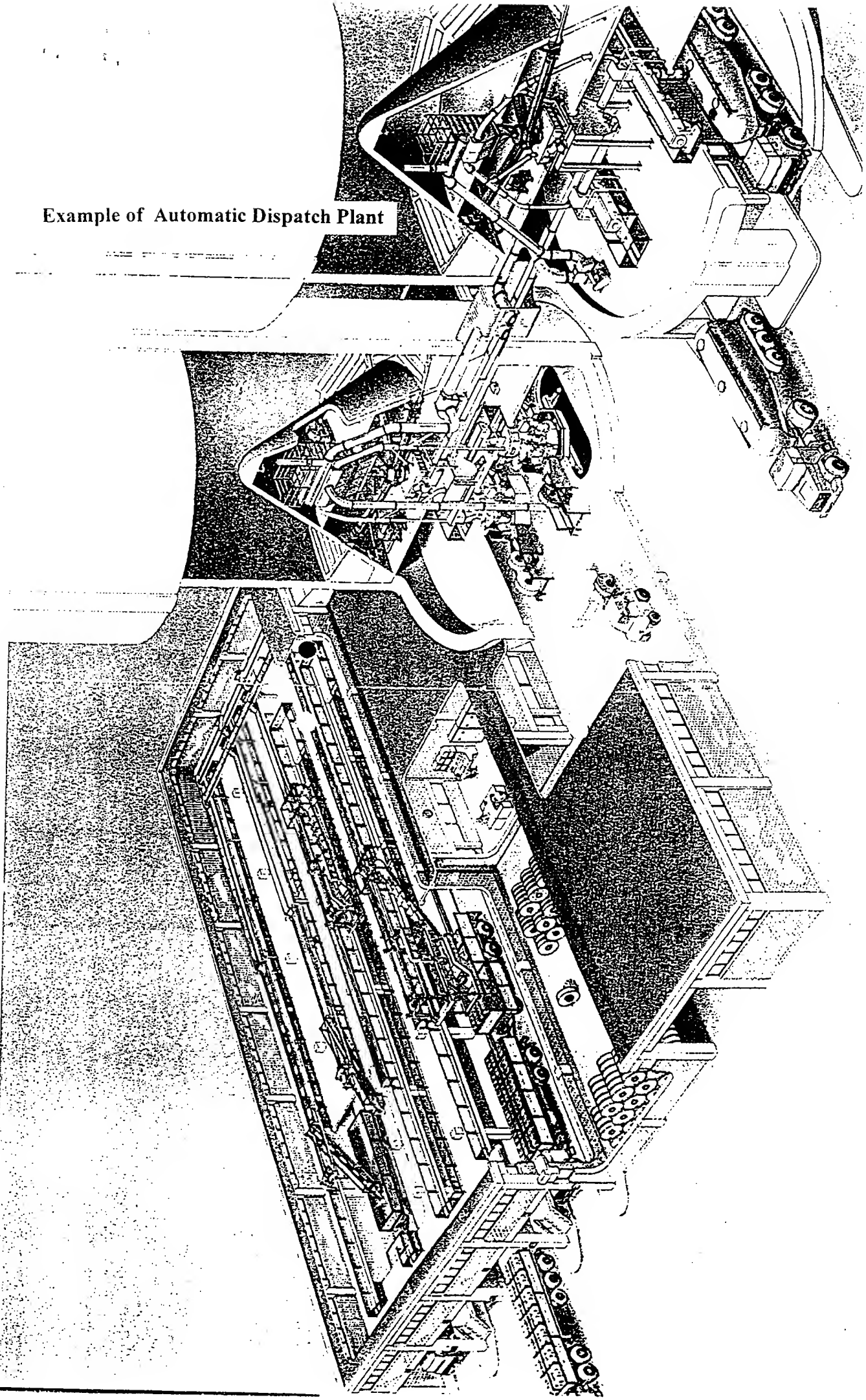
Example of Manual Bag Loading



Example of Automatic Bag Loader



Example of Automatic Dispatch Plant



Benefits of Automation ?

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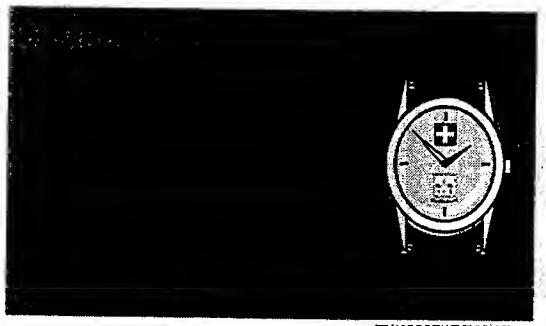
Benefits of Automation



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Benefits of Automation

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Benefits of Automation

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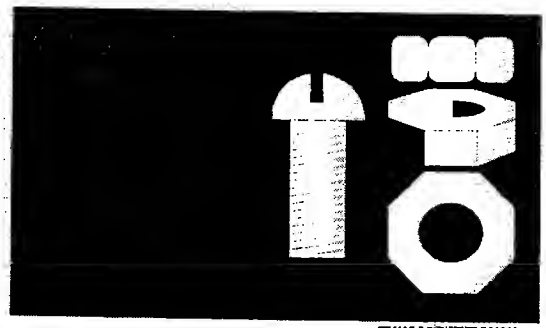
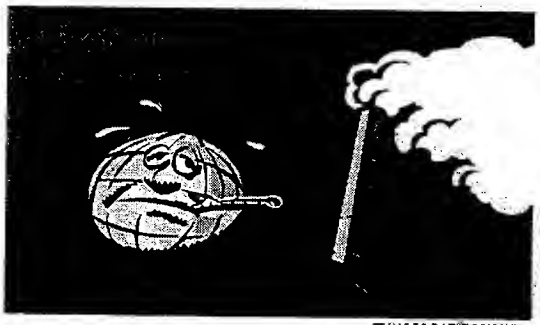


Benefits of Automation

Benefits of Automation



Benefits of Automation

Benefits of Automation

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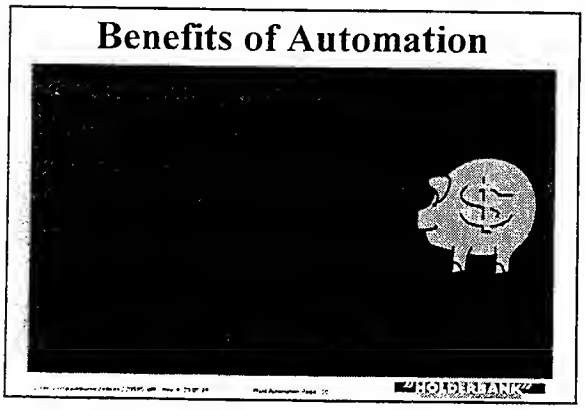
Benefits of Automation



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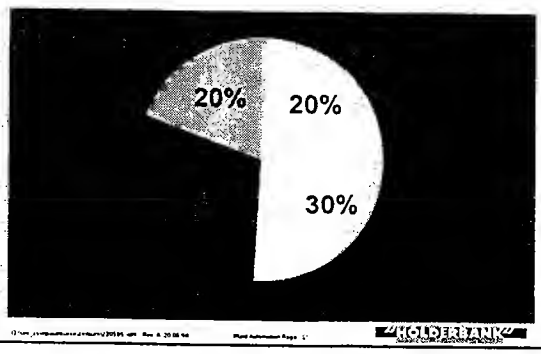
Allocation of Production Costs



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Production Costs

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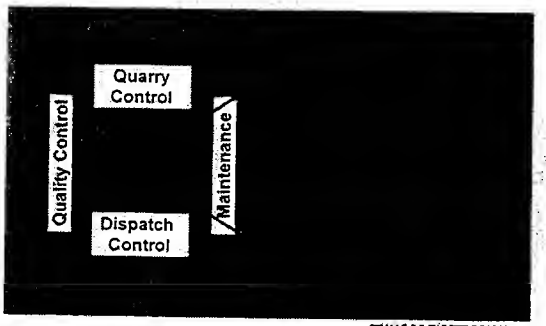
Areas of Automation in a Cement Plant



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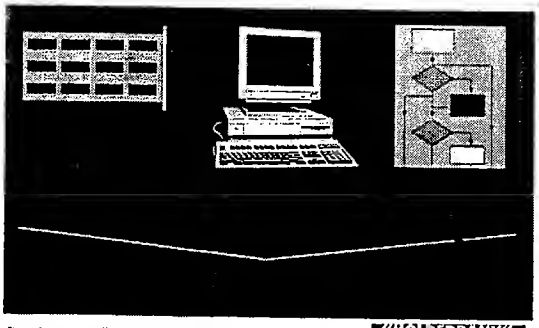
Automation ?



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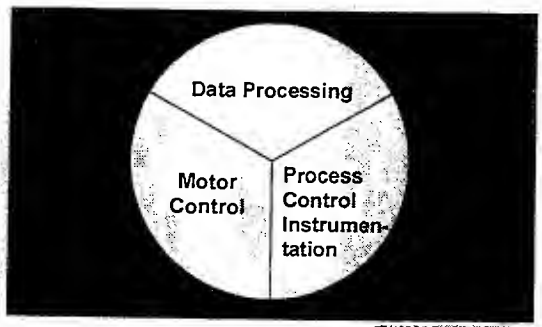
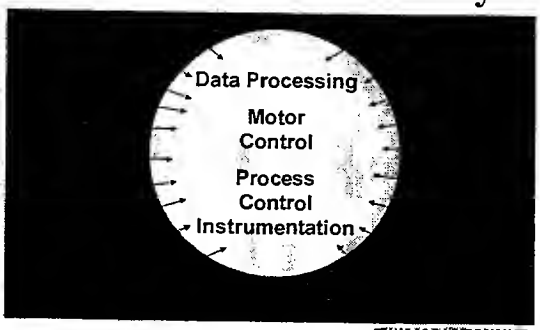
History of Automation Systems

Year	Complexity of Control.	Plant Size
1940	Local control of individual machine, local pneumatic/mechanical indication.	50 Motors, 20 Instruments
1960	Central control rooms, sequence control of motors (by relays), some remote indication of instruments.	200 Motors, 70 Instruments
1975	CCR's decentralized programmable controllers, monitoring- or process computers.	400 Motors, 150 Instruments
1985	Centralized controllers with remote I/O's, graphic displays, communication links.	800 Motors, 300 Instruments
1990	Distributed control systems, optical cables, automated documentation tools, integration of management information.	1000 Motors, 500 Instruments
1995	Optimized haman-machine interface, Fuzzy logics, high level control.	1000 Motors up to 600 Instr.
1998	Field Bus technology for analog and digital sensors/actors.	
Future	unattended factory ??	more Instruments

Traditional Automation



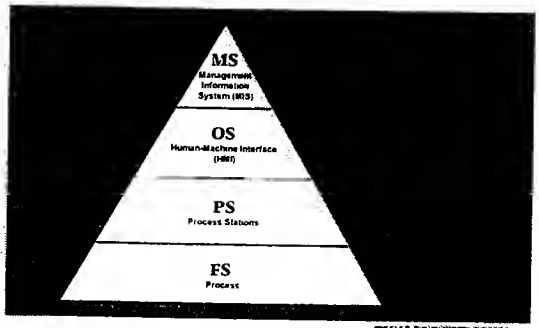
Process Automation today



Automation Today

- **Automation no longer refers to a control system of an industrial plant or facility**
- **It now refers to the entire electrical system which needs to operate and communicate as one integrated system from the main HV substation all the way to the motor and the device**
- **No information islands in the system**

Structure of Plant Automation



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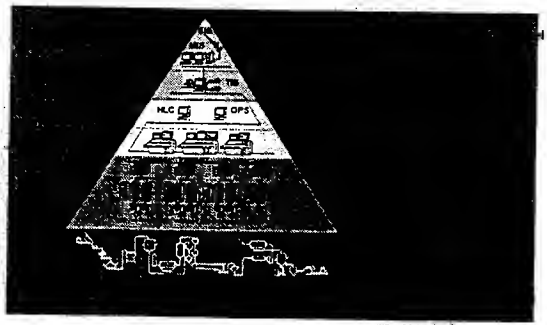
CIM - PYRAMID



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The Field Level consists of:

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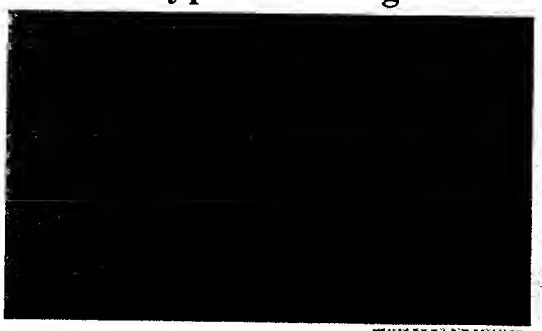
Per Production Line

Today		Future
6'000	Digital Inputs	8'000
1'300	Digital Outputs	2'000
350	Analog Inputs	1'000
40	Control Loops (PID)	70
10'000	Alarmpoints	15'000
150	Calculated Values	500
1'000	Archived Values per Minute	4'000

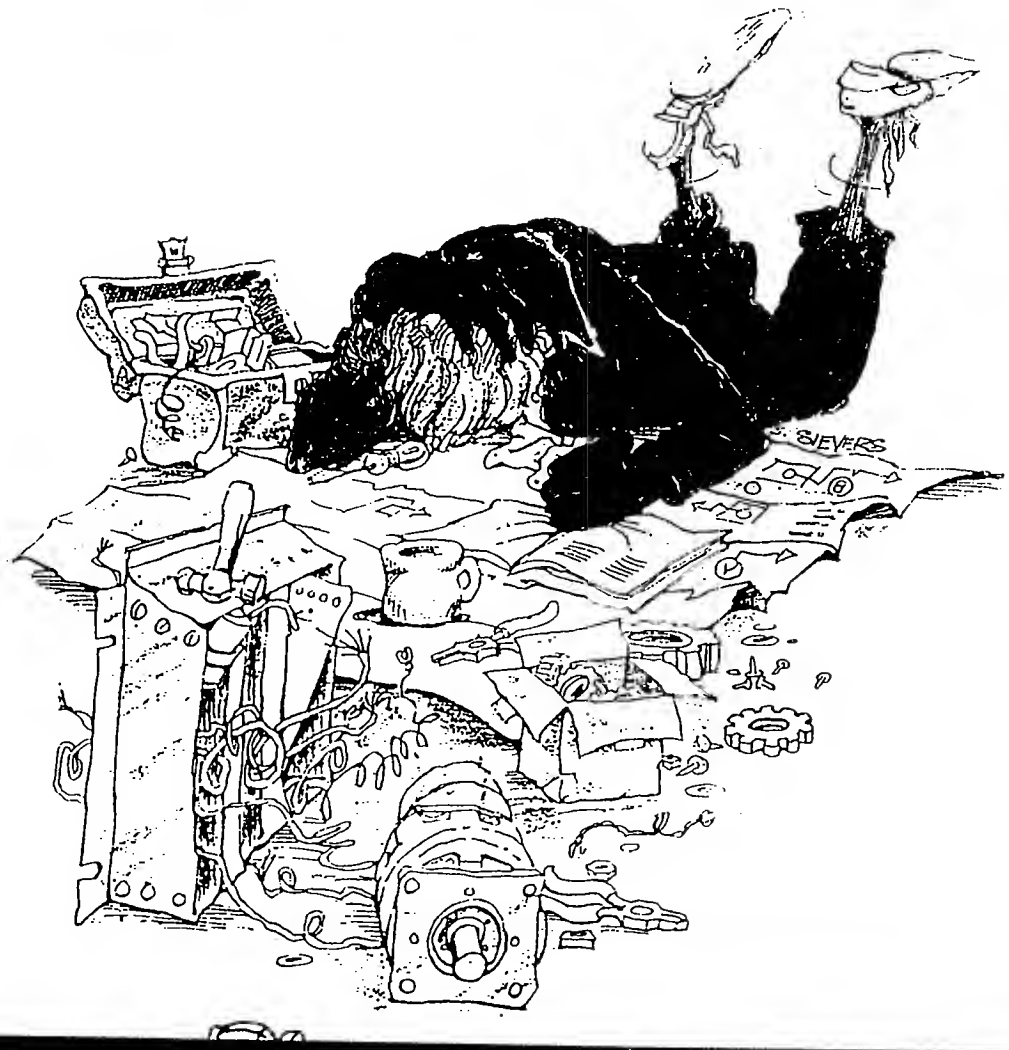
100'00....150'000 Connection Points

Typical Wiring



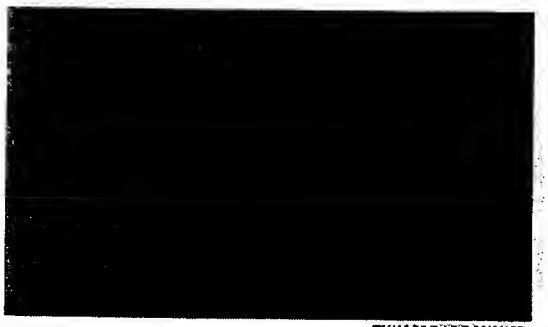
Documentation

Documentation

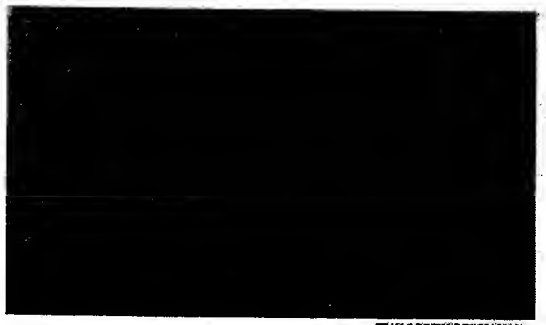


Tasks of the Control and Instrumentation Level

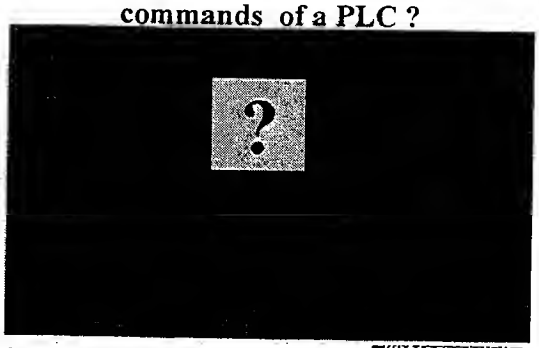
Who can all do this?



PLC ? What is it ?

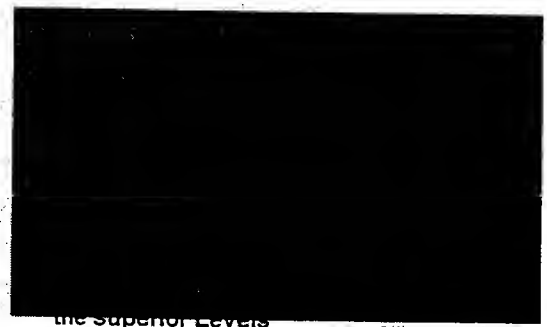


What are the basic functions / commands of a PLC ?



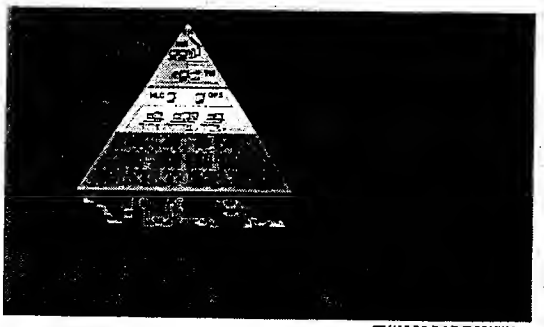
Basic functions of a PLC

Functions of the Process Control Level

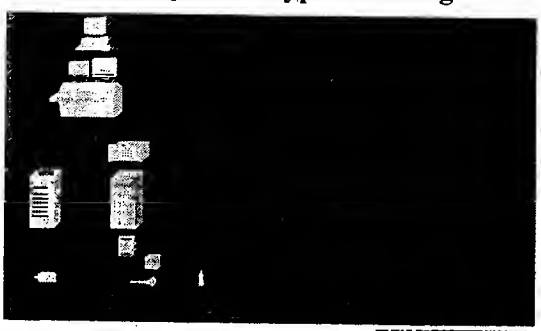


Control Room with Large Screens

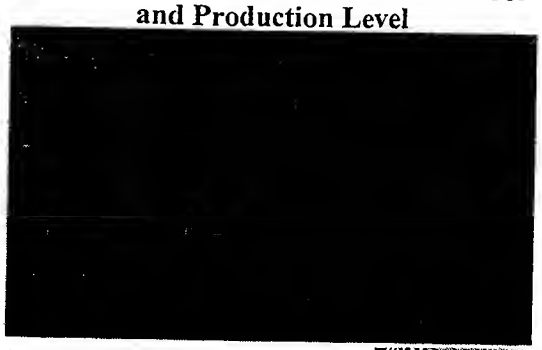
Communication



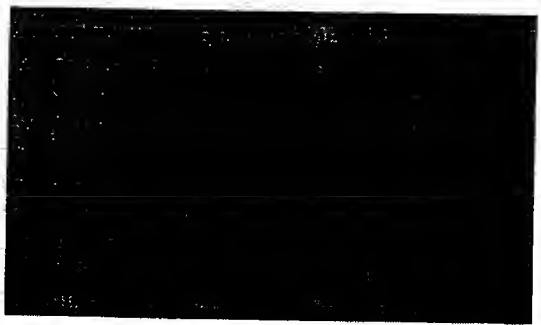
Automation Systems Typical Configuration



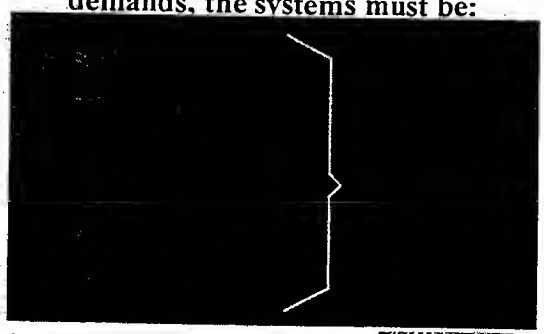
Accepted Standards for Process Control and Production Level



Cement Plant = Long Term Investment

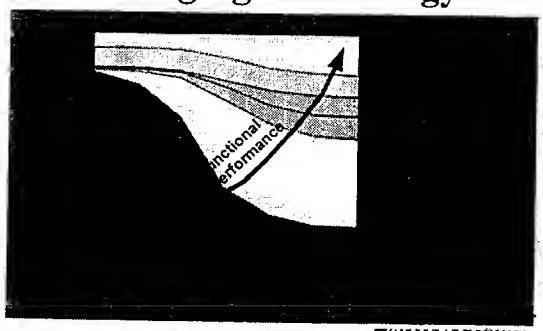


In spite of the high complexity of the demands, the systems must be:



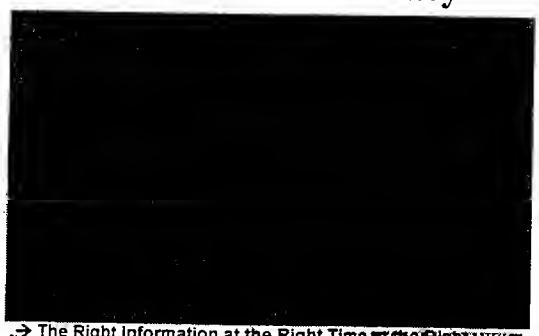
Control Philosophy

Changing Technology



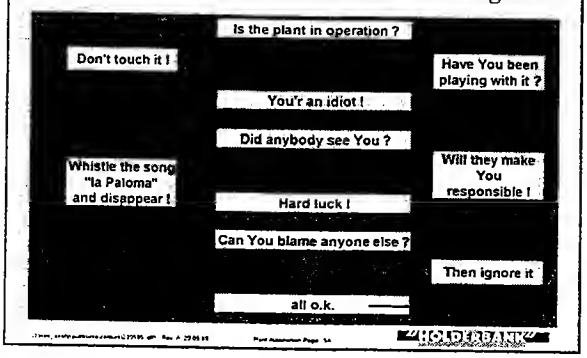
Important Design Principles

Automation Tendency

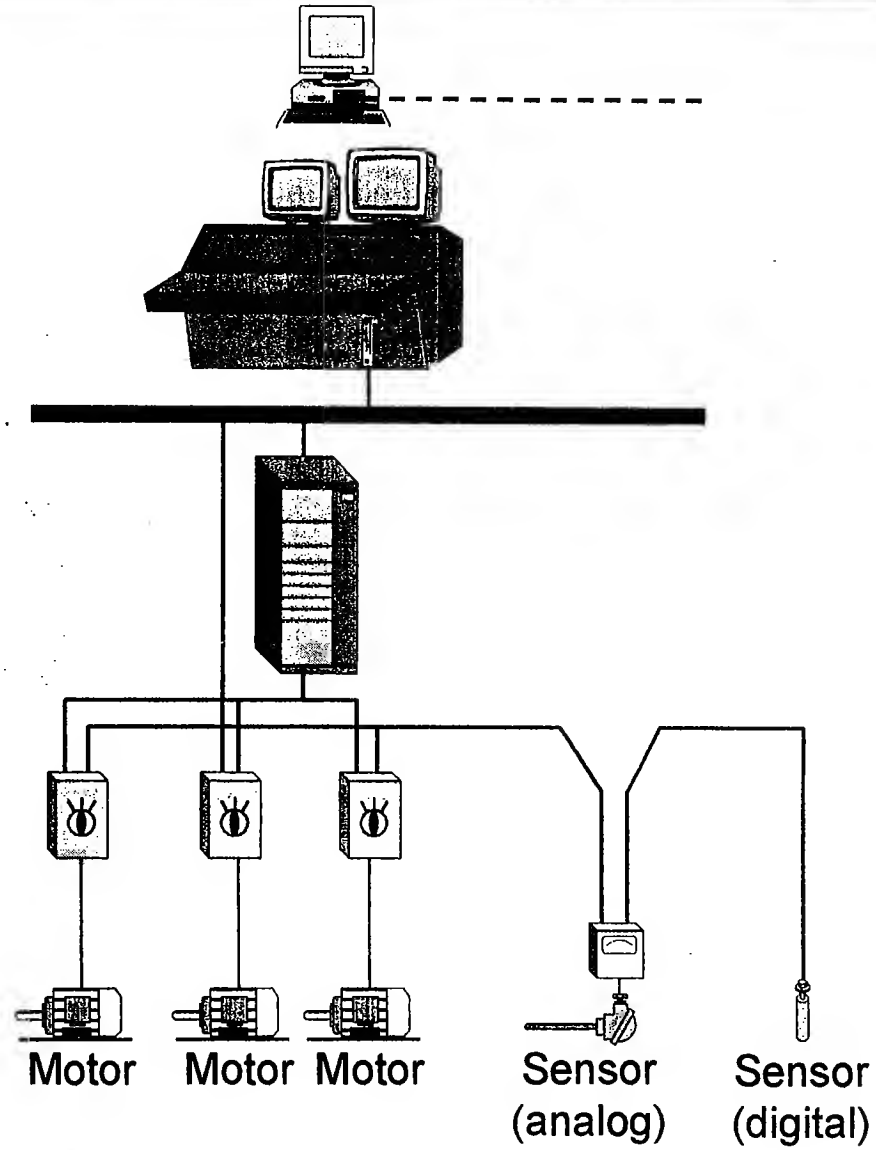


→ The Right Information at the Right Time at the Right Place

Guidelines for Electrical Maintenance Engineers



Automation Systems Future Configuration



← Management Information Network

← Human/Machine Interface
Operator Station
Control System w. Soft PLC

← Network

← LVD Low Voltage Power Distribution

← Cabling, Field Bus, Power Bus

← Distributed Motor Controllers

← Transmitter

← Primary Element

Chapter 4

Plant Automation

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2. Modernization of Control and Supervisory Systems in the Cement Industry 129

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Plant Automation

Roland Luder

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2.2 Uniform operation 122
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2.5 More efficient maintenance 123
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4. PROCESS AUTOMATION 125
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1. INTRODUCTION

Automation can be defined as a physical system which is capable of reaching a certain target without any human action.

Applied to the cement manufacturing process, different targets can be formulated. One could for example be: load bags to the packing machine without any human action. The most extreme target would be an automatic bank transfer to the shareholders of the dividends of the totally automatic cement manufacturing.

But already the small example of the bag loading shows that automation very soon reaches limits: An automatic bag loader today is no longer a problem, but the target "loading bags to the packing machine without human action" has by far not be reached. Who does the unloading of the bags? And who does maintenance on the bag loader? Theoretically, these activities could also be made automatic, but it would generally not be feasible and there are of course still limits (e.g. who does the maintenance of the "automatic bag loading maintenance machine"?).

When we mention "automation" we nowadays immediately associate computers or electronic equipment to it. This is only natural since almost all automation systems are today based on electronic components. When the term automation system is used it shall therefore mean a system based on electronic components.

2. BENEFITS OF AUTOMATION

The example given in chapter 1 shows that an automation project has to be clearly analysed for its ultimate benefit within the cement manufacturing process. The achievable benefits can generally not be expressed or calculated in exact figures, automation is always related to or has to be compared with human factors and these factors are difficult to determine.

Nevertheless, some fields where automation can yield some benefits shall be further elaborated:

2.1 More reliable operation

The installation of modern electronic equipment instead of elector-mechanical components guarantees a higher reliability of the control system. Equipment downtime can be reduced due to the availability of detailed process warnings.

2.2 Uniform operation

The operator is released from all routine operations, checking and controlling. He is thus in a position to fully concentrate on the optimum and efficient operation of the process. In this objective he is greatly supported by the system which presents all relevant information in a logic and easily understandable way.

2.3 Energy saving

A modern control system automatically starts and stops motors according to the process requirements. Inefficient continuous running of motors and high energy losses during unproductive start-up trials are eliminated. The control system can easily include the control of the peak load to the plant (energy management). A better stabilised process can have a very positive influence on thermal as well as on electrical energy consumption.

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2.4 Manpower saving

Achievable savings depend on actual labour situations, labour costs, labour policies (unions) etc.

2.5 More efficient maintenance

The maintenance on control and instrumentation can be kept to a minimum due to the installation of electronic equipment. No time-consuming troubleshooting will be required since failures are displayed in clear text. Mechanical maintenance can be optimised and preventive maintenance can be introduced due to the availability of detailed failure and warning messages and statistical evaluation of all events.

2.6 Better quality

The market demands for less tolerances in the cement quality. An uniform operation, more precise on-line measurements are a guarantee also for better quality.

2.7 Protection of environment

A modern automation system not only controls the process, it is furthermore and more responsible for continuous environmental protection.

Example: Exhaust gas analysis
Waste water treatment
Energy management

The given examples are typical closed loop control systems. Thus, it is not dependent on human observing abnormalities and reactings. It is a continuous process acting in very narrow limits.

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3. AREAS OF AUTOMATION IN A CEMENT PLANT

Automation can be applied in many different fields of the cement manufacturing process.

Table 1 gives an overview of those fields which are mainly concerned with automation. It also shows what type of hardware normally is applied to perform this automation.

The table clearly shows that a vast variety of equipment can be used for the different automation tasks. When it comes to software this variety is even bigger. In order not to end up with a patchwork of different automation systems, it is therefore very essential to carefully plan and to evaluate any automation project. It is mainly important to always keep the entire process and the entire system in mind. Automation can start with the modernisation of a single machine, but the automation or control of this single machine should be designed from the beginning to fit into an overall automation concept.

Table 1

	AUTOMATION AREA	AUTOMATION HARDWARE
1)	QUARRY CONTROL	
	Quarry planning	Workstations, PC's
2)	PROCESS AUTOMATION	
	Motor control	Programmable controllers
	Process control	PC's, Workstations
	Monitoring	Graphics Displays
		Process computers
3)	DISPATCH CONTROL	
	Loading	Programmable controllers
	Administration	Workstations, PC's
4)	QUALITY CONTROL	
	Sampling preparation	Programmable controllers
	X-ray	Including micro-computers
	On-line raw mix control	PC's, Workstations
5)	MAINTENANCE	
	Materials management	Workstations
	Maintenance planning	Personal computers
	Reporting, documentation	
6)	MANAGEMENT INFORMATION, ADMINISTRATION	
	Production reports, statistics	Workstations
	Commercial data	Personal computers
	Personnel administration	

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4. PROCESS AUTOMATION

The next chapters will concentrate on process control and some aspects of management information only. The other fields of automation are mainly dealt within the respective chapters.

Process automation or process control has become more and more important together with the increasing capacity and complexity of cement plants. Table 2 shows how the number of drives and instruments (which can be used as degree of complexity) has increased in the last decades. The figures given represent very approximate values of a medium-size cement production line.

The technology of control system components has drastically changed during these periods, whereby most new developments have been based on the respective underlying developments in the field of electronic/computer technology.

To predict the future is of course always a difficult task. Some trends, however, can already be observed. One trend is surely that there will be developments for more memory at a lower price which will allow better and faster information handling. This again will have an influence in the further development of self-learning ("expert" or "intelligent") systems.

The trend in graphic displays goes towards bigger, flatter screens with higher resolution.

An additional tool as operator interface might be the development in voice information or voice command. For the moment it is, however, more of a nice game than an efficient tool to increase production.

Table 2

-	1940	Local control of individual machine, local pneumatic/mechanical indication 50 motors, 20 instruments
-	1960	Central control rooms , sequence control of motors (relay), remote indicatin of instruments 200 motors, 70 instruments
-	1975	Central control rooms, decentralised programmable controllers, monitoring or process computers 400 motors, 150 instruments
-	1985	Central control rooms, centralised controllers with remote input/outputs, graphic displays, data-highways 800 motors, 300 instruments
-	1990	Central control rooms, distributed control systems, automated documentation tool, integration of management information 1000 motors, 500 instruments
future		Integration of expert knowledge, voice command, use of transputers, field bus to the level of transmitters, intelligent sensors

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5. PROCESS AUTOMATION COMPONENTS

The different main components of a process automation system are shown as the three lower levels of the pyramid in Fig. F49591 "Structure of plant automation".

The total system of these component functions in a similar way as a human being:

Sensors and the cabling perform similar tasks as the nerves - capturing and transmission of information to the brain.

The controller performs similar jobs as the brain. It processes the information.

The result of this data processing is transmitted again by the nerves to the muscles which transform the information into a physical movement as in a plant control system, where the controller output is transmitted - again over cables - to the actuator or the MCC where the information is transformed into physical power.

Needless to say, a plant control system plays a very important part in the performance of a plant, similar to the one of the brain and the nerves of a human being.

The individual components "analogue sensor/transmitter" will be further explained in the chapter SENSORS, the component "logic controller" in the chapter MOTOR CONTROL.

In order to understand the subdivision process controller / logic controller / computer / man-machine interface within the so called "integrated controller" one has to go a little bit back in history of control systems.

Some years ago the subdivision of an automation system into the parts

- motor control
- instrumentation and process control
- data logging

used to be clearly defined by the corresponding type of equipment:

- relays, then programmable controllers for motor control
- separate instruments and closed loop controllers for process control
- recorders and operators day-book, then computer data logging equipment.

Every type of equipment used to have its own type of man/machine interface

- push-buttons and lamps for motor control
- potentiometers and instruments for process control
- keyboards and displays for data logging.

This control equipment has also been offered from more or less specialised manufacturers, as e.g.:

- Modicon for Motor Control
- Honeywell for Instrumentation
- Digital Equipment for Data logging

Today there is a market tendency for the established manufacturers to expand outside the field in which they have hitherto specialised and to enter the other areas of automation more and more; this means that modern programmable controllers can perform process control tasks and condition data; computer systems are also able to perform sequence control and they are all linked by computer bus systems and networks.

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6. PROCESS AUTOMATION SYSTEMS

For the automation of a cement plant which has to combine all the three functions of motor control, process control and data logging, it is thus possible to select the most appropriate component for each of the three areas and to integrate them in a complete system. In practice, though, this procedure is often obstructed by the lack of compatibility of the units (i.e. difficulty in interconnecting them) because, owing to the rapid progress made in electronics, neither hardware nor software are sufficiently standardised. Furthermore, the user is confronted with the problem of maintaining three inherently different systems.

Fortunately, we have seen that manufacturers have expanded their fields of activity and that today, a level has been reached where all three functions can be realised with only one or perhaps two different control systems.

The structures of complete systems from different suppliers, however, still shows considerable differences.

Since in the cement industry motor control represents the most extensive part of the automation system, it is regarded to be optimal to choose a programmable controller as basic control unit.

Among the various types of programmable controllers available (in Switzerland about 130 manufacturers are represented on the market), it is the medium to large units that are most suitable for the cement industry. "Large" units are regarded as being those capable of catering for the motor and process control of a complete department (e.g. raw mill) with all the associated processing of data for the man/machine interface. As regards configuration, documentation and ease of modification, large systems are preferable to a number of medium or small units.

With this concept the automation system of a complete cement production line generally consists of one programmable controller for each department (crusher, raw mill, kiln, cement mill, coal mill, cement dispatch), each controller with its separate operator station and each performing motor control as well as process control. For general management information and reporting, the operator stations are interconnected and linked to a separate data base computer being part of the information management system.

- Power supply:
  In order to absorb brief interruptions of the plant voltage and in order to correctly alarm any real voltage supply failure, it is recommended to connect the entire control system to an uninterrupted power supply.

- Test system:
  The spare parts required for the automation system are preferably assembled to form a "test system". This system should be procured before anything else in order that the software can be set up and tested before the actual process control system is installed. After commissioning, the test system is used as "spare parts stock" as well as for testing possible changes to the programs and also for training new personnel. Naturally, the "spare parts stock" has to be replenished according to the consumption.

- System documentation:
  To establish the system documentation, every possible use should be made of the facilities offered by computers today. A special chapter will be allocated to this topic.

- Communication:
  A good communication system can help a lot to improve efficiency in operation and maintenance. Here, too, a clear concept should be established. All possible methods:
  - telephone/paging, walkie-talkie, intercom, loudspeaker, fax, video etc. should be considered and possible applications evaluated.

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7. CONCLUSION

Modern process automation systems can contribute a lot to the enhancement of the efficiency of plant operation. Nevertheless, the degree of the most feasible level of automation has to be carefully evaluated. And most important: even the highest automated plant needs a good maintenance in order to run efficiently. Or in other words: The most luxurious process control system with the most brilliant graphic displays cannot guarantee smooth operation if it does not receive reliable, correct information from the sensors.

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Modernization of Control and Supervisory Systems in the Cement Industry[1]

Rene Säuberli

1. Reasons for modernizing control systems 130
2. Planning of modernization projects 132
3. Design of a control system 133
4. Practical Aspects of the Execution of Modernization Projects 135

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At times when financial means are restricted and the competitive situation on the market is tense, all efforts must be directed towards savings - and thus towards optimization - of the production process and maintenance. Modern technologies in the plant control and monitoring systems can contribute appreciably to a reduction in costs, provided they are properly applied.

1. REASONS FOR MODERNIZING CONTROL SYSTEMS

Depending on the plant concerned and its environmental conditions, there can be a variety of reasons for undertaking modernization projects. Generally speaking, though, the main reasons for such projects are one or more of the following:

a) Plant and equipment is worn out and requires too much, too expensive maintenance

b) The plant or system is no longer reliable enough

c) Owing to a lack of spare parts the plant or system can no longer be repaired

d) Maintenance agreements are expensive

e) The plant or system does not permit any further automation of the production process

f) The system does not permit any further optimization of the process, the information, the maintenance and /or energy consumption

g) Environmental conditions, such as public opinion, environmental protection, emission limits, etc., make further use of a system or process questionable.

Item c) above points to a new aspect of electro-mechanical equipment, that if its "maintainability". This aspect is very important and therefore deserves to be discussed at greater length, especially because in most cases works managements fail to attach sufficient importance to it. The conventional mode of looking at maintainability is restricted to the ease of maintenance, when components, machines or systems have to be maintained or replaced because they are showing signs of wear or old age. The same also applies to automation systems, but here a new aspect of maintenance becomes apparent, that of systems having to be replaced because, due to technological changes, systems or their components become obsolescent without being "worn out".

The extremely rapid evolution of electronics, together with the low cost of hardware, has provided advantages as regards reliability, usability and application potential. On the other hand, this rapid evolution has resulted in a reduction in the useful life, especially on account of the short presence on the market of certain electronic components. In other words, components which only appeared on the market five or ten years go, can no longer be supplied.

The replacement of electronic equipment gives rise to problems, in that electronic units cannot simply be replaced by new products. The reason for this is that standardization is hardly possible, on the one hand owing to the very rapid evolution and, on the other, due to the growing complexity of the units. For this reason, it may be impossible for a manufacturer to design his own products so that they are compatible. The presence on the market of a modern electronic control system may be assumed to last for about seven years, i.e. about one third of the normal life of a machine that is controlled by it. In other words, nowadays a machine outlives roughly three generations of electronic units.

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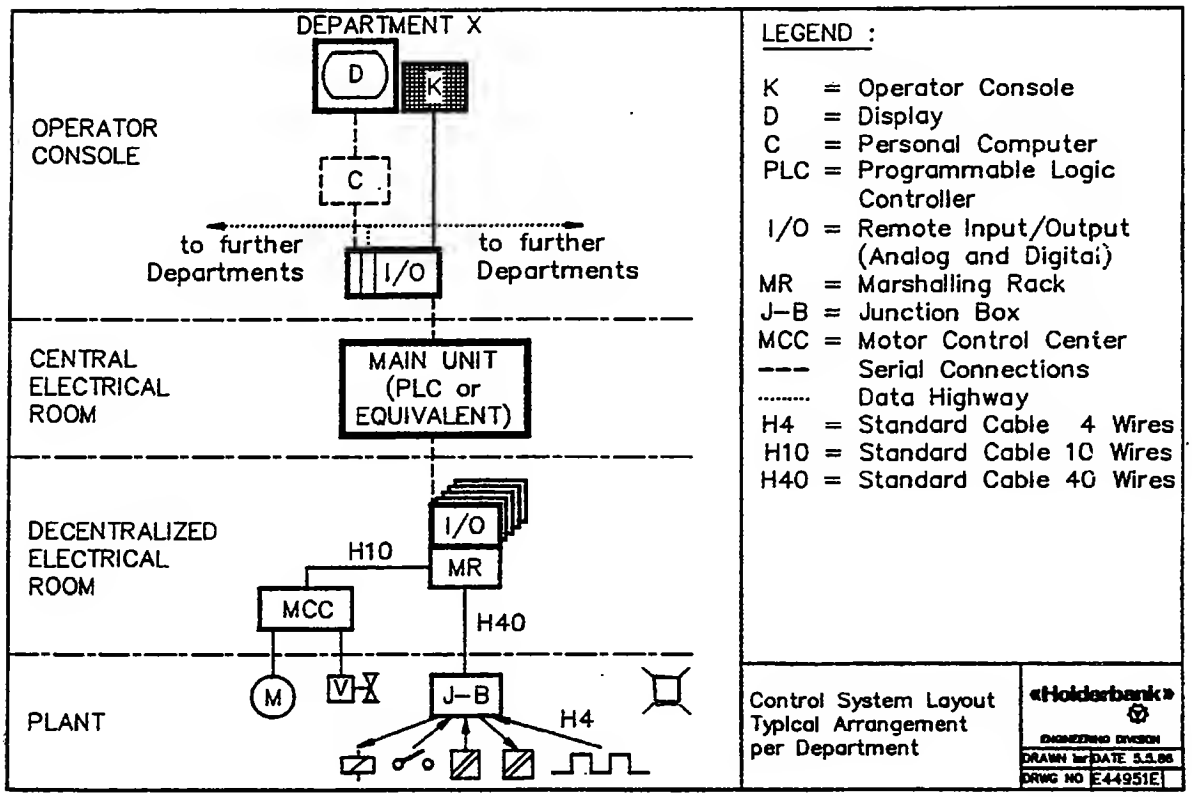
Fortunately, though, there are signs that the situation with regard to compatibility of electronic components is slowly but surely improving. But despite this the trend will not produce material effects at one. The users of the present - and the next - generations of control and supervisory systems will for the present have to put up with the fact that their systems become obsolete much earlier than desired.

Figure **Control system layout**
Typical arrangement per department



2. PLANNING OF MODERNIZATION PROJECTS

Although nobody will question the ability of modern electronic equipment to improve operational sequences, the task of determining the return on investment for such a project is highly complex and differs from one plant to another. The properties of an electronic control system compared with a conventional system have here to be viewed in the light of human factors, i.e. direct human intervention as opposed to automatic operation, human errors compared with the dependability of electronic components, human negligence compared with the robot-like stability of programmed control sequences. How can these factors be taken into account in a feasibility calculation? The advantages attainable with modernization projects should always be measured against the existing equipment and actual operational procedure. To do this the first step is always to analyse the momentary state.

Modernization projects for the control and monitoring systems in cement works should be included in the long-term planning of the company. The momentary state of maintenance, maintainability and performance of the existing system has to be determined from time to time, in order that the start of modernization can be specified without haste. If the replacement of obsolete equipment were suddenly rendered necessary, this could give rise to some unforeseen shutdowns in the plant. If such unscheduled replacement measures occur frequently, it will be automatically necessary to replace the entire system step by step, without having achieved any structural improvements, or profiting from new technologies, i.e. the whole job is only patachwork. Then, in the end, the whole system may be even more complicated than it was before. To avoid this, it is always advisable to keep pace with modern standard by replacing a control system as soon as important parts no longer perform their functions satisfactorily or are no longer obtainable on the market.

The planning of a modernization project must begin with a detailed assessment of the existing facilities. This should cover the following points:

1) Determination of the momentary state.
2) Specification of a strategy with regard to the technical and operation objectives which shows in what steps, in what time and at what costs they are to be attained.

The difficulty in such an assessment is that the various staff members in the works have with time become blind to what is going on and - despite their best intentions - are no longer able to recognize the significant and the really crucial points. It is therefore recommended in all earnest that external advisors be entrusted with this task. But the production and maintenance managers of the works must be included in the planning process as early as possible.

Planning becomes particularly important when modernization projects are involved, especially for control and supervisory systems. This is reflected in the costs: the outlay on planning and engineering of the modernization of a control system can amount to more than half the total cost of the project. A well thought-out control concept also takes into account aspects of changes in technology and the maintainability, as outlined earlier on. The components of such a system are therefore "designed to be replaced". That may sound somewhat strange, "designed to be replaced", but it does correspond fairly accurately to the prevailing circumstances.

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3. DESIGN OF A CONTROL SYSTEM

The control units available on the market can be combined to form an almost unlimited number of control systems. Therefore, the choice of suitable units and of the most suitable system for a given application should not be left entirely to the suppliers and/or the users.

One must be fully aware of the fact that "control" does not consist solely of the instruments and graphic monitors in the control room. They are merely the visible part, the "tip of the iceberg" so-to-speak. The suppliers of electronic units generally tend to offer only this "tip of the iceberg", which nowadays is the cheapest part of the whole control system .The rest of the "iceberg" is generally ignored by the supplier or he simply leaves it to his customer, although that is usually more than he can cope with, when it comes to a full estimate of all aspects of the modernization project. For a user it is normally difficult to analyse his own requirements, to analyse the market correctly and to employ the most suitable engineering tools. The following are some design principles which have been successfully applied by HMC/TC to a number of cement works in past years.

a) A control/automation system should be constructed from the bottom upwards.

b) The existing facilities for sub-dividing the cement production process into independent sub-units (departments) should also be taken into account when designing the control system.

c) Owing to its importance for the operation of the works, an automation system should primarily be selected with regard to its reliability and maintainability, i.e. it should be as simple as possible.

d) An automation system must be designed in such a way that future requirements can be satisfied easily and the replacement of obsolete parts is facilitated.

Requirement a) signifies that a start must be made on the concept at the lowest level of control tasks. This includes the numbering system (labelling, the wiring, the standardization of sensors and signal generators, the detailed emission of first alarms, the control and the supervision of auxiliary functions, and so on. All of these have a very definite influence on the overall performance of the system as a whole, where performance applies not only to the execution of functions, but also to the maintainability, extendibility, ease of future replacement, etc. Poor design at his lowest level can possibly be compensated by using additional computers at higher levels of the system. But the overall structure and, thus, the factors mentioned under c) are certain to be affected by such a procedure.

Point b) is taken to imply that each department should be decentralized as far as possible. Only general functions of a plant, e.g. reporting, statistics, documentation, process optimization, should be dealt with centrally. This enables the departmental maintenance of the control system to be practised without any risk of causing a stoppage in other departments.

Point c) says that the system should be s simple as possible, i.e. it should consist of a minimum of "black boxes" with a minimal number of standardized connections between them. In this context a "black box" means a hardware or software unit, requiring no internal maintenance or adaptation.

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Point d) implies that even the most modern control system does not exist as long as the machines in the cement industry. It is therefore very important to set the standards for the introduction of new control or automation tasks to the existing control system well in advance and even to plan for the future replacement of the whole system. That corresponds to what is meant by "designed to be replaced". Experience has proved that a decentralized, standardized control system can satisfy such demands very effectively. By replacing in easy stages, spare parts are made available for the still existing system, which is not to be replaced yet in all departments. Another positive aspect of this mode of procedure is the improved distribution of the investment costs over a period of time, without incurring greater risks for availability of the spare parts.

As the result of such considerations, HMC/TC has developed and put into practice an integrated, decentralized control system, tailored to the needs of the cement industry. The diagram shows the structure of the system schematically for one department of a plant (for all the other departments the structure remains in principle the same). In a system of this kind each main unit e.g. a programmable logic controller (PLC) processes the signals of the motor control and simultaneously those of the instrumentation. The following advantages accrue from this procedure:

- Only one make of units is required.
- The interlocks between analog (process) signals and the motor control can be effected direct - at the lowest level - without wiring or without using the data highway. On account of the high reliability, analog sensors are increasingly being used for the protection of machines (e.g. analog temperature measurement in motor windings or gear bearings, for the measurement of oil flow, etc.). In an integrated system these sensors are wired direct to the PLC. Preliminary alarms, shutdown alarms and the shutdown interlocks are programmed in the same unit.
- The alarms from digital or analog signals are given by the same unit; there is only one kind of alarm processing. Alarms from the process can easily be interlocked with operating conditions of the motors, for example.
- Control of a variable-speed drive is very easy (start/stop = motor control; changing speed = process control). Dividing these two tasks between two separate units/would unnecessarily complicate the system. In view of the growing number of variable-speed drives, the integration becomes more and more important.
- Closed control loops can be easily integrated, in order that hey can meet the various conditions during starting and stopping operations.
- Hardware and software can be largely standardized, each department using the same type of black box (hardware and software). Since all the control functions are allocated, the corresponding software does not have to be nearly as intricate as in a centralized system.

The computer capacity and the facilities for direct operation of displays by the central unit must be utilized as man-machine interface, thus achieving a fully integrated system. If the selected central unit does not offer enough scope, modern microcomputers or personal computers should be employed. In this case the microcomputer calls all the information required off the main unit. However, it should not be used to initiate any intervention in the process, but should remain exclusively reserved for the efficient processing of information. All control commands and motor/process interlocks, sequences, etc., should be processed direct by the central unit.

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4. PRACTICAL ASPECTS OF THE EXECUTION OF MODERNIZATION PROJECTS

As guidelines for the execution of modernization projects the following general rules have proved their worth in the past.

a) The control system should not be included in the scope of delivery of the machinery supplier, because he looks for the system that will be most economical for his own purposes. The customer, however, has to make sure that he is going to receive the system that will be most economical for his process, in which maintenance and future developments of the plant have also to be taken into account (integrated system). The only exceptions where the control system should be included in the delivered package are the very special control systems for electronic weighing equipment on bagging machines or for the power electronics associated with drives, etc.

b) The equipment numbering system must be examined to make sure that all parts can be clearly identified. Can the system also be used for detailed indication of events and alarms, for data acquisition and for electrical and mechanical maintenance?

c) The safety regulations (existing and planned) in the user's own country must be examined very closely before the appropriate safety concept is drawn up. It must be checked whether local stop switches for each motor are sufficient or whether a main isolator has to be provided next to the motor. Where do pullcord switches have to be installed? What warning devices (horns, sirens, flashing lights, etc.) are needed when starting up the machines and where are "panic stops" necessary?

d) The existing wiring has to be examined and a clear wiring concept drawn up. The wiring should be such that it is not affected if the control system is replaced at some time in the future. An adequate number of spare lines must be available, should it be necessary to extend to monitoring of the process by additional sensors. These will have to be individually wired also in the future. It is therefore essential to consider carefully how this lowest level of the system should be designed, to be standardized and as simple as possible.

e) Dependable, standarized interfaces should be provided in the control system and with all external control sub-systems (hardware and software). Problems are usually caused in control systems by unclear, complicated or uncertain communication processes and/or interfaces. The communication problem, however, cannot be solved by installing a data highway or bus, a local network or similar facility. They are merely tools like a telephone or telex. What is important is the contents of the message or information that is to be transmitted and its interpretation by the receiver.

f) Sufficient time must be spent on drawing up a clear concept and on adequately detailed basic engineering, before any system is purchased. All functions to be performed by the automation system - as well as those it has not to perform - must be clearly defined in advance. As far as possible, employees from production and maintenance should be included in this planning. It is indeed true that modern control systems permit a program to be simply changed, but that is only true of minor changes. Major changes in the concept are never simple: that is why a detailed specification of the system and its components is so important. The degree of automation has to be estimated very carefully. The higher this degree of automation is, the greater the number of decisions that have to be programmed in advance and the more complicated and expensive programming becomes. The aim of the specification of a control system is to stipulate the optimal degree of automation for every single task.

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g) A detailed alarm system has to be specified. This system is one of the deciding points in the improvement of operation and maintenance.

h) The complete system has to be tested before it is installed, especially the communication interfaces. In this phase it is important to include the works personnel.

i) The spare parts have to be purchased as early as possible and utilized in the construction of a training system. This enables the works personnel to become familiar with the new system at an early date. A system of this kind, composed of spare parts, can later be used for testing programs, as a training centre and as a "stock for warming up" these spares.

j) An important factor is the careful compilation of the documentation for the equipment and spare parts. It is well worth while to store the documentation on the computer from the very start of modernization. Experience has proved that existing documentation is usually out of date and has to be renewed in any case. Computerizing - when it is carefully planned and carried out - is a good method of ensuring that the documentation no longer becomes out of date.

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Chapter 5

Sensors

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Sensors

Wolfgang Kornberger

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1. INTRODUCTION

When looking at the automation pyramid, it becomes obvious that instruments and sensors form the foundation for any control and automation. It is here where the information is gathered which is then used further in the automation pyramid for either:

- interlocking and control for automated production
- regulation with PID-controller and high level control to ease the workload of the operator and to improve the plant performance (reduce energy consumption and/or increase production)
- display and register process values to inform the management and the operator about the plant performance

Note: It is important to remember that it is impossible to control anything unless the parameters have been accurately measured in advance.

2. SENSORS (INSTRUMENTATION) BASICS

The task of an instrument or a sensor is to convert a physical value into an electrical signal. A signal is picked up with a primary element, then converted in the transmitter to an electrical signal and finally transmitted to a control centre where the signal is further treated for either display, alarming or control. (See drawing F44570-1)

The example in the drawing F44570-1 shows a pressure transmitter. The pressure (connected on either side) distorts the bellows. This deformation is moving a lever which is connected to a plunger moving in a coil. The movement of the plunger in the coil evokes an electrical signal which then is converted to a standard electrical signal of 4-20 mA.

All transmitters work on a physical principle which depends on the process media, the desired type of measurement and the accuracy required. Some principles are as simple as in the example in drawing F44570-1 given. Others, like gas analysers working on light diffraction are more sophisticated and therefore not only more expensive but as well prone to high maintenance.

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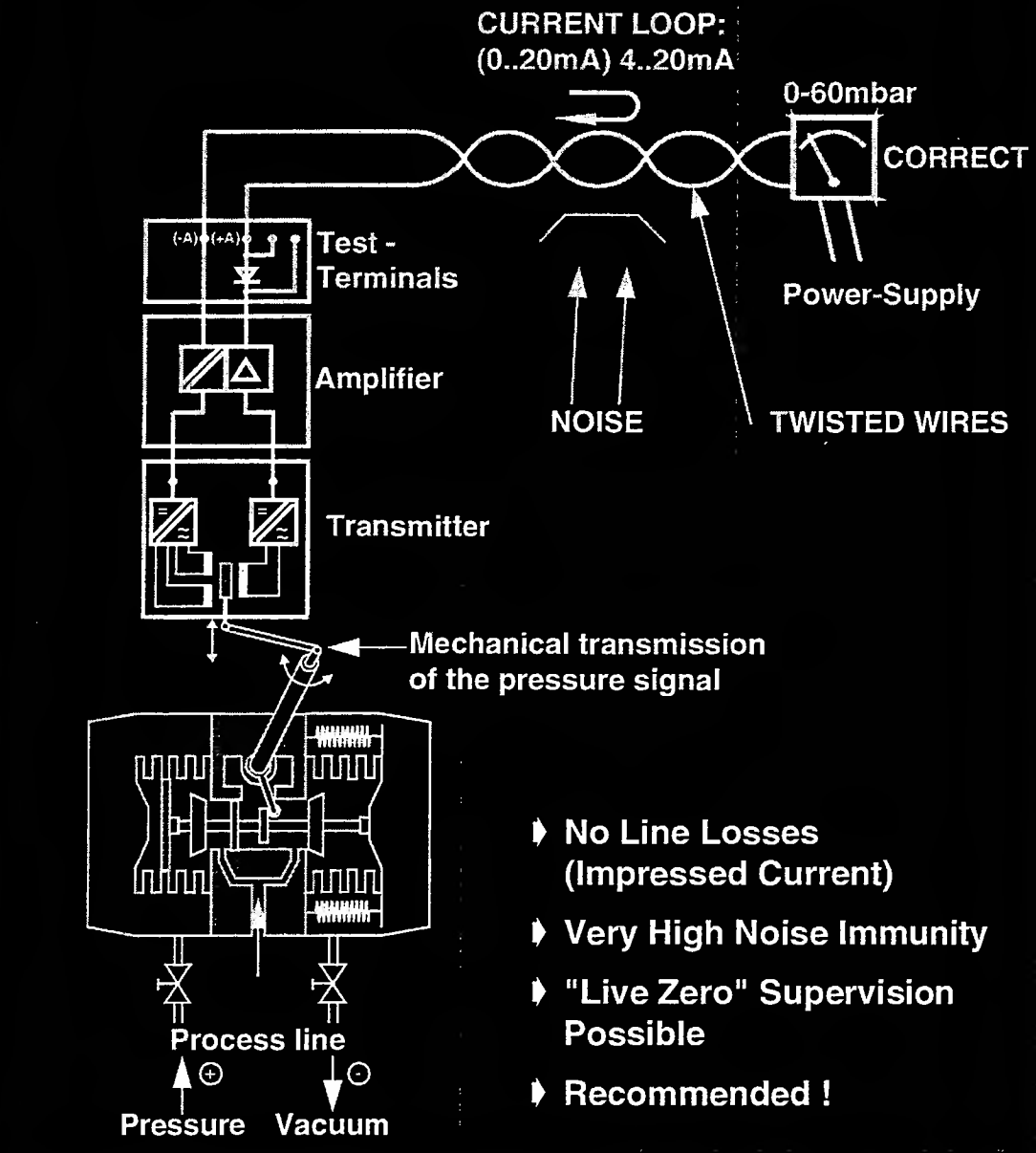
Transmitter with Current Output



- No Line Losses (Impressed Current)
- Very High Noise Immunity
- "Live Zero" Supervision Possible
- Recommended !

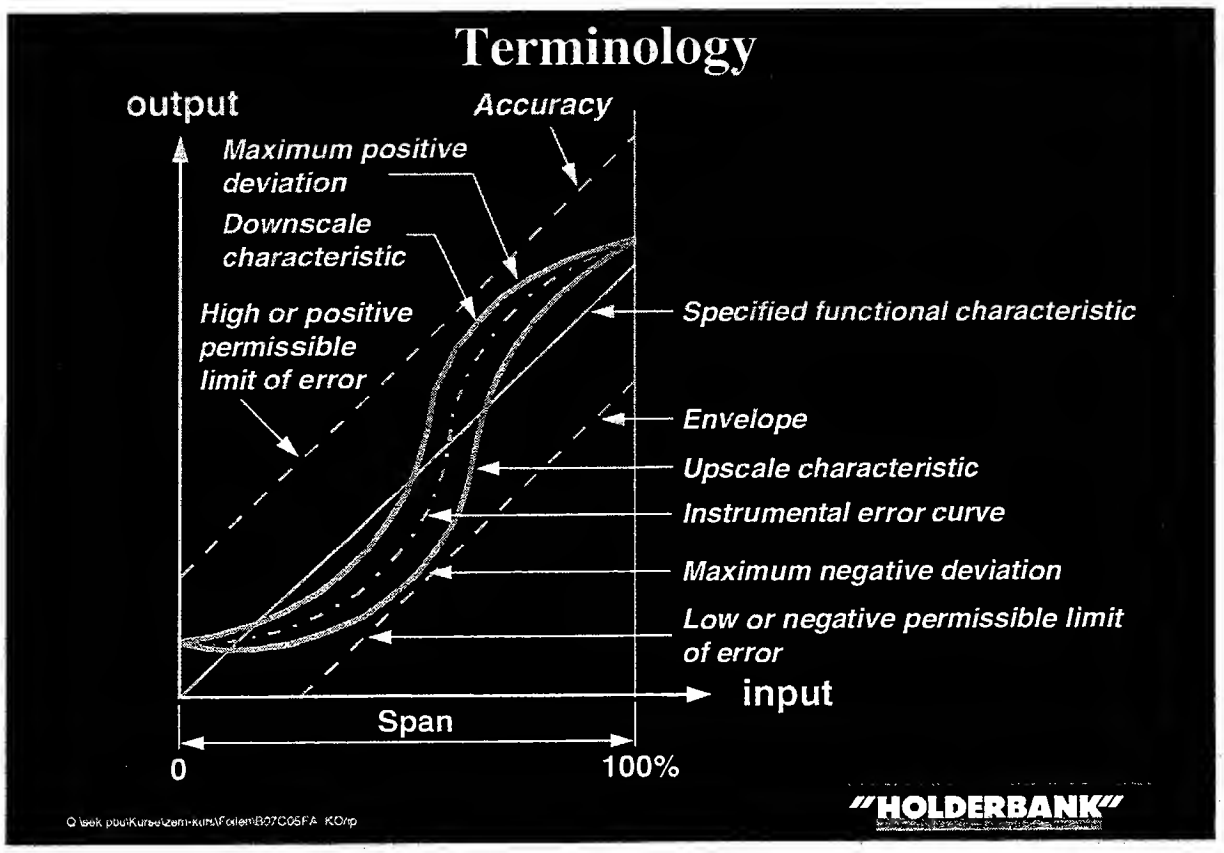
Q:\sek-pbu\Kurse\zem-kurs\Folien\B07C05FB KO/ip (F44570-1)

"HOLDERBANK"

2.1 Terminology

Like in all engineering fields, instrumentation has its own kind of terminology; and to be able to read a technical specification these terms have to be known. The following list gives a short overview of the most important terms used.

Example **Ampere meter .5% accuracy**
Temperature meter ± 5° C



Accuracy:	A number of quantity (usually expressed in % full scale) which defines the maximum error.
Calibration:	The ascertain by the use of a standard the locations at which scale or chart graduation of an instrument should be placed to correspond to the required value. To adjust the output of an instrument to bring the desired value within a specified tolerance.
Deadband:	The range throughout which an input can be varied without initiating response. Deadband is usually expressed in percent of full span.
Deadtime:	The interval of time between initiation of an input and the start of the resulting response.
Damping	Reducing of the oscillation of a process input or the output of a controller.
Drift:	Undesired change of an output over a period of time.
Deviation:	Departure from a desired or expected value also difference between measured value and true value.
Error:	(see drift) Error = indication minus true value = setpoint minus measured value
Elevated Zero:	A range where the zero value is greater than the lower range value.
Feedback:	Positive answer to a demand in change
Gain:	Is the ratio of an output change to an input change. (Reciprocal to proportional band).
Hysteresis:	The maximum difference between the upscale and downscale indications of the measured signal during a full range traverse for the same input.

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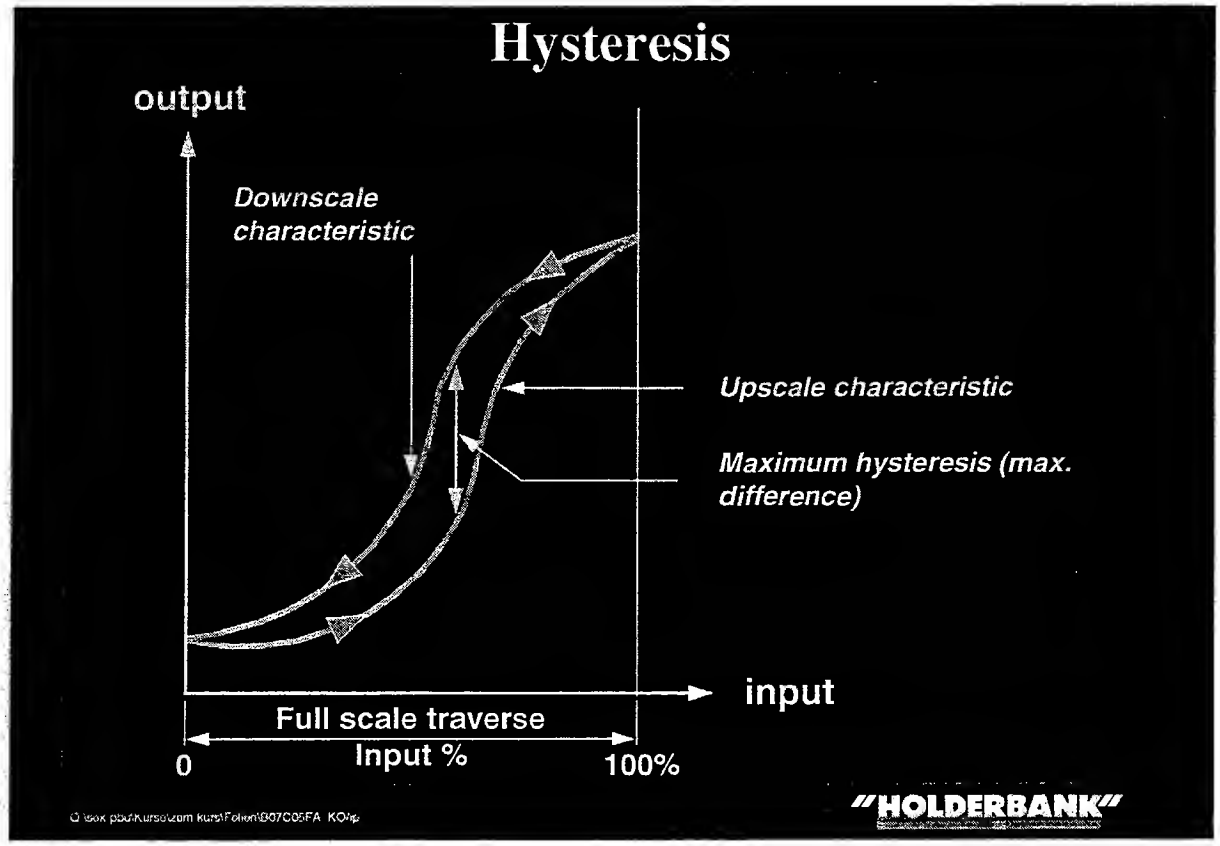
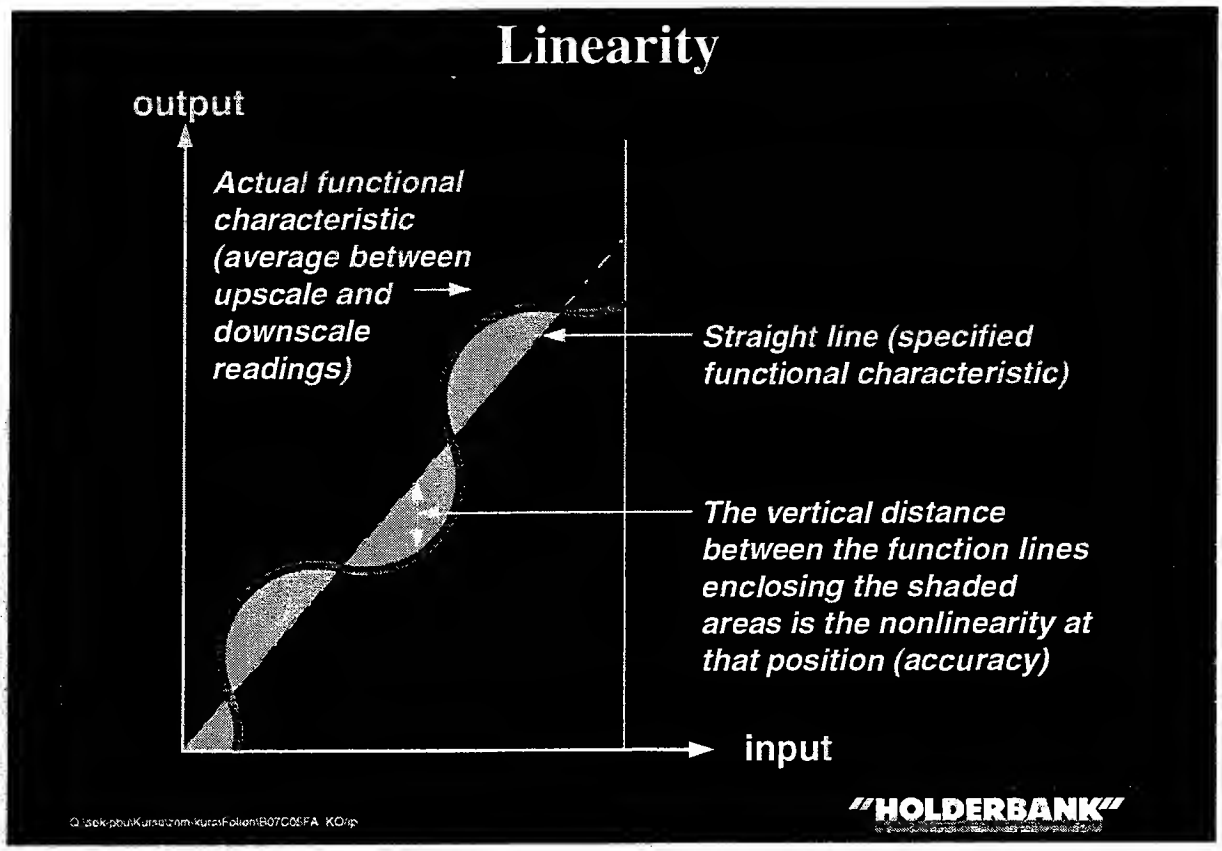


(Alarm limits for example are equipped with a hysteresis in order to prevent repeated signals around the alarm point).

Impedance:	Resistance of a network of resistors, capacitors and/or inductors.
Interference:	Noise (spurious voltage or current arising from external sources or interference between measuring circuit and ground).
Input:	Device to convert the electrical signal into a digital information for further treatment in a Process Station or Programmable Logic Controller (PLC).
Linearity:	The closeness to which a curve approximates a straight line.



Limit:	Alarm limit
Lag:	(Time lag) time elapsed between process and measuring point as well as measuring point and control device.
Noise:	False signal picked up in the transmission line (see interference and signal-to-noise ratio).
Output:	Signal from a device (instrument).
Range:	Region between limits of measuring device expressed by stating the lower and upper range values.

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Illustrations of the Use of Range and Span Terminology

Typical Ranges	Name	Range	Lower Range Value	Upper Range Value	Span	Supplementary Data
0 +100		0 to 100	0	+100	100	---
+20 +100	Suppressed Zero Range	20 to 100	20	+100	80	Suppression Ratio = .25
-25 0 +100	Elevated Zero Range	-25 to +100	-25	+100	125	---
-100 0		-100 to 0	-100	0	100	---
-100 -20	Elevated Zero Range	-100 to -20	-100	-20	80	---

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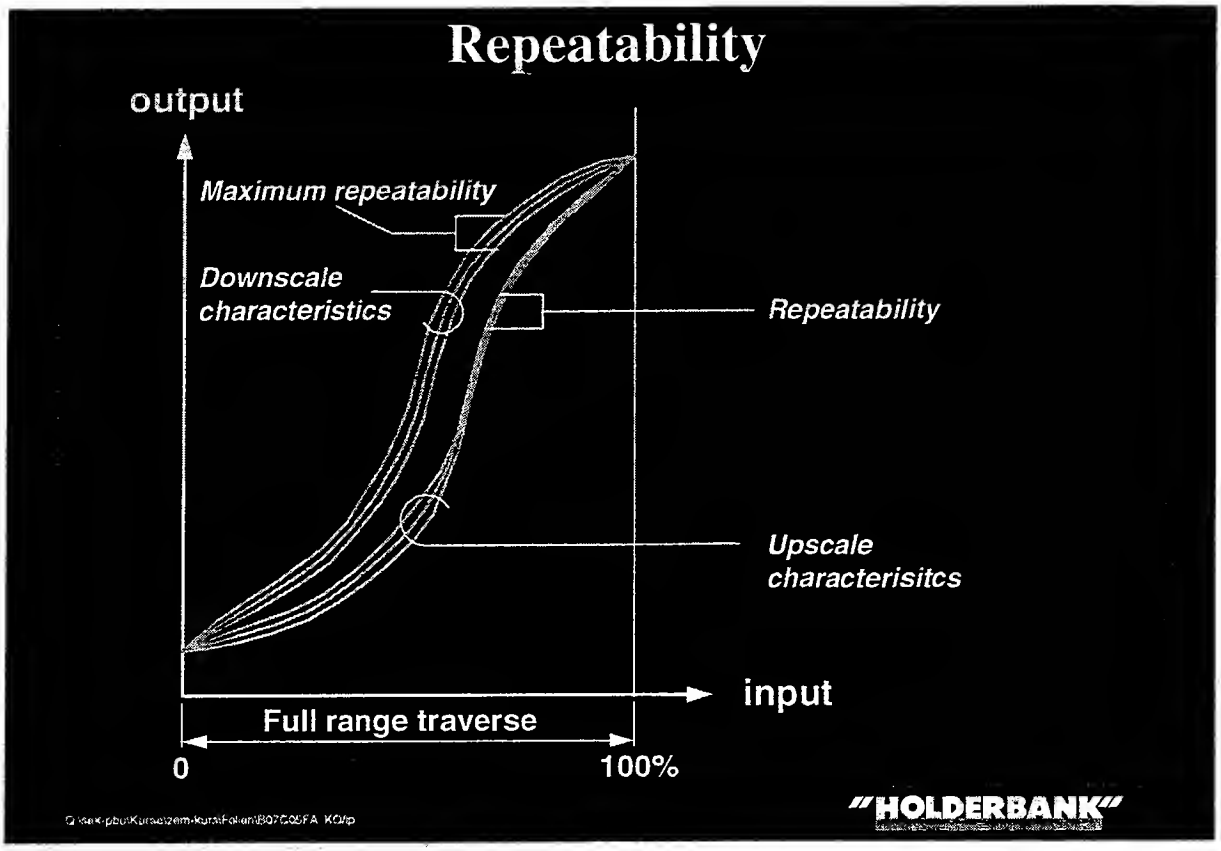

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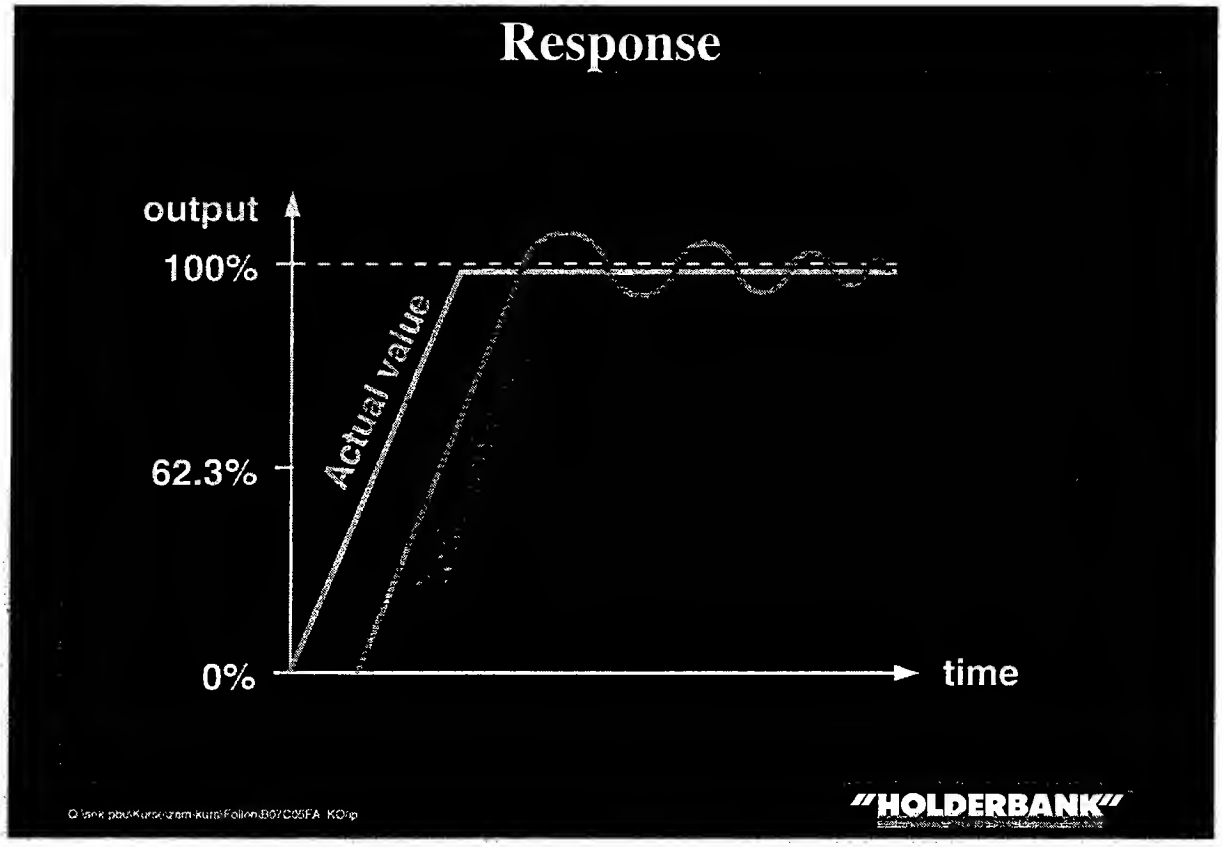
Response: General behaviour of the output of a device as a function of an input.



For additional information regarding PID control refer to the relevant paper in the process technology department.

Sensitivity:	(see deadband and gain).
Signal to Noise Ratio:	Ratio of signal amplitude to noise amplitude.
Span:	The algebraic difference between the upper and lower range values.
Suppressed zero:	The zero value of the measured variable is less than the lower range value. (Zero does not appear the scale).
Time constant:	Time required for an output of an instrument to complete 62.3 % of the total rise or decay.
Zero:	Zero point of scale (to be calibrated frequently due to zero shift resulting in parallel shift of the input output curve).

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3. SIGNAL TRANSMITTER

As mentioned in the introduction, the task of the transmitter is to convert a physical signal into a suitable electrical signal. This electrical signal is then converted into a standard analogue signal of for example 4-20 mA or 24 V digital on/off. Other standard signals exist but the "Holderbank" standard analogue signal is 4-20 mA, for digital on/off signal 24 VDC (Exception America: 110 VAC).

It is, in most cases, necessary to calibrate or verify (adjust zero, span and range) a transmitter. Normal adjustments are Zero = 4 mA and Span = 20 mA. Thus the actual electrical signal representing a process value of 0 - 100% is represented by 16 mA. Calibration is usually performed by simulating the physical signal. Thus, a true zero and if feasible a 100% signal should be evoked in order to calibrate the transmitter over the entire range. The smaller the range of the calibration signal is the more inaccurate the calibration. Each type of instrument transmitters requires its particular way of calibration. It is therefore mandatory to provide the proper instruments for calibration purpose. Additionally, it is important not only to calibrate the transmitter but the entire instrument loop. Thus, the transmission and the signal treatment in either a display instrument or a PLC must be included in the calibration procedure. (See drawing F44570-1)

Some modern instruments require an initial calibration during commissioning and only an occasional check up during their lifetime. Others, like for example power transducers cannot be calibrated nor do they require any adjustments since they are factory precept.

The instruments described above are analogue instruments. That's why the signal varies continuously between 0 and 100%. Often, however only one single point is required. For such a purpose a sensor with an on/off output is sufficient. It saves programming of an alarm limit in case a PLC is used, respective the use of an extra alarm device to produce a thresh hold. However, using an on/off device only can be controversial since this device cannot be checked about its proper function. A 4-20 mA signal can be supervised if it is functioning properly (signal <20 mA and signal >4 mA). An on/off signal can be connected fail safe (contact closed under healthy condition) and a dynamic supervision (contact changes when the process is stopped) included but an analogue signal is easier to verify.

The trend of automation in process engineering leads to "intelligent" field devices. A new generation of instruments called "smart sensors" is on the market. A smart sensor cannot only perform its dedicated task (e.g. measure the temperature) but monitor its performance at the same time. These smart sensors are microprocessor-based field instruments which are designed to communicate with a control unit. A lot of these sensors are operated via hand-held terminals or PC's. Usually the signal picked up by the primary element is converted into a digital signal by an analogue to digital converter. The digital signal is linearized, ranged (0-100% as required), dampened and if required multiplied or squared. The micro controller also controls the digital-to-analogue signal converter for 4-20 mA output and drives the digital communication.

Configuration- and sensor linearization data are stored in a non-volatile EPROM memory. The control unit communicates via a superimposed digital signal over the 4-20 mA signal or via a bus with the smart sensor.

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Each manufacturer has his own communication carrier (bus or via frequency shift keying FSK) over the 4-20 mA signal and his own protocol. Usually communication is performed without interrupting the control loop. Some of the following tests and functions can be carried out via link, smart sensor and control unit:

- loop test of the 4-20 mA signal
- inject a specific mA signal and check the display
- check the configuration data and call up its values
- check changes of the performance of the smart sensor
- name (tag) a device and give an alarm or message text in the smart sensor. Store data about spare parts for the device.

Today, neither in the operator control unit nor in the operator philosophy a compatibility or standardisation is discernible. Due to this situation user acceptance is very low. Additionally, the tasks as mentioned can be performed by the "normal" transmitters connected to a PLC. Thus, it remains questionable to whether smart sensors and Profibus are required today for the cement industry.

The enclosed instrument list shows the most frequent measurements applied in the cement industry and the approximate amount of instruments. The number of measurement, approximately 3000, applied in a modern cement plant is quite impressive. And the tendency is certainly not diminishing in the near future. Especially in connection with environmental control and with rising energy prices, the number of additional measurements will increase.

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Measurement	Application	Typically installed	
		Analog	Digital
Active Power	Current- and Voltage Transformer	200	2
Position	Inductive, Potentiometric, Capacity, Proximity, Mechanical	50	1000
Electric Voltage	Voltage Transformer	4	
Electric Current	Current Transformer, Thermal overload	4	1000
Level	Electro-Mechanical, Sonar Systems, Capacity, Radioactive	50	100
Speed	Tacho-Generator, Inductive and Magnetic (Proximity Type)	40	100
Flow (Volumetric) Liquid, Gas	For Gas Flow: Venturi, Orifice, Pitot-Tube; For Liquid Flow: Turbine-, Piston- and Oval Type Counter, Electro Magnetic	15	50
Flow (Solids)	Belt Weigher, Impact, Flow Meter, Nuclear Belt scale	20	
Temperature	Thermocouple, Resistance, Thermometer, Radiometric	400	
Weight (Mass)	Load Cells, Strain Gage	30	
Pressure	Barton-, Bellows-, Diaphragm Cell Burden Tube, Pizo Crystal	40	20
Sound Level	Sonic Detector, "Electronic Ear" (Microphone)	2	
O_2 Content	Paramagnetic Oxygen Analyzer, Zirconium Oxide Probe, Electrochemical Cell	6	
SO_2, NO_X	Infrared Absorption	2	
CO	Infrared Absorption	4	
Dust	Opto Electronics/Light Absorption	2	
Closed Circuit Television	TV Camera and Monitor Kiln, Cooler, Raw Mill Feed, Crusher Feed	2 4	
Around 900 Analog	2200 Digital = 3000 Sensors + 2000 Control Elements + 2000 EL Circuit Protection		

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4. SIGNALS

When talking about signals at first two different sides must be distinguished:

- the primary side is the actual physical measurement which is detected with the primary element (e.g. thermocouple, diaphragm of a pressure transmitter)
- the secondary side is the signal leaving the transmitter and being transmitted back to the control centre.

This and the next chapters deal with the signal transmitted to the control system since this is an important factor for the installation. When looking at the secondary side of the signal transmission four different signals have to be distinguished:

1) Analogue signal → current e.g. 4-20 mA DC, or voltage e.g. 2-10V DC
2) On/off signal → on/off e.g. 24V DC
3) Pulse → frequency e.g. speed detector pulse
4) Field Bus → code e.g. 500°C as a BCD code

The cement industry is concerned with all four types of signals. In the field it is mainly the analogue 4-20 mA and digital 24V DC; to a lesser extend with pulses and, if at all, they are converted as soon as possible to an analogue signal. The classical cement industry was not concerned with a bus except for communication between PLC's or computers. However, the market shows that the near future is in the application of the Fieldbus. The respective standards are set and respective commercial advantages result. The signals 1) - 3) will become less important.

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5. SIGNAL TRANSMISSION

For safe and efficient operation of the plant it is most important to have a reliable signal transmission between the field, - where the signal is generated, - and the control room, - where it is used for indication, recording, limit supervision, process control etc. The distances from the filed to the control centre may range between 100 meters and 1000 meters, or more. And it is well known that problems with electrical disturbance, interference, noise and losses, increase with longer transmission distances.

For signals as mentioned in the previous chapter, several alternatives for the long distance transmission are applied; some of them are becoming obsolete due to new developments in the filed of electronic components.The simplest method would be to run any sort of signals (pressure, electrical) back to the control room as performed in the early stage of instrumentation when the control centre was local and closed by. On the example of a thermocouple (TC), the problems encountered are discussed.

Is it possible to run a thermocouple extension wire with a mV signal all the way from the thermocouple junction to the indication in the control room? Why not? Mainly because the thermocouple extension wire is expensive. And unless it is very well shielded, which adds to the expense, it will pick-up all sorts of unwanted noise from radio transmitters (walkie-talkies), motors, high voltage cables etc. Since the signal from the thermocouple is only a few milli-volts to begin with, any noise is a problem and it doesn't take a lot of noise to blanket the signal entirely. Such a millivolt signal cannot be transmitted together with other signals in a multi-core cable and it cannot be brought to several users in parallel, such as to an indicator and a recorder, although PLC's with TC input exist.

Though, it is true that in some instances by using thermocouple wires over a long distance and achieving satisfactory results, the odds are against it, making it a risky method to try in a cement plant!

Even the idea to amplify the voltage signal (to reduce the signal to noise ratio) is not good enough since the noise picked up may be several hundred volts high.

5.1 Current output

If a thermocouple transmitter with 4-20 mA DC (or 0-20 mA DC) current output is used, instead of a voltage output, some important advantages are gained. The controlled current line eliminates losses due to the wire resistance (line losses), because the resistance of the wire merely drops voltage along the line - the current remains constant (impressed current). Also, the noise pick-up is all but eliminated by the very high noise immunity of the current line due to the very low output loop impedance.

This allows to use a twisted pair of ordinary signal wires. The wires are twisted, so that any noise that appears on the line will be on both lines. It can be eliminated by means of specific electronic circuits at the input of the upstream connected instrument. (="common-mode rejection", meaning the ability of a circuit to reject signals of equal amplitude on both input leads.)

Current signals can be collected in the field (field junction box) and transmitted to the control room with low-cost multi-core cables.

Summarising, it can be said that the beginning of the measuring range of any type of analogue measurement is represented on the transmission line by a current of 4 mA (or 0 mA). The end of the measuring range of any type of measurement is represented on the transmission line by a current of 20 mA. That means, an unscaled value in electrical units is transmitted. To produce an indication scaled in the desired physical unit the indicator has to be provided with the respective scale.

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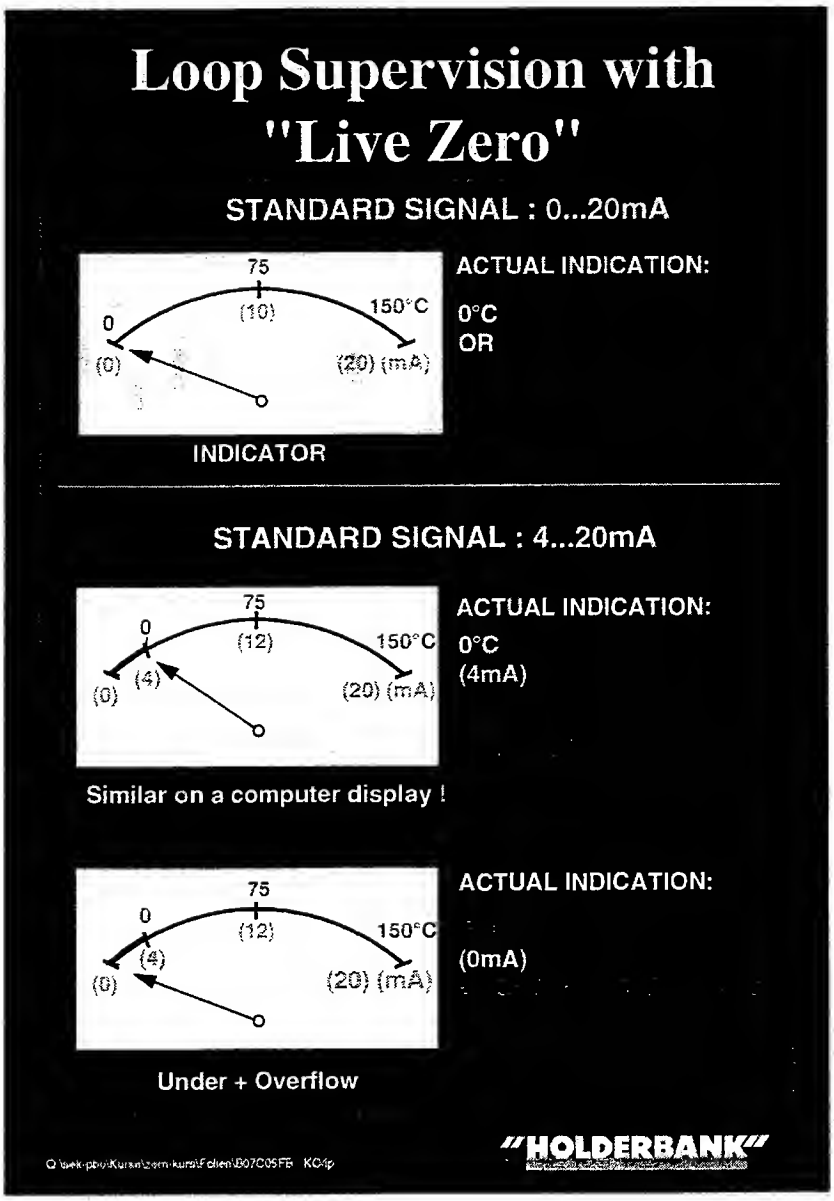
5.2 "DEAD ZERO" and "LIVE ZERO"

In a standard 0-20 mA the zero-point of the measuring range e.g. 0°C, is represented with 0 mA ("DEAD ZERO"), and the end-point of the measuring range, e.g. 150°C, is represented with 20 mA signal current. However, the signal current also becomes 0 mA (no current flow), in case of a transmitter failure, broken cable, or loss of power.

In a standard 4-20 mA the zero-point of the measuring range e.g. 0°C is represented with 4 mA (live zero) and the end-point of the measuring range e.g. 150°C with 20 mA. Therefore, for the transmission of an analogue measurement, only a range of 16 mA is available.

A signal current of 0 mA (no current flow, or a current <4 mA), can only be caused by a transmitter failure, broken cable or loss of power. Thus, the 2 cases "FAILURE" and "ZERO-POINT of MEASURING RANGE", can easily be distinguished.

An electronic circuit can monitor the measurement-loop with "live zero" and immediately generate an alarm if a failure occurs.



5.3 Power supply

A 2-wire transmitter can only operate with a "live zero* standard signal (4-20 mA), because the first 4 mA are used to supply the electrical power to the 2-wire transmitter, 4-wire transmitters are available with "live zero" (4-20 mA), or "dead zero" (0-20 mA), standard signals.

5.4 4-wire and 2-wire transmitters

4-wire transmitters need 2 wires for the transmitter operating power supply and 2 other wires to transmit the output signal to a remote location for indication or other purposes.

2-wire transmitters need 2 wires only to bring the power to the transmitter and to transmit the output signal. The basic idea is to use the first 4 mA of the output signal to cover the transmitter's power consumption and the remaining 16 mA for signal transmission.

4-wire transmitters are available for any DC or AC power supply voltage. The output signal is usually a standard signal of 4-20 mA or 0-20 mA DC. The admissible external burden can go up to 3000Ω; however, a typical burden is 500Ω.

"Zero" and "span" are independently adjustable, which facilitates the commissioning and calibration of a transmitter. To prevent electrical disturbances caused by "earth-loops" only transmitters with **galvanical isolation** between power supply, input and output should be applied. Where feasible interconnections between measurement loops should be avoided. (See also the chapter "Non-isolating and isolating transmitters".)

Summary

Goals and drawbacks of the 4-wire concept:

- "Compact transmitter", i.e. power supply is integrated in the transmitter.
- Large variety of power supply voltages possible (220 V, AC, or 110 V AC is already available at most locations of the plant).
- 4-wire transmitters can perform all measuring functions between "very basic" and "very complex".
- Independent "zero" - and "span"-adjustment.
- Very high external burden possible.
- Output signal 4-20 mA or 0-20 mA possible.
- Cabling and installation expensive, due to separate power supply cable.

The 2-wire transmitter converts the input signal to a standard output signal of 4-20 mA and receives its power from the same 2 wires.

The output signal consists of two components:

- The 4 mA component (also called "live zero"), is a constant drain on the remote located power supply. This current is used to provide operating power to the transmitter.
- The second component, 0-16 mA, is a variable drain on the power supply that is proportional to the transmitter's input signal, which may represent a temperature, pressure, flow etc.

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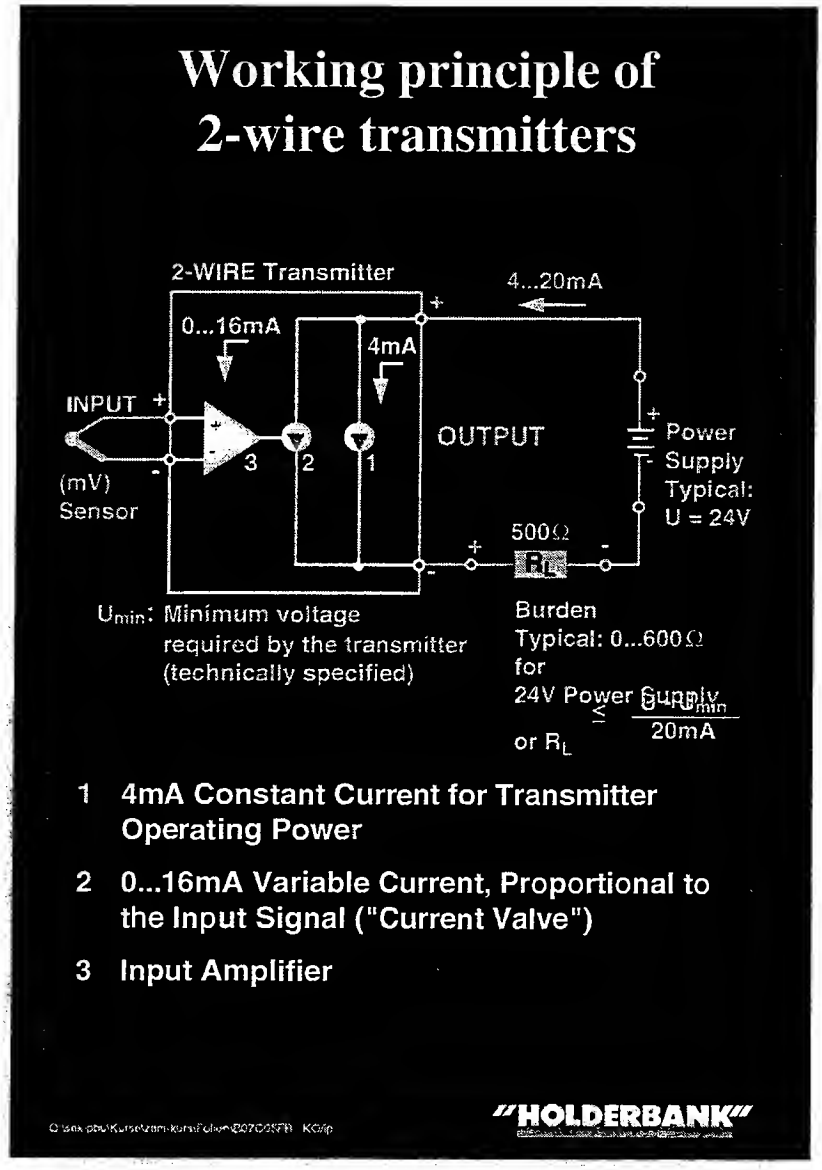

The sum of these two current components results in a 4-20 mA current that flows in the measurement loop at the transmitter's output. It is obvious that 2-wire transmitters may operate only with the "live-zero" standard signal 4-20 mA. The typical measurement loop in 2-wire technology shows the power supply, the transmitter and the receiving elements, such as analogue inputs, indicators, recorders etc., connected in series with the loop.

Usually the power supply is located in a "clean room", e.g. in the control centre, however, the transmitter in a dust and water-proof housing is placed "in the field", next to the detecting point.

2-wire transmitters using the most advanced electronics technology may operate with power supply voltages between 12 V, DC and 50 V, DC. The admissible maximum burden connected to the output loop depends on the power supply voltage.

In the 2-wire mode it is technically impossible to provide independent adjustments for "zero" and "span". Therefore, calibration and maintenance is slightly more time-consuming and needs more experience compared with 4-wire transmitters.

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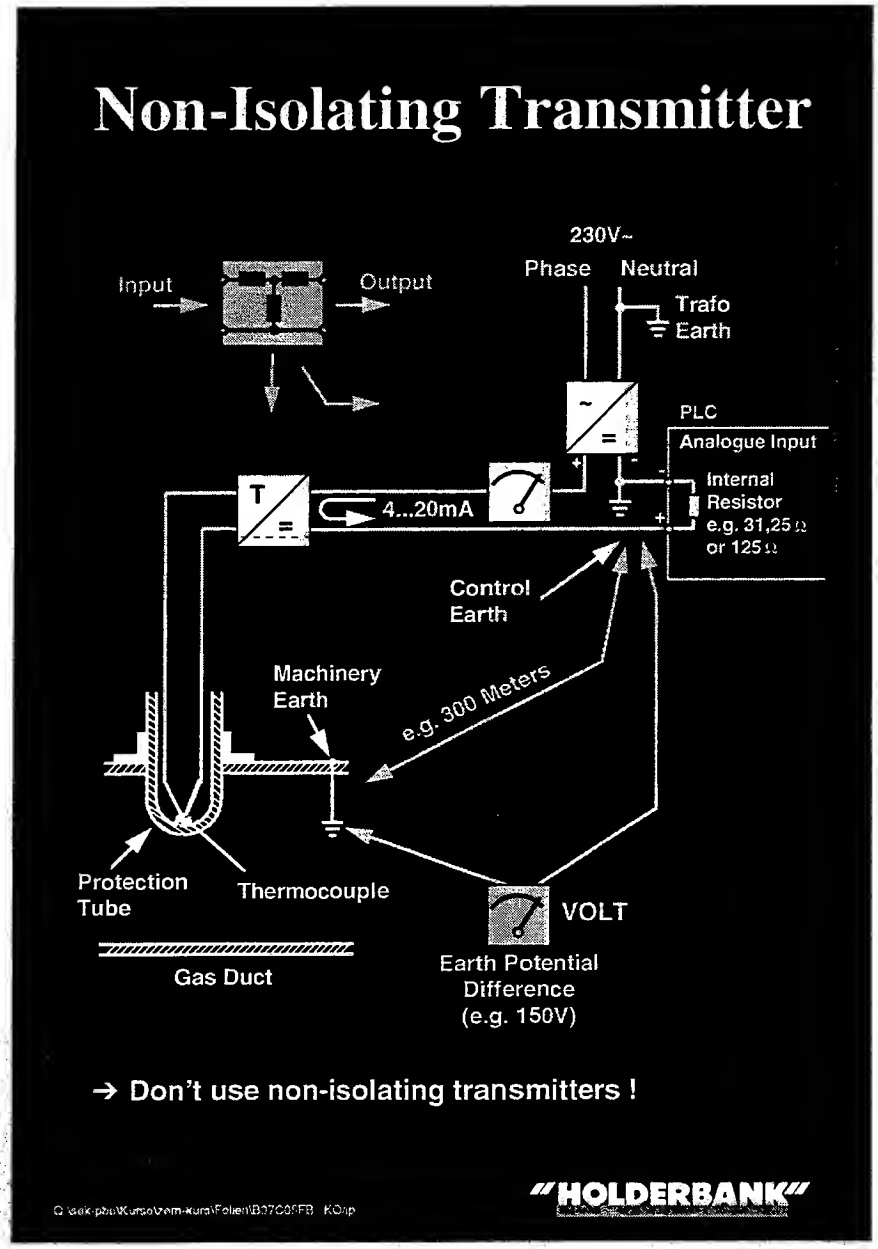
5.5 Non-isolating and isolating transmitters

While driving grounding rods into the ground at two points several hundred meters apart and connecting a voltmeter between them, a voltage difference becomes noticeable.



This potential difference exists between practically any two points along the earth's surface. From this resulted voltage problems can cause when trying to measure a process at a remote location.

It is necessary to ground the sensor at the remote site to reduce noise and to protect the equipment from damage caused by lightning. But if grounded thermocouples are used and if it's tried to ground one side of the transmitter output loop at the control room, the voltage difference between the two points will induce an error current along the line, resulting in an erroneous measurement indication or in equipment damage!

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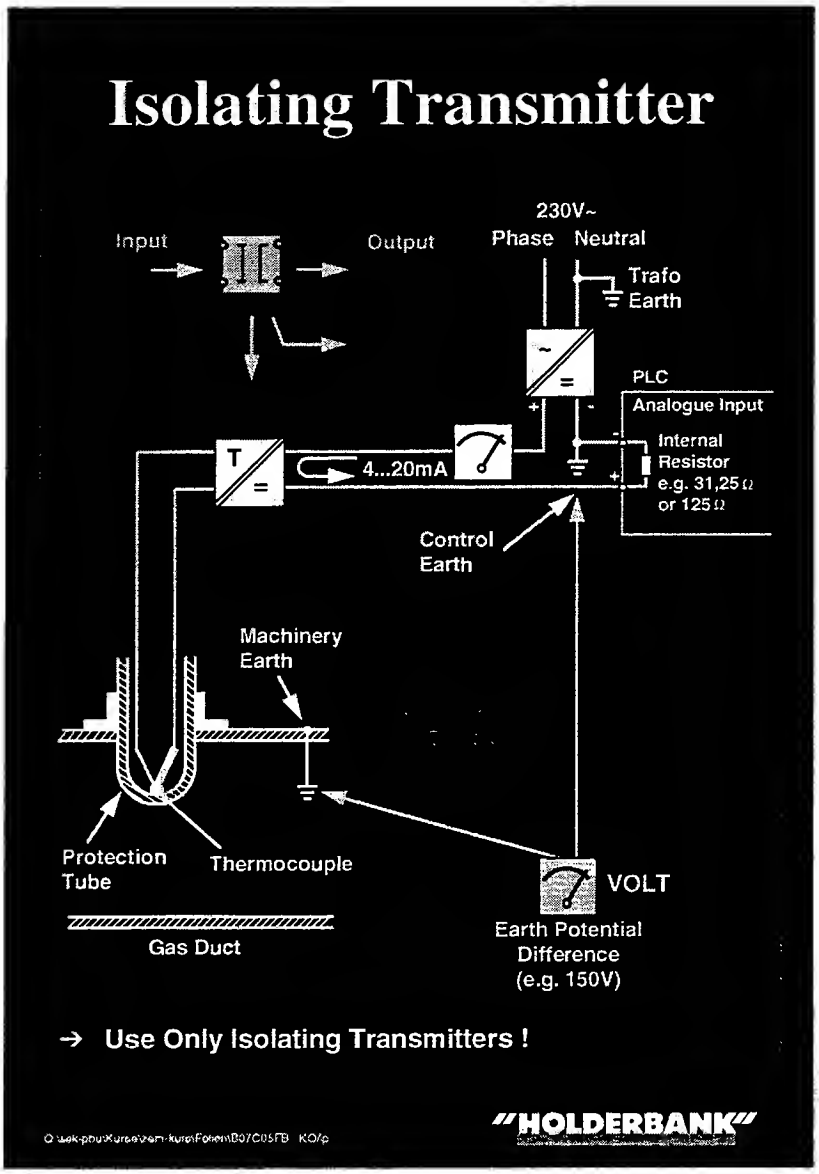
To eliminate this "ground loop", an isolating transmitter can be used. This type of transmitter electrically isolates the transmitter's output loop from the sensor signal as well as - in case of a 4-wire transmitter -, from the power supply, and allows to ground both, the sensor and one side of the output loop.



Everybody knows: a transformer can transform only alternating current (AC). Actually, that's the reason why direct currents (DC) are first converted into AC, then transformed by the transformer and finally rectified again to obtain DC, reproducing exactly the DC at the module's input.

Recommendation, Conclusion

- The induced error current, caused by the earth potential difference or by lightning, can cause erroneous measurements and equipment damage!
  - Do not use non-isolating transmitters!
- The galvanical isolation in transmitter and power supply opens the induced error current path!
  - Use only isolating transmitters! Galvanic isolation between signals and power supply.

6. CONTROL, ALARMING AND DISPLAY

When the signal at the desired location (e.g. central control room) arrives it has to be further treated either for display, recording, alarm or control. In a modern cement plant the 4-20 mA are fed directly into the Process Station (PS). This is the simplest method since any measurement can be used for any purpose without any further effort provided a good standard and user software is installed in the PS. How the signal is treated further can be red in chapter "Motor Control".

7. MEASUREMENT USED IN THE CEMENT INDUSTRY

The cement industry uses in most cases common instruments but faces some cement specific problems. As shown in the instrument list, the cement industry applies not too many different types of sensors, respective measuring principles. However the tendency is increasing.

Problems are encountered mainly with high temperature, clogging and coating. Many of these problems can be evaded by selecting the proper instrument, respectively primary element and/or picking a suitable location. Maintenance and regular calibration avoid break downs and prolong the lifetime of the primary element.

Note: To facilitate easy maintenance, accessibility of the primary elements and the transmitter is vital.

Cement specific sensors and measuring systems are illustrated in the next chapters.

7.1 Temperature

In the cement industry generally thermocouples, PT 100 resistance bulbs and pyrometers are used. For kiln shell measurement, temperature scanners are often applied together with a display system. These scanner systems range from a simple pyrometer connected to a recorder or from a scanner head connected to a PC with an elaborated software giving information about the shell temperature, interpretation about the inside of the kiln and even a brick management and slip detection can be included.

7.1.1 Thermocouple

Mostly applied for temperature measurement in the cement industry are thermocouples which use the peltier effect as measurement principle. A thermocouple consists of two dissimilar metals. Between these metals a voltage is generated. The electro-motoric force (emf) developed by a thermocouple depends on the temperature of both, the measuring (hot) junction and the reference (cold) junction.

Important for a thermocouple is therefore:

- the type of thermocouple and the manufactures data (e.g. type K thermocouple)
- the extension cable of the appropriate type (e.g. type K for type K thermocouple)
- temperature range and type of thermocouple (e.g. type K for 200 - 1200°C)
- cold junction reference temperature 0°C, 20°C or others.
- the type of protection sheath. The length of the TC as well as the protection tube should be standardised (e.g. 800 mm and 1200 mm). The material for the sheath may however be different for various applications since a high temperature protection sheath is very expensive.

To calibrate a transmitter for a thermocouple, a mV source is required. This source is connected in the measuring loop instead of the thermocouple. According the manufacturer's

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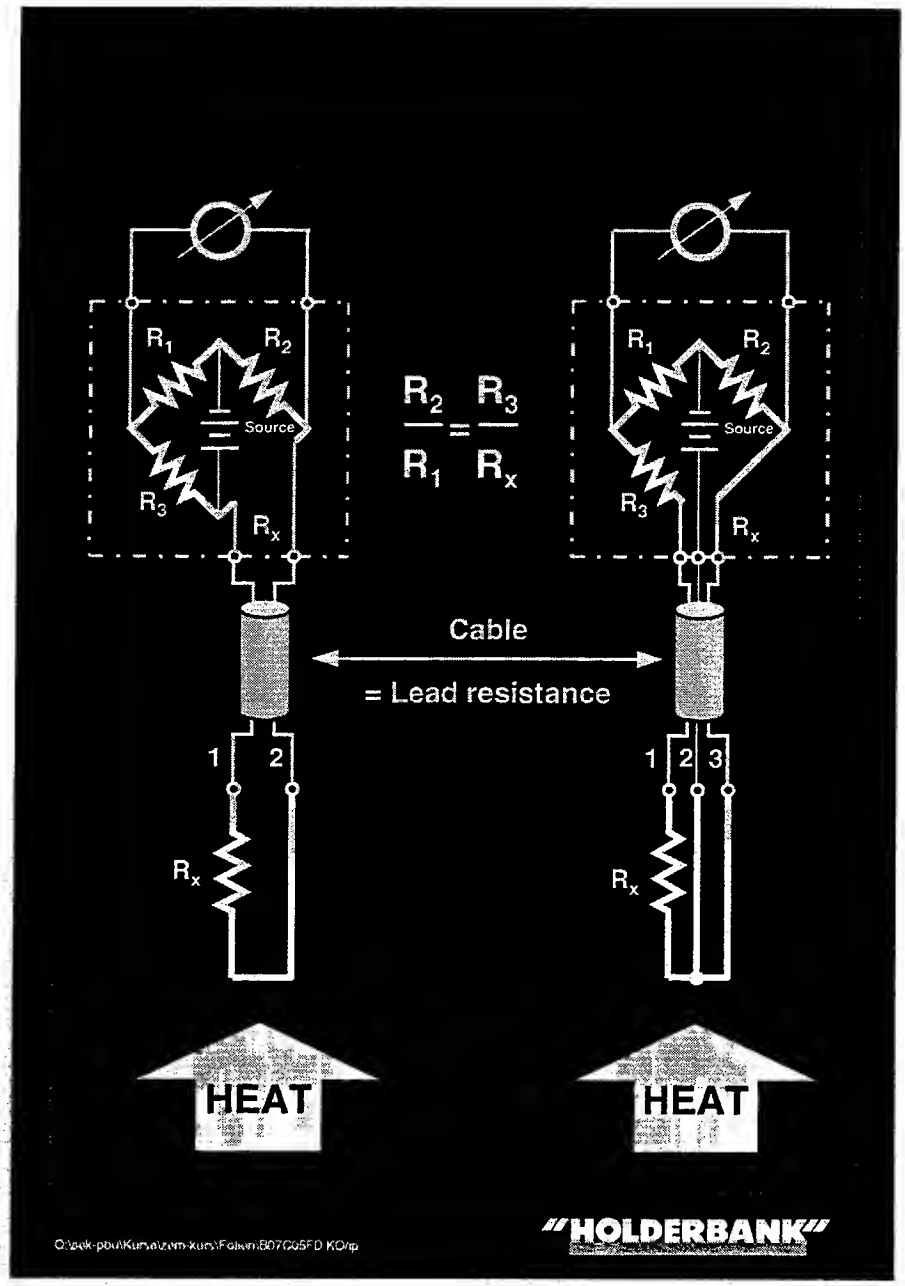
data sheet, mV for 0°C and mV for the maximum temperature are fed into the loop. Then the output of the transmitter 0°C = 4 mA and max. temperature = 20 mA are checked as well as the display in the control room together with any alarm limits. The temperature range from -200°C up to +2000°C can be covered with thermocouples.

7.1.2 Resistance bulb RTD PT 100

For lower temperature (e.g. for machine protection) resistance thermometers are used. RTD's work on the principle of a resistance changing when the temperature varies. Mostly used in the cement industry is the Pt 100 platinum resistance bulb. The PT 100 has a resistance of 100Ω at 0°C and 158.8Ω at 150°C. When selecting a resistance bulb, it is important to specify a 2-, 3- or even a 4-wire type. To compensate for the line resistance a 3-wire type normally is sufficient. If the transmitter is installed nearby, even a 2-wire bulb is good enough.



To calibrate a transmitter for a PT 100 a resistance decade is required. This resistance box is connected in the measuring loop instead of the PT 100. According the data sheet the

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base resistance (e.g. 100Ω for PT 100) for 0°C and the maximum resistance for the maximum temperature (e.g. 158.8Ω for PT 100) are fed into the loop. Then the output of the transmitter 0°C = 4 mA and max. temperature = 20 mA are checked as well as the display in the control room together with any alarm limits. PT 100 transmitters require initial calibrations of the entire loop. After that only an occasional check up is required. With PT 100 bulbs temperatures from -250°C up to +1000°C are measurable.

7.1.3 Pyrometer

The cement industry uses two types of pyrometers. The radiation pyrometer which detects radiation through an optical lens system onto the thermopile or photo cell and the two colour ratio pyrometer which compares the ratio of the radiation intensity of two different wave lengths.

7.1.4 Scanner

The central feature of a scanner is a motor driven optical system which scans the entire kiln with a certain frequency (e.g. 16 Hz). The front of the scanner measures parallel along the axis of the kiln, in the back of the scanner a reference temperature is used to calibrate the system.

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7.2 Pressure

Pressure can be measured either as liquid column (e.g. U-tube) with a mechanical principle (e.g. diaphragm, burden tube) or electrically (e.g. piezo crystal, strain gauge). The pressure measured in the cement industry is usually very low. Therefore, the differential pressure (gage pressure that is the difference between the absolute pressure and the atmosphere) is measured. Most pressure transmitters applied in the cement industry are of the mechanical type where one side is connected to the process and the other side is left open to the atmosphere.

Drawing F44570-1 is a typical example of a differential pressure measurement. The bellows are subjected to a pressure change and move, via mechanical links, a plunger in an electrical field. The electronic senses the movement and converts the change in the field into an electrical standard signal.

To avoid problems with pressure transmitters some installation points have to be observed:

Location:	locate the transmitter near the pressure tapping easy accessible. Mostly applied as unit by the manufacturer.
Process line:	the pressure tapping can be above or below the transmitter. In any case the process line must be installed such that no water or dirt can accumulate. Thus, always have at least 2% slope in the process line. If the tapping is above the transmitter a water trap is required.
Tapping:	install a tapping in such a way that cleaning is easily possible. Thus, a simple removable cover should allow pocking of the process tapping. In large ducts two or more tappings connected with each other give a better result.

To calibrate a pressure transmitter a pressure source is required. This source is connected in the measuring loop on the primary side of the transmitter. To check the zero both sides of the transmitter are left open to the atmosphere. To check the maximum, an equivalent pressure (or vacuum) is applied. The output of the transmitter 0 kPa = 4 mA and max. pressure = 20 mA are checked as well as the display in the control room together with any alarm limits.

7.3 Flow (gas and liquids)

Flow measurement is thoroughly discussed in the corresponding paper of the process technology department.

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7.4 Level

Level measurements, whether continuous or just as level alarms are often applied and often not 100% satisfactory. A poll carried out in 1992 querying the performance of the different level measurements in some 30 factories showed results from very satisfactory to useless. Depending on the method applied, the installation and the maintenance the same level measurement is rated different with regards to performance. The following level measurements are used successfully in the cement industry. Some mechanical types like paddle, ball etc. are not explained since their function is very simple.

7.4.1 Capacity probe

The capacity probe uses the measuring principle of two plates being isolated by a dielectricum ε where the capacitance depends on the area of the plates, the distance between the plates and the medium between the plates the dielectricum. When installing a capacity probe, the plates (the probe and the silo wall) are fixed and the distance is fix provided, the probe is mounted properly. The variation in the measuring loop is therefore the dielectricum ε which is formed either by material between probe and wall or air. The following problems may occur thus hampering the performance:

- sticky material on the probe. This can be avoided using the proper type of level probe that is insulated or partly insulated probes.
- probe installed too close to silo wall thus evoking bridging.
- sensitivity adjusted to fine. Humidity in the air or in the material changes the property of the dielectricum.
- mechanical damage may result from coarse material or from sheer force. (E.g. coal dust has very high sheer force). Use an other measuring principle or use an insulated capacity probe.

7.4.2 Vibration

The vibration fork level probe is only used for a single point measurement. A driver induces a vibration in the probe and a controller senses a change when material dampens the vibration. The tuning fork is obtainable in two forms, one being the actual fork and the other in form of a tube. The following problems may occur thus hampering the performance:

- material stuck between the fork. Use a tube type to avoid this problem mostly occurring with coarse material.
- material coating the probe. Use the fork type probe since this problem takes place mainly with fine material.

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7.4.3 Electro mechanical

Quite successful practice is the level measurement with the so called silo pilot. A rope or measuring tape connected to a weight is lowered into the silo. As soon as the weight touches the material surface, the rope (tape) tension ceases, the motor reverses and pulls the weight back into its original position. During the upward travel the tape is measured, such giving an indication of the level within the silo. The following points have to be considered by using a silo pilot:

- position on top of the silo so that no material can fall onto the weight. When lowering the weight it should be in the centre of the material cone.
- access to the rope respective belt and the weight must be easy. Install an inspection door to assist maintenance.
- select the proper weight for the corresponding material

7.4.4 Contactless level probes

Several methods allow level measurement without being in contact with the material. Ultra sonic is the most popular and is used with coarse material, e.g. gypsum, limestone. In connection with dust in the sonic beam or on the material surface <40°C, e.g. cement, raw meal, this method should not be applied. Level can be measured up to 45 m under good conditions. Rather new on the market are infrared and radar. With infrared no experience has been gained. Good experiences have been gained with radar in environment where ultrasonic fails. In clinker and in raw material silos the measurement with radar proved to be successful even with high dust load. In raw meal the measurement did not work and the problem seems to be the conductivity. Only material with a certain conductivity respective a low dielectricum can reflect the radar beam. Up to now only measurements for 35 m depth are available. This is due to the strength of the source which must be within the limit of the wireless regulation of the respective country. The radar method works in a temperature range of -40°C up to +250°C.

7.4.5 Radiation level probes

Well-known and well-proven are the nuclear type of level probes. From a measurement technical point of view no restrictions are known. In most cases where all other methods fail the nuclear level probe serves well almost maintenance free. In many countries, however, importing or handling the source is very difficult. Additionally the disposal of the source is in many cases almost impossible and requires a lot of responsibility of the person in charge. And more restriction for the future are to be expected.

An alternative (even in the pre-heater cyclones) offers the micro wave level measurement. The arrangement is similar to the nuclear level measurement a microwave source on the one side and a detector on the other side. The microwaves beam of approx. 20° angle, some 5.8 GHz or as high as 24.125 GHz penetrates any non-conductive material. The maximum distance through air is around 8 m.

Alternative measurements around the corner are possible if the space is limited. The installation must be such that a light beam would be properly reflected.

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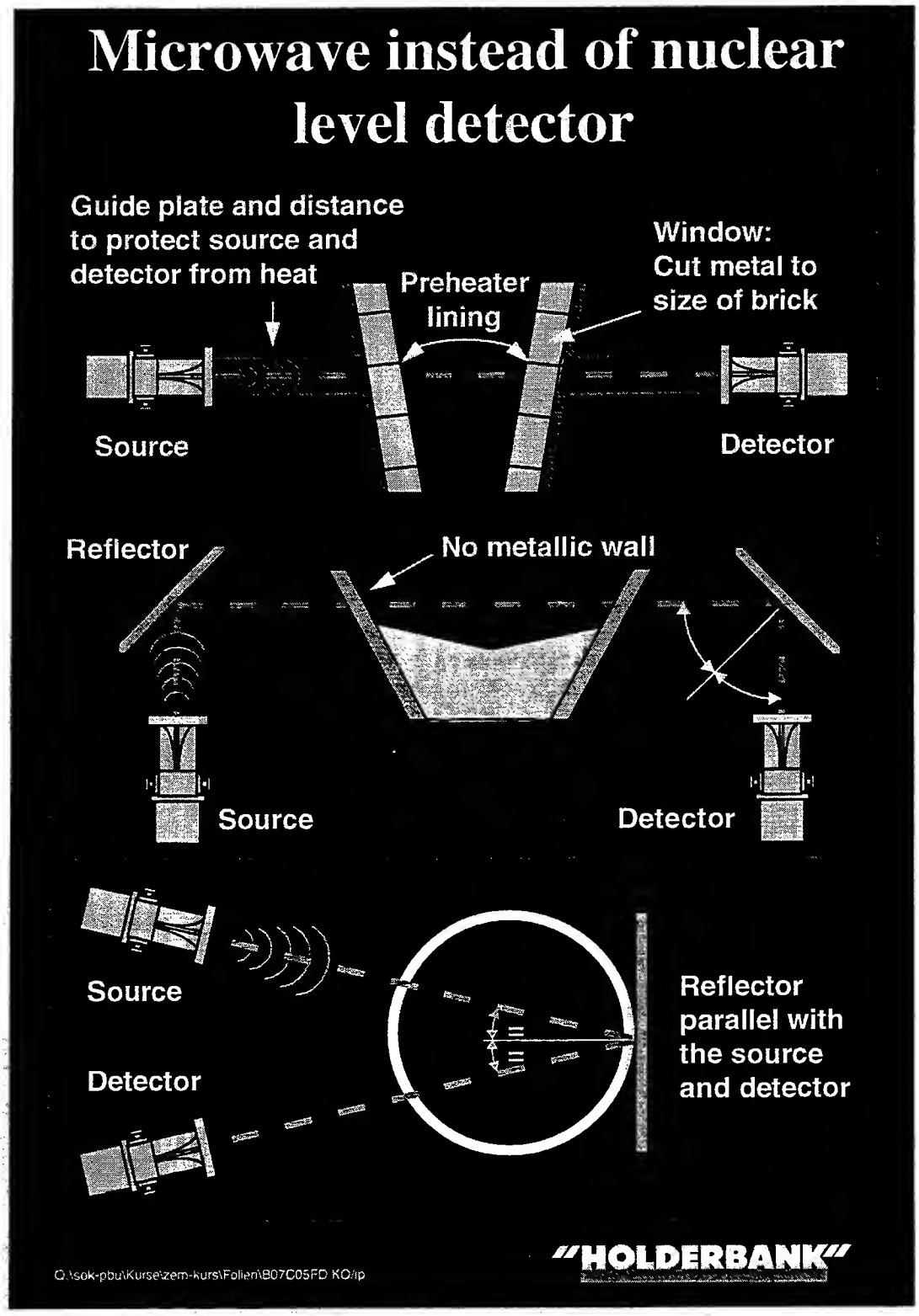

Microwave level measurements are sensitive to moisture. In fact, microwave is used as well for moisture measurement. The microwave source is so weak, that no danger comes from the measurement. (No cooking can be done with this microwave source, the energy emitted is less than 25 mW.)

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7.5 Weighing

Weighing and weigh feeder play an important role in the cement industry. To produce a good cement quality, accurate weigh feeding of the different components is important. To bill the customer agreeable again, weighing plays the key role. Already these two important requirements show that for weighing two different ways are possible: the static weighing (weigh bridge) and the dynamic weighing (weigh feeder).

The weigh bridge at the factory entrance is regarded as the most important and accurate unit. The weigh bridge is in many countries subjected to stringent government regulation and must be checked and calibrated on a regular base. For weigh bridges the measuring principle applied is usually one or several load cells. Due to the strict government roles and due to the simple measurement principle these scales usually are fairly accurate.

Small bins are as well placed onto load cells to weigh the contents. And, as long as the construction is suitable, that is three load cells, free moving construction and protection against wind, the measurement can be very accurate. The total weight (xy tons in the bin) may not be accurate but loss in weight is accurate. Thus, for calibration of a weigh feeder or for volumetric weigh feeding a loss in weight is well-qualified.

The next important weighing principle is the continuous measurement of material to constantly feed an accurate amount of material, the dynamic weighing with weigh feeder. Several principles are applied with different accuracy, different efforts for maintenance and different prices.

Thus when selecting the weigh feeder the following points must be taken into consideration:

- Accuracy required.
- Mechanical suitability for the material and the environment.
- Space availability (height and area). Especially building height can be reduced (cost saving) with certain weighing arrangements and weighing principles.
- Signal availability and signal transmission. (4-20 mA and digital signals or communication via a bus system).
- Maintenance that is time interval between calibration, access for calibration e.g. re-routing of material onto a lorry, cleaning required, complexity of the control, spare parts etc.
- Measurement principle.
- Silo discharge system. Most problems of inaccurate weighing arise from poor flowing material.

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7.5.1 Belt weigher

The most common measuring principle applied in the cement industry is the belt weigher. A section of the belt runs over idlers supported by a frame section placed onto load cells.

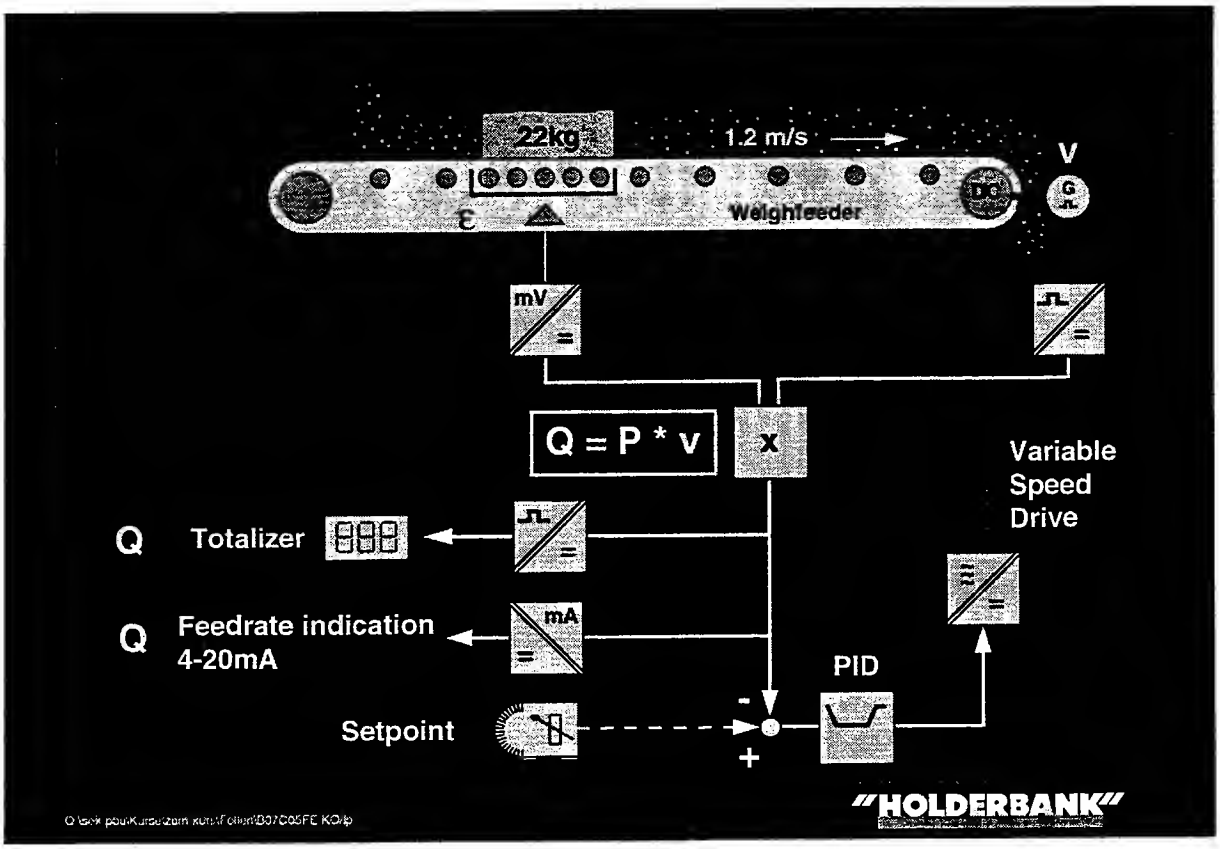
The weight over this belt section multiplied with the speed represents the feed rate Q = P * v whereas:

Q = feed rate [t/h]

P = weight per width [kg/m]

v = speed [m/h]

This principle is well-known and if maintained properly very accurate (error < 1%)



Also different methods of calibration are offered, but only the following are accurate and repeatable: Run for e.g. 5 minutes material onto a lorry, weigh the lorry and calculate the feed rate weight x 12 (if the calibration time was 5 min.). Thus when designing the weigh feeders a means to calibrate onto a lorry must be included or alternatively weigh bins above the feeders to calibrate with the loss in weight method which is a very accurate method too.

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7.5.2 Gravimetric feed system

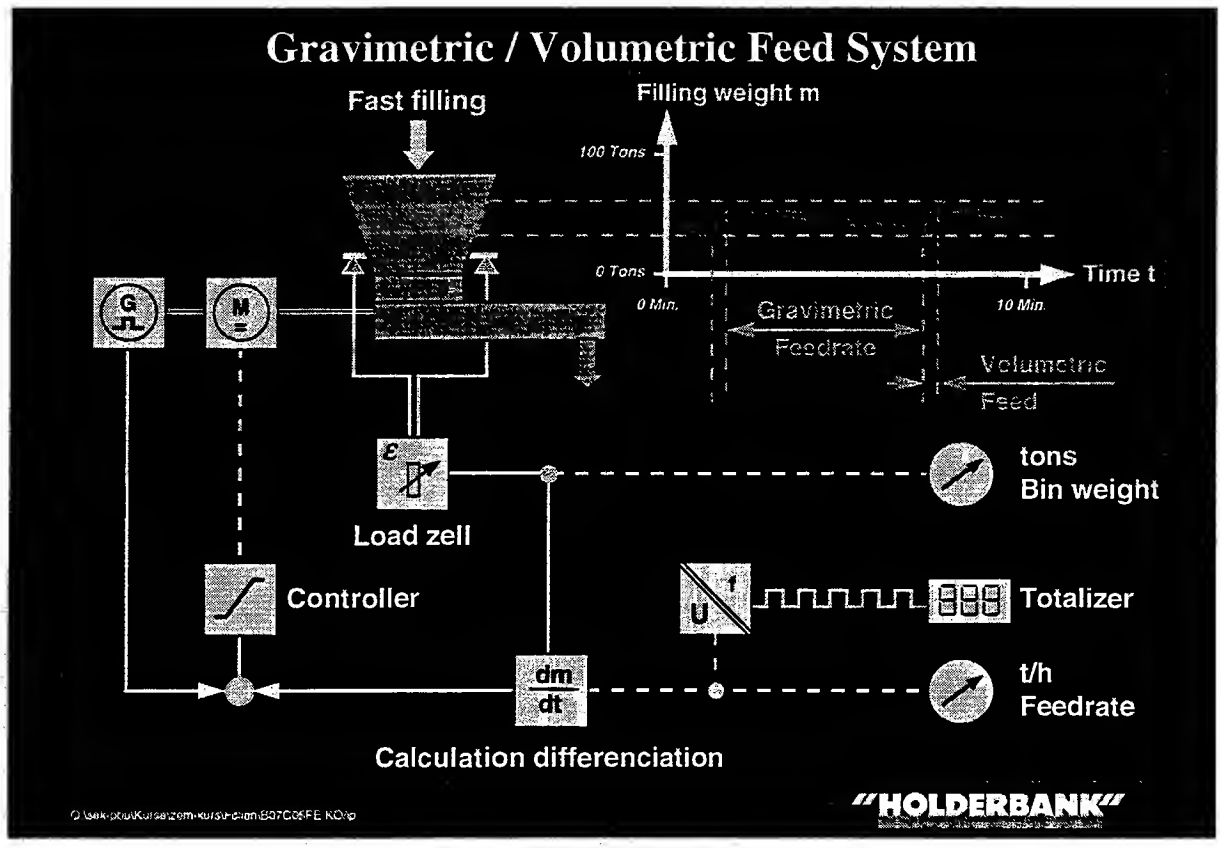
A gravimetric feed system is mainly used for coal feeders. This weighing system is complex but accurate. A bin on load cells is rapidly charge with material. The filling then stops and for n seconds the bin is emptied. (t_n = discharge time depending on bin size).

When reaching a certain low level the bin is recharged again and during this time the speed of the discharge feeder is maintained at the previous feed rate. The feed rate Q is calculated by the formula:

$$Q = \frac{m * 3600\ s/h}{t_n}$$

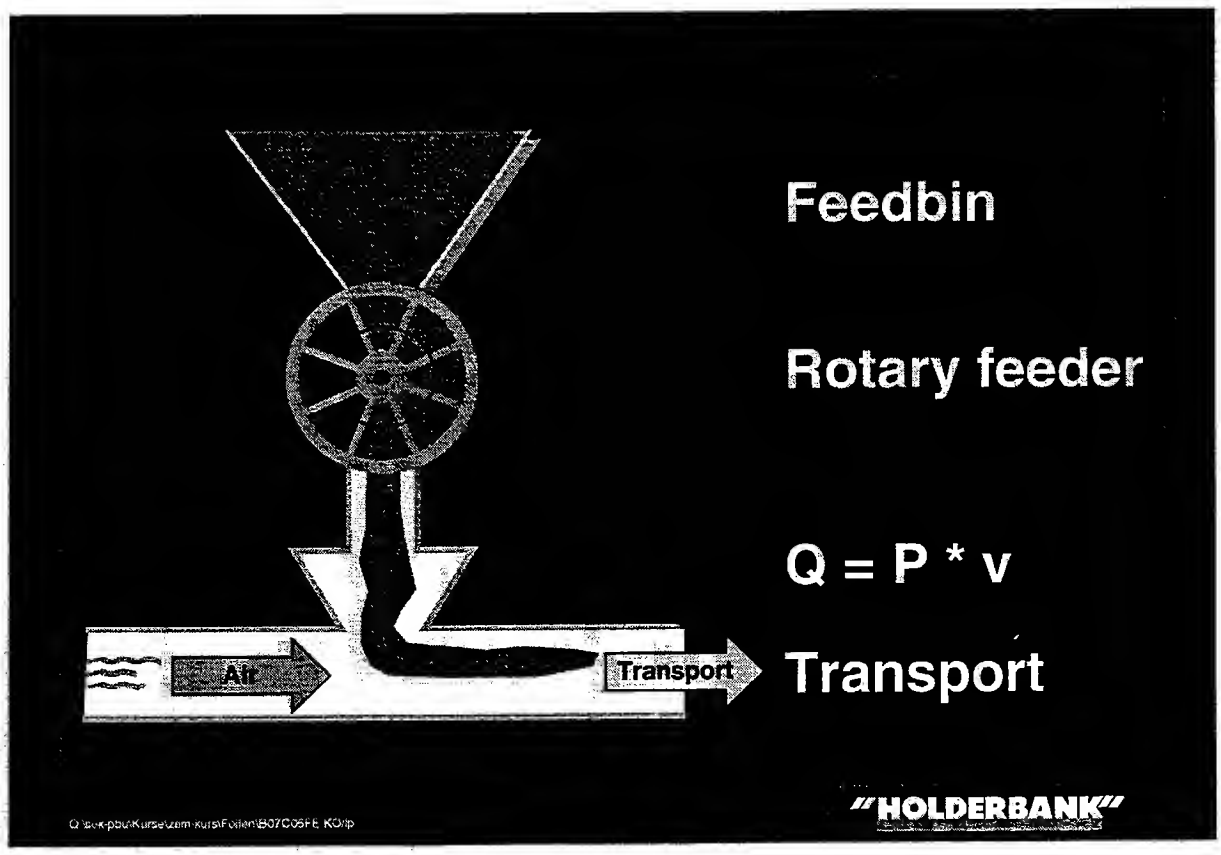
m = loss of weight during t_n [kg]
t_n = discharge time [s]

Modern systems calculate the feed rate during a very short time period and take the whole time to calibrate the system. The accuracy of such a system is < 1%. The maintenance however is high and the system is complex.



7.5.3 Volumetric feeders

A simple method is a volumetric feeder. The accuracy is largely dependent on the feeder itself and on the flow property of the material. Depending on the required accuracy, a periodic calibration onto a lorry is needed. If a higher accuracy is required a weigh bin has to be introduced prior to the feeder and a similar measuring method as mentioned in the gravimetric flow measurement has to be applied.



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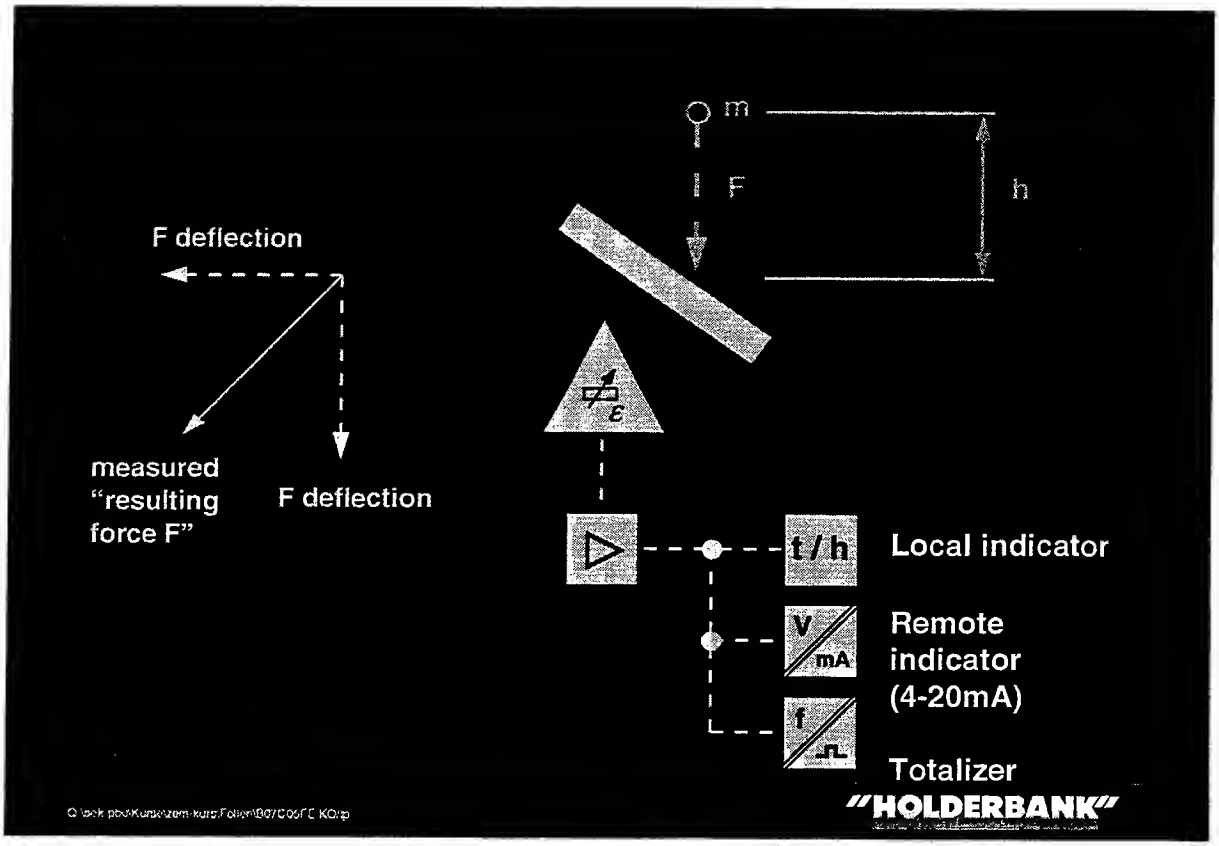
7.5.4 Impact flow meter

A controversial measurement is the weight measurement with an impact flow meter. Material flows on to a plate which in turn is placed on a load cell. The impact on to the plate is proportional to the impact created by a mass falling from a height h. The impact is in practice also depending on the flow properties of the material which in turn is depending on various factors. Thus, this measurement requires a fair amount of mechanical design around the actual measurement. Dedusting, the feed to and away from the impact flow meter are crucial for the accuracy of the measurement.



In practice the impact flow meter is mostly applied together with a pre bin arranged with load cells. This arrangement allows an automatically periodical calibration of the flow meter, in order to compensate sticking material at the plate.

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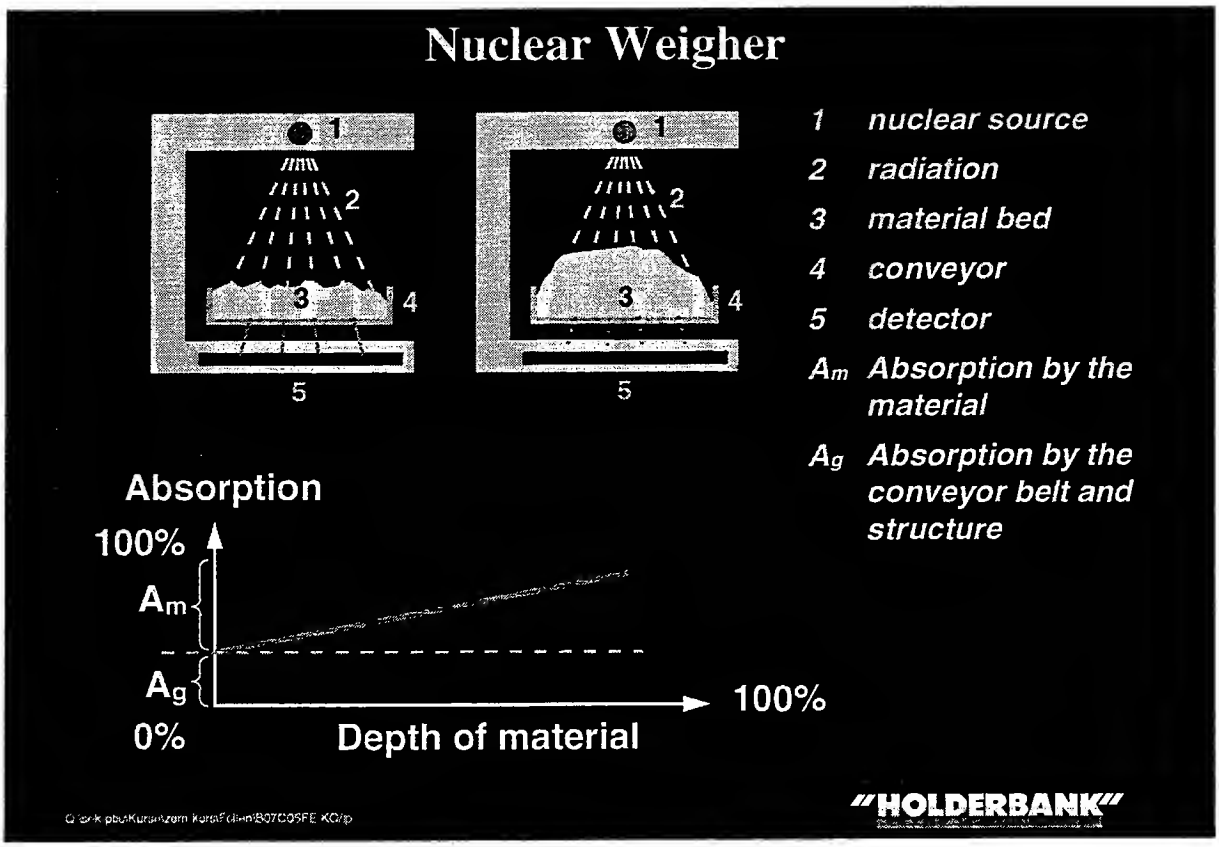
7.5.5 Nuclear weigh feeder

Weighing with nuclear weigher is nothing new in the cement industry but is and remains a controversial measurement. Although, a simple and reliable measurement the import and the disposal of the nuclear source is problem. A nuclear weigher consists of a gamma source and a gamma ray detector. Both are connected on a mechanical frame which is located across the conveyor. The beauty is that almost any conveyor (e.g. belt, apron feeder, screw conveyor etc.) can be fitted with an A or C frame nuclear weigher. Even in an existing installation does the installation of an A or C frame seldom present a problem.

A nuclear weigh feeder determines the weight by measuring the absorption of the material. Every material absorbs radiation according the exponential law. The absorption is proportional to the thickness and the density of the material bed. The absorption is then related to the mass of the material which, when multiplied with the speed, results in the mass flow.

The main absorption (basic absorption Ag) is in many cases given by the construction. The absorption with a given density is then proportional with the bed thickness (Am). To receive good results the absorption Ag should not be larger than 95%.



Due to a less thick material bed (3) the absorption measured by the detector (5) is lower. The absorption (2) by the conveyor (4) is remaining constant and calibrated as zero.

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When installing an nuclear weigher the following points should be taken into consideration:

- import regulation for nuclear devices
- disposal of nuclear sources
- building and conveyor construction
- basic absorption (Ag)
- electronic with source decay compensation
- even material flow (an irregular bed which influences the measurement negatively)
- the initial accuracy is around 2% remaining constant even with hardly any maintenance.

7.5.6 Head flow meter

In an air lift the air pressure measured is more or less proportional to the amount of material transported. However, pressure influence other than the amount of material is coming from dedusting and from pressure changes from the system following the air lift (e.g. pre-heater and kiln). The head flow meter as an indication to the material flow is quite acceptable.

7.6 Analytical measurements

For further information, please refer to the appropriate paper "Quality assurance" in the material technology II.

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7.7 Electrical energy and power measurements

7.7.1 Introduction

In the cement industry, the topic "Energy" will become more and more important. - In the past until today the price for electrical energy is still low enough that nobody cares to much. But the near future will show us the opposite. The energy consumption will still increase but the energy production cannot follow this rising demand. Thus a bottle-neck will occur. The power companies have already started to think about increasing energy prices and how to introduce new tariff structures (Energy Exchange). Consequently, we have at least to stabilise our energy consumption or, even better, decrease the consumption. The first step will be to **measure** before other steps can be taken. - The following chapter treats this topic.

7.7.2 Definition of energy and power

What is electrical energy?

Energy, generally, is stored work or the ability to perform work. Electric energy (W_{el}) is potential energy or expressed in an other way, the product of electric voltage (U) and electric charge (Q). The electric charge can be replaced by the product of current and time ($W_{el} = U * Q = U * I * t$; whereas I = current and t = time). The electric energy is comparable to energy of position in the mechanic that is:

- potential difference: height -> voltage U.
- quantity: weight -> electric charge Q.

The unit is volt-ampere-second [VAs] or in a more practical way kilo-watt-hour [kWh]. One kWh is equal to 3.6 MJ [860 kcal].

What is electric power?

To express performance the work to complete is related to the time required to do it. Similar, the more powerful a machine is, the more work can be done in a shorter time.

Thus, the power is proportional to the work and inverse proportional to the time to complete the job. The electric power (P) is the product of voltage (U) and current (I). At the first sight, there is no time any more in the formula, but the definition of the current is the relation between the transported charge quantity and the time. The electric power is comparable to the mechanical power. For example, the power of a hydro power plant is dependent on the height of fall (voltage) and the flux (current). - The unit is volt-ampere [VA] or more practical kilo-watt [kW].

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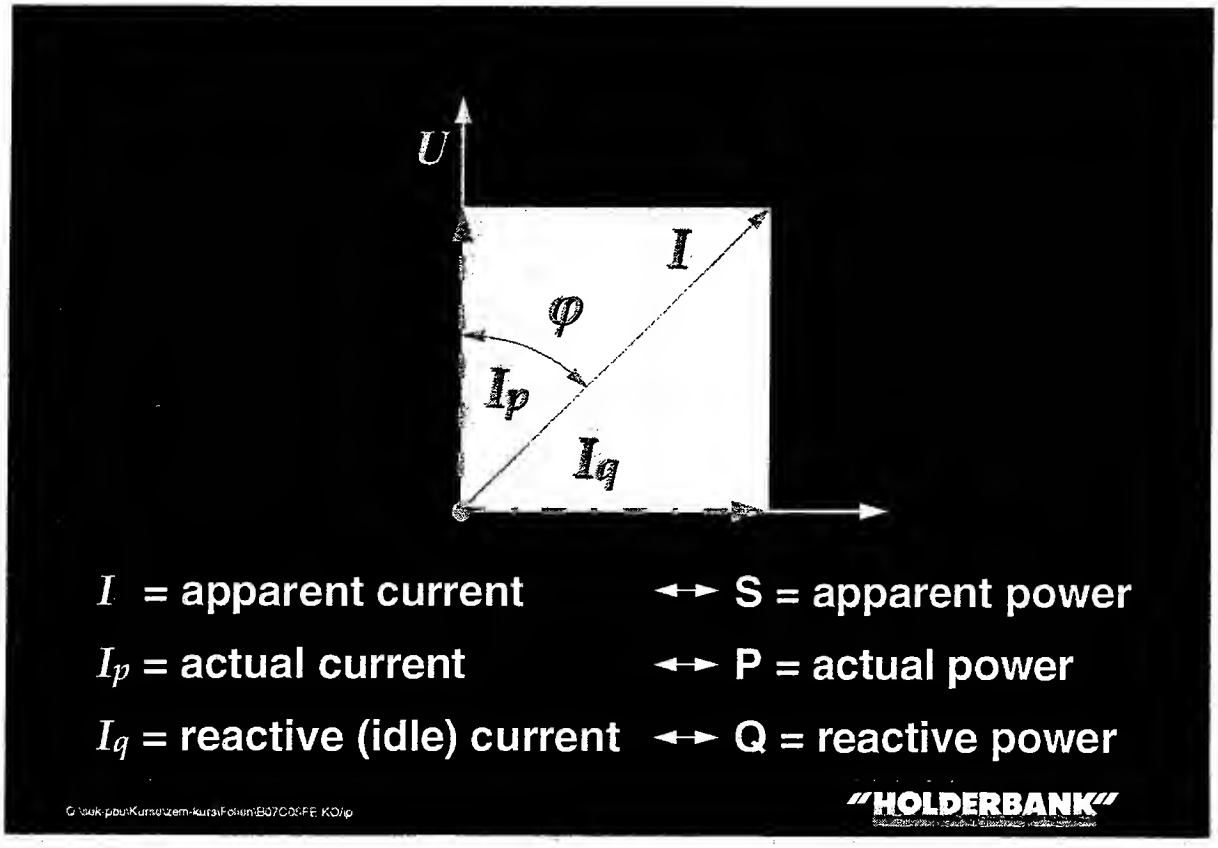
Apparent, actual, reactance

The power companies bill in most cases the actual energy only. To avoid producing too much apparent energy, they demand a power factor correction between 0.87 and up to 0.9. Few power companies ask for a reactance energy meter and if certain values are surpassed adjust the bill accordingly. Some power companies bill the client already according the apparent energy (measured actual and reactive energy). And this may be the future method of accounting the client since a power company has to produce the apparent power and not only the actual power.

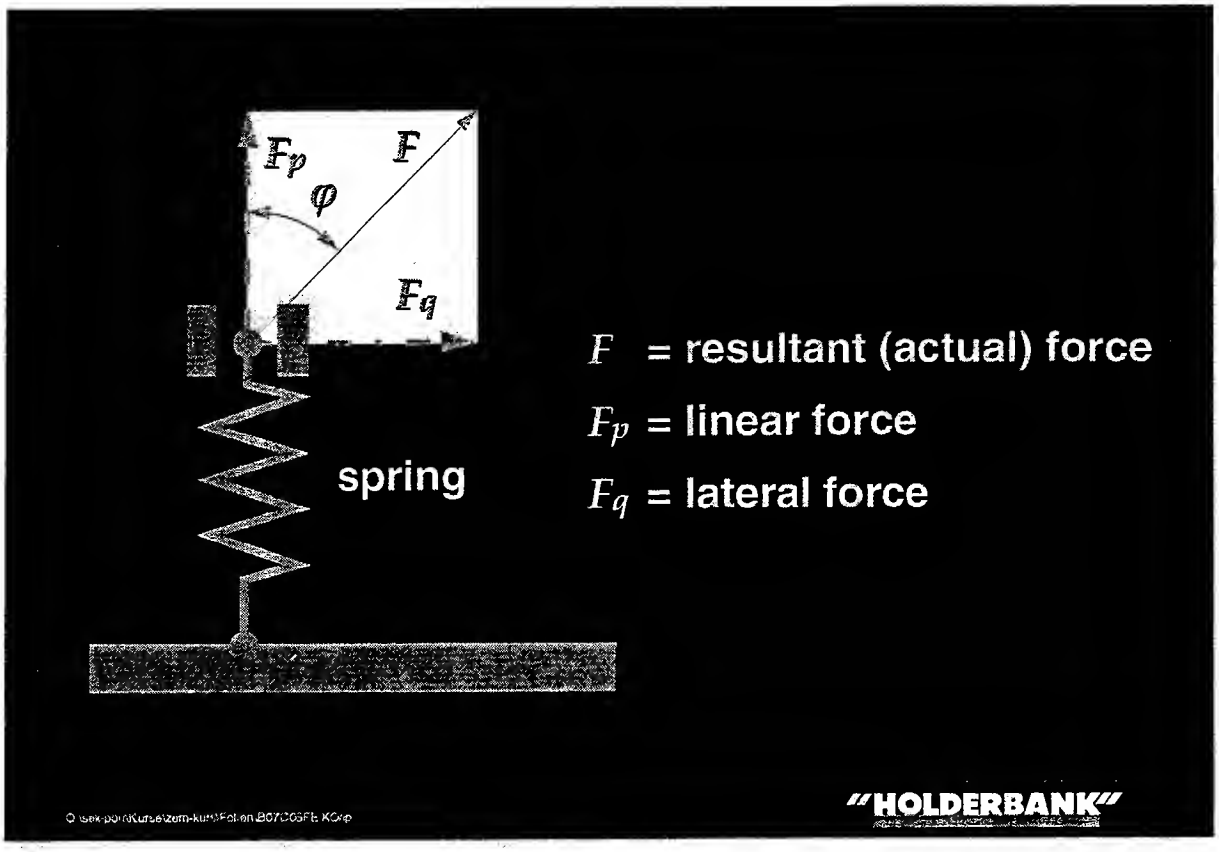
Using a mechanical diagram the three electrical components: apparent (resultant), actual (linear) and reactive (lateral) are explained.

The following picture explains the three different electrical components, apparent, actual and reactive current. Whether current, power or energy is used is immaterial for this example.



When looking at the force diagram the energy required to pull the spring is obviously the linear force F_p (electrical actual force). The spring is subjected to the following two forces:

- The lateral force F_Q (electrically reactive force) which is not required.
- The resultant force F (electrically apparent force) has to be produced in order to gain sufficient momentum to pull the spring apart.



When measuring an electric circuit with a transducer the actual power is measured which is required to drive a machine. The power company in turn produces the apparent power. If the cosine φ is multiplied with the apparent power, the result is the actual power. The difference between the produced and the consumed power is the reactive power.

Apparent power	S = U * I	[VA] or [kVA]
Actual power	P = U * I * cos φ P = S * cos φ	[W] or [kW]
Reactive power	Q = U * I * sin φ	[var] or [kvar]

Why measure?

As already mentioned, Electrical Energy centributes 20-30 % of the cement production cost. Thus electric energy is important factor as a **production component in the cement industry**. It is therefore imperative to measure the energy consumption in order to be able to determine solutions to save energy and thus maintaining the costs at least at the present level.

The energy consumption [kWh] and the specific energy consumption [kWh/t] are even units to compare for example departments in different plants or different types of machines (e.g. ball mill/vertical mill) to each other.

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Where to measure?

Drawing E310015 shows the process structure in a cement plant which is ideally represented with the corresponding electrical structure and thus the ideal energy measuring points.

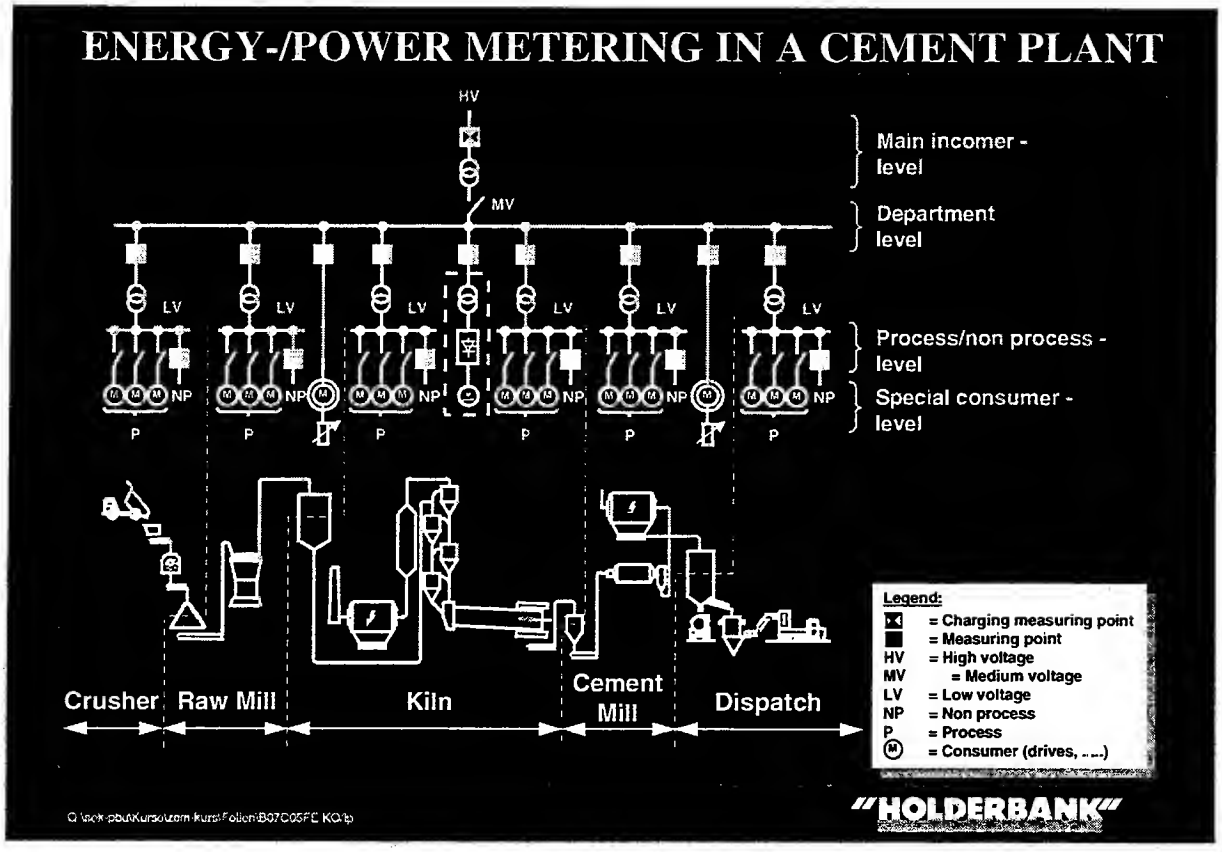
The measuring points are arranged in different levels:

- Main entrance-level; measuring point for charging by power company

At this point, the local power company measures the total consumption of the plant; it should also be a comparable-measuring of the plant; but attention, the result has not to be the same (two measuring circuits).

- Department-level;
- Every department should be measured separately to have an overview of the electric consumption in the different departments. Likewise for large consumers which are directly connected to the medium voltage distribution.
- Process/Non-process-level;
  It is necessary to measure the non-process part to distinguish between process- and non-process-consumers. Since less measuring points are used for non-process measurement, costs can be reduced by measuring the
- Total - non-process = process. (see drawing E310015)
- Special consumer-level;
  This level contains consumers of special interest from an energy point of view.

Energy-/power metering in a cement plant



If all these measuring points are counted together, the total will be around 50. But in practice it is very difficult to find such an electric distribution. So in reality the amount of measuring points will increase to 500 points which will be rather costly. The conclusion is, that measuring starts with **proper electrical departments thus a proper distribution.**

How to measure energy?

Today there are two measurement principles for electric energy, a direct measuring method and an indirect one. - An important criterion is the measuring principle. The method applied depends on the kind of load, the connected voltage (voltage - and/or current transformer necessary) and the accuracy required. The figure F44942 shows different connection diagrams for the measuring method with kWh-meter. The same principle is utilised for a power transducer.

kWh-meter (direct measuring method)

The kWh-meter forms with the current-path and the voltage-path a mechanical torque, which is proportional to the electric power. This torque sets the metering disc in a corresponding number of turns per unit of time. The multiplication with the time to receive energy, follows with the addition of the number of rotations. The kWh-meter shall be equipped with an on/off output module. This module includes a pulse contact system and a pulse amplifier, suitable for further handling in a control system. For example; In the control system, the energy will be calculated with the time between two pulses. The practice shows that this "digital" method is not such exact than the method described below. - The above described measuring method is the principle of "Ferraris"-counter or eddy current motor.

Today more and more static kWh-meters are utilised. The same measuring method is done in an electronic way and not any more in an electro mechanical way, but there is still a pulse contact output.

Power transducer (indirect measuring method)

The power transducer measures current and voltage separately and calculates internally the electric power. The output signal is an analogue signal (4-20 mA) for electric power. *This analogue signal is an input in the control system* where the signal is integrated with the time which results in a more accurate measurement as opposed to a pulse-signal from a kWh-meter.

Process measurement display

The display of the energy measurement is just as important as the measurement itself. The measurement has to be displayed in form of power or energy as well as specific energy measurement (e.g. for the raw mill → kWh per ton of raw meal. The display is further described in chapter "MMI, Visualisation".

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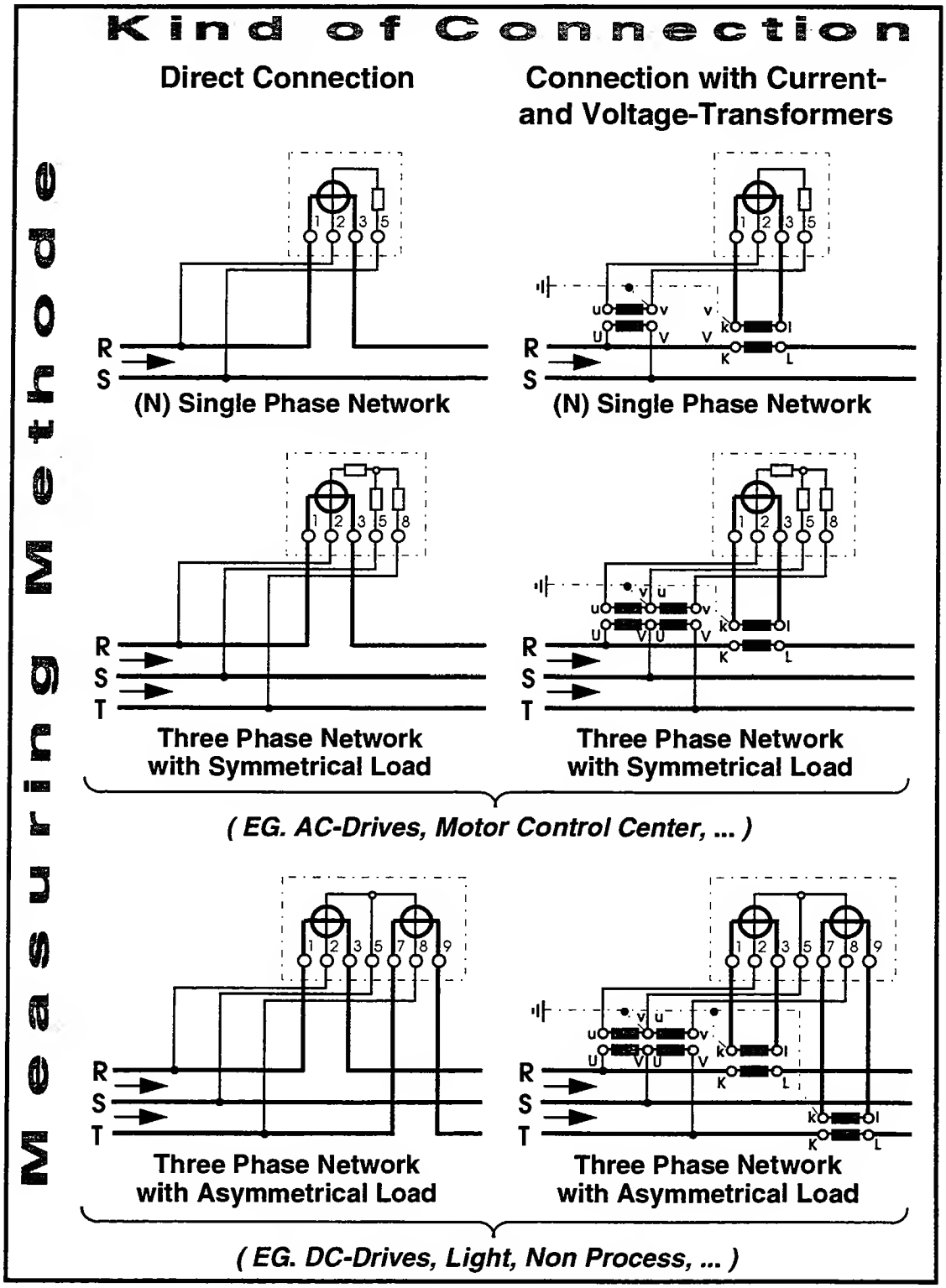
Power Measurement/Connection Diagrams for kWh-Meters



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7.8 Field devices

Just as important as analogue instruments are the on/off sensors, control and field devices. Most measuring principles as described in the previous sections are available as on/off devices. Few on/off sensors are available as smart sensors and even less can be connected to a bus (Profibus). The problems encountered with field devices are similar as with analogue instruments and the same provision has to be made as with the analogue instruments. Sensors like limit switches, pressure switches, speed switches etc. are available in different price classes. Do not feel tempted to buy cheap sensors as the money saved will be spent tenfold on maintenance and on down time. Only the best is good enough. And only the best installation will give acceptable performance. And where possible replace on/off sensors with analogue sensors.

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Chapter 6

Motor Control

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Motor Control

Roland Luder

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1. INTRODUCTION

As already mentioned in the chapter AUTOMATION SYSTEMS, Motor Control is a very important part of cement plant control. It is the part which switches drives and valves etc. ON and OFF.

These switching operations have to be performed under normal as well as under special operating conditions of the process. Motor control further includes monitoring and alarming of these conditions. Physically, a motor control system consists of

- an operating panel as interface to the operator (with push buttons, lamps, mouse, displays)
- a logic controller which performs the logical interlocking of all information (with relays, electronic or programmable controllers)
- a plant/process interface with sensors and command elements (switches, contactors, valves)
- cabling, which interconnects the different components of the system.

A motor control system for a modern cement production line has to control approx. 1000 motors and a few hundred valves, heaters, acoustic and optical devices. The correct and efficient command of all these items requires a systematic planning of the system and a careful selection of the components.

The tendency in systems design is to integrate motor control and analogue control functions into the same physical unit. This set-up has the advantage that only one type of hardware is required and that interconnections between the binary and the analogue systems can be programmed and do not have to be wired.

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2. PROGRAMMABLE CONTROLLERS (PLC)

2.1 Introduction, History

The logic controller is the brain of the motor control system, i.e. the decision-making part. It takes care of the safe, convenient sequential starting, running and stopping of single motors or of whole groups of machines.

There are basically three different techniques to build-up an electrical logic controller:

2.1.1 Relay Technology

A relay uses the electro-mechanical equipment which uses the electromagnetic force of an electrical coil to open or close electrical contacts. The internal wiring of these coils and contacts determines the specific function of the set-up.

Relays control has been the only technology until approx. 1960.

Today relays are still used for interfacing (power amplifying) purposes and for very small control systems.

2.1.2 Electronic Card System

This technology was developed in Europe and was applied approx. until 1975. It consists of electronic components, pre-assembled on printed circuit boards and performing specific logic functions. The function of the total system is determined by the wiring between the different logic cards.

Today these systems are hardly anymore installed.

2.1.3 Programmable Controllers

This technology was introduced in the market approx. 1970, and is the today's dominating technology. The tendency is to replace relays by programmable controllers even in very small applications. The programmable controller uses basically the same idea as a computer does: it uses a memory to store information and it uses this information to execute step by step a procedure which is determined by this information (program; software).

The advantages of the programmable controllers as opposed to other technologies are:

- No mechanical wear
- no rewiring when modifications are required
- easier planning (hardware / software can be planned in parallel)
- higher level of automation is possible
- integrated solutions with analogue control and data acquisition are possible.

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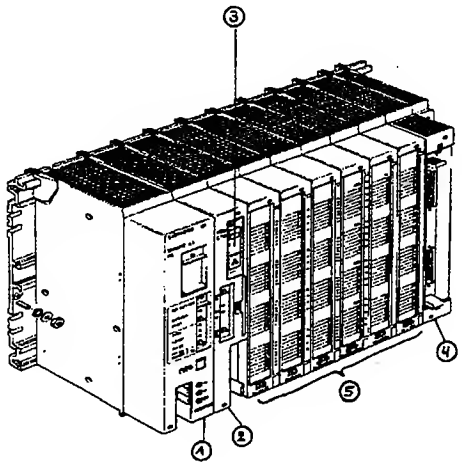
2.2 Hardware of Programmable Controllers

A programmable controller is basically a 'BLACK BOX' with INPUTS and OUTPUTS and a connection to a PROGRAMMING UNIT.

With the programming unit, the user determines how the different inputs and outputs must be logically correlated and sequenced to perform the desired control task. The programming nowadays uses a PC to display, to enter or to modify the program as well as an additional storage devices to safeguard it.

The 'BLACK BOX' is composed on one or several chassis or racks which basically contain the following elements (see Fig. 2.2.1)

Figure 2.2.1 Programmable Controller Hardware Configuration



1) **Power supply**
   To provide the internal stabilised control voltages of the programmable controller
2) **Processor**
   To perform the actual logic / timing and internal control functions
3) **Memory**
   To store the program and data
4) **Communication port**
   To communicate with external devices (as programming panel, computer, other programmable controllers) or with remote input/output devices.
5) **Input/Output (I/O) Devices**
   To transform the signal coming in (INPUT) from outside (e.g. from a level switch) or going out (OUTPUT) to the outside (e.g. to lamps, to the MCC). Today most programmable controllers handle as well analogue I/O signals.

Physically all these elements are generally grouped on one or several cards of the draw-out type. The cards are internally interconnected via a system bus which performs a fast exchange of the necessary information between the different cards.

This arrangement provides great flexibility in hardware planning: if additional memory space is required, for examples, it is generally sufficient to plug in an additional memory card. The draw-out technique, of course, also simplifies troubleshooting of the hardware (defective card out - new card in - restart). It must be noted, however, that once a system is correctly set-up, there are generally no more hardware failures.

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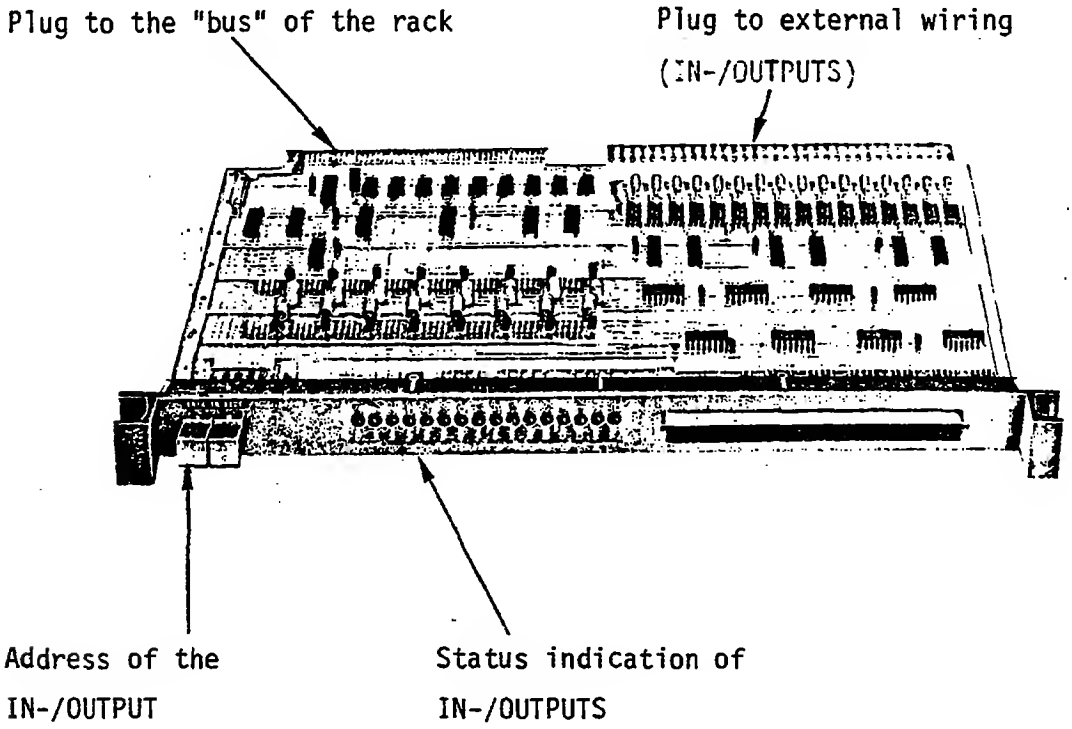
Figure 2.2.2 shows an example of an input card of the draw-out type which contains 16 inputs. Output cards may be arranged in a similar way.

Figure 2.2.2: INPUT (OUTPUT) Card of a Programmable Controller



The size of a programmable controller can be expressed by different figures. The most important figure is the number of I/O which a programmable controller can handle. The smallest units start at approx. 15 I/O, the biggest go up to 8000 I/O.

The maximum memory size is another key figure which, however, generally goes in parallel with the number of I/Os. Normal sizes range from approx. 0.5 K up to 1000 K (1 K = approx. 1000 program steps).

The cycle time (see 'software') generally goes in parallel with the size of the memory used and is generally expressed in ms/1 K. It ranges from approx. 0.5 to 50 ms/1 K.

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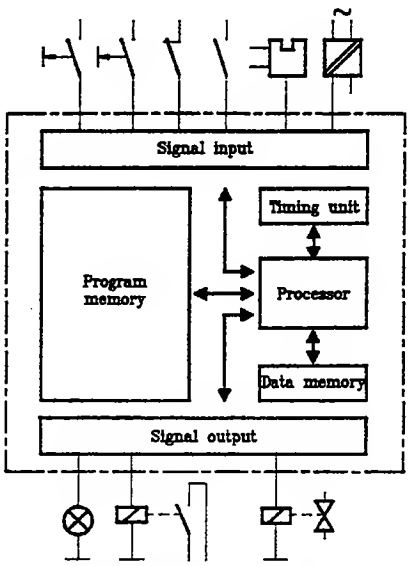
2.3 Structure of a PLC

The diagram below shows the structure of a PLC. The main functional elements of a programmable logic controller are the control unit with one, or sometimes several micro-processors and the corresponding memories for data (timers, counters, markers, etc.) and programs (programmable memories).

Block diagram of programmable controller



The program memory, processor, counter, data memory and the input/output units are interconnected. Here connection via bus has become standard practice. By means of this bus the data are exchanged between data memory, processor and program memory.

2.3.1 Differences between PLC and a Computer

What are the main differences? In a PLC so-called bit processing is used. This is special processing method which processes only one bit. In contrast, the computer always uses word processors, i.e. single bits can only be addressed by programming.

A PLC functions in much the same way as a computer, but with the following main differences.

- User's programs are executed cyclically
- A PLC needs a very simple operating system
- The information is processed a bit at a time (facilities for word processing are available)
- A PLC is a real-time system, i.e. the results of operations are obtained within a short, clearly defined time
- The set of command is especially intended for control requirements and is therefore limited in its scope

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- The hardware is designed for rough industrial conditions (temperatures between -10 and +60°C)
- Programming is simple, can be understood by electricians and is easy to learn
- Addressing of input and output cards is transparent
- To program a PLC a special programming unit has to be connected to it.

2.4 Software of Programmable Controllers

In the chapter 'Hardware of Programmable Controllers' we have seen that a programmable controller can be represented as a 'BLACK BOX' which contains inputs and outputs.

The different inputs and outputs have now to be logically and sequentially interconnected to perform the desired control task. To represent this "logical and sequential interconnection", special languages have been elaborated. Unfortunately, these languages or graphic presentations are not standardised. Main differences can be found between European and American presentations but even within one language, practically every suppliers uses his own 'slang'.

For a long time there was a certain market tendency toward the ladder diagram, influenced by the American market where generally only this language is used.

The ladder diagram is based on the representation which was used for relay systems. It can, therefore, easily be learned by people who worked with those systems but it does not well represent the new thinking in inputs/outputs.

Nowadays a standard is on the market named IEC1131.

Any logic can actually be represented with only 4 different instructions:

Logic AND, logic OR, logic NOT and TIME instruction. In order to make programming easier, all programmable controllers use additional logic instructions which are composed of specific often-used combinations of above four elements.

When a programmable controller is equipped to accept analogue inputs, additional instructions for arithmetic operations and file handling are available.

For bigger applications (as e.g. in the cement industry) a structured programming making use of 'macros' or subroutines should be applied. The same applies when analogue control capabilities are included in the programmable controller.

The actual program is now composed of a series of program steps, every step uses a combination of instructions or 'macros' to define how the different inputs have to be linked with the outputs.

It is important to mention that the program is executed step by step. This means that one instruction after the other is read, interpreted and executed.

At the end, the program automatically restarts at the beginning. The total time which a program needs to come once from the beginning to the end is called the system cycle time. This time is generally very short (approx. 1...300 ms). For an external observer of the system it, therefore, behaves as if everything (all commands) would be immediately executed - all at the same time. In reality, as explained, only one instruction is executed at one time.

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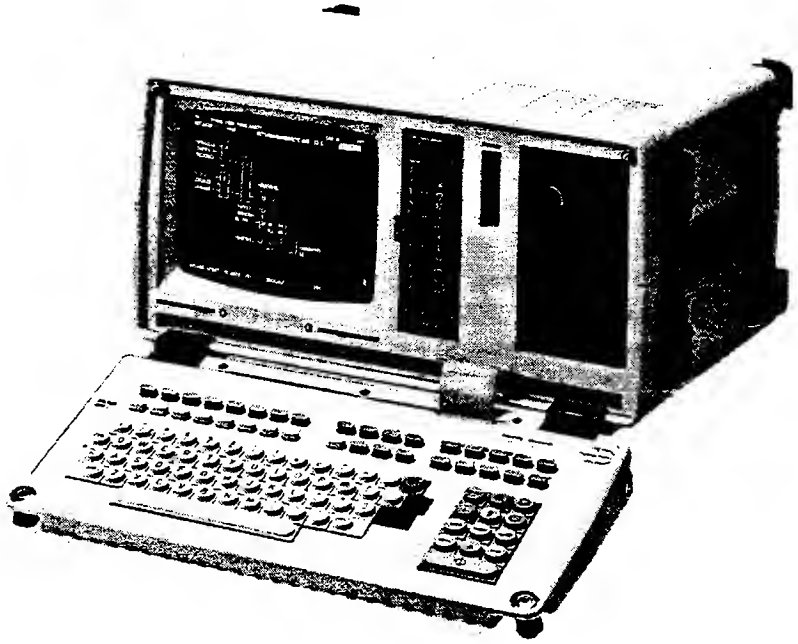
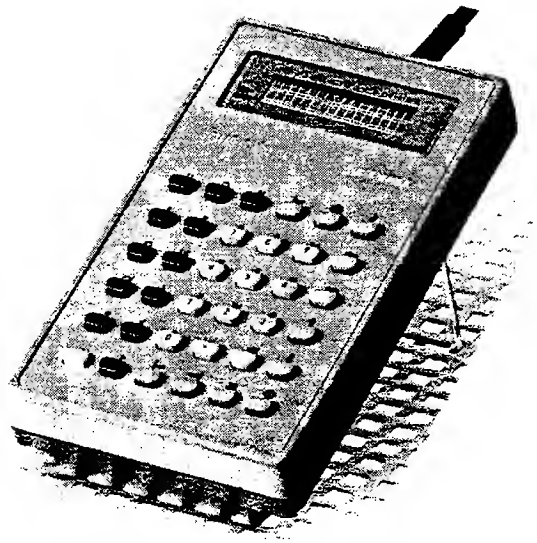
2.5 Programming a PLC

In order to program a PLC a special programming unit is required. The pictures below show different versions of programming units.





Programming units are very efficient, they are PC's of special industrial design, equipped with large-area LCD or with a monitor. They can translate the functions entered by the user in a higher-level programming language specifically intended for control tasks, e.g. as a list of instructions (Fig. a), as functional diagram or contact diagram, direct into the machine code of the control units. They can also translate from the machine code back into the higher-level representation.

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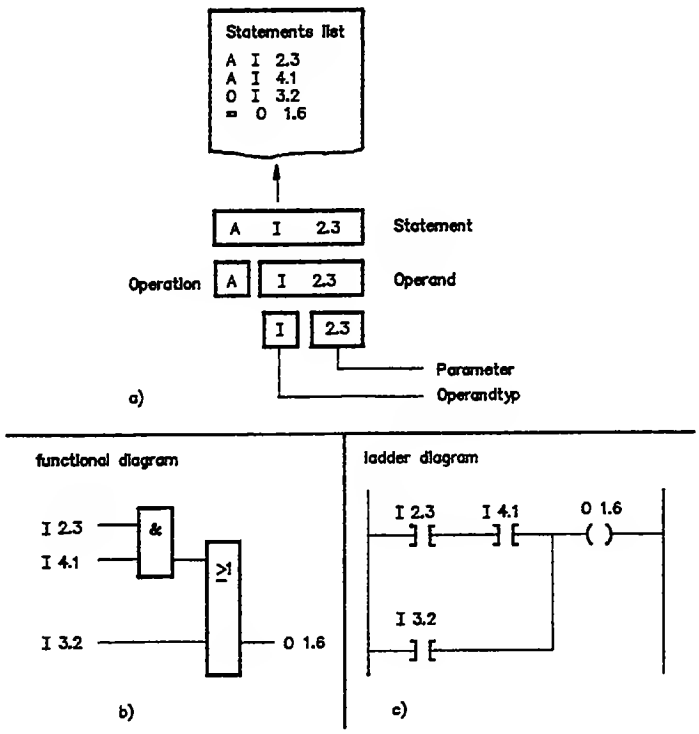
Figure (a) Programming languages (Siemens)



A control task can be expressed in different 'languages'; the list of instructions (a) can also be translated by simple programming units into a machine language understood by the control system. The functional diagram (b) which is easier to understand by engineers and the ladder diagram (c) which is very popular in USA, impose more exacting demands on the programming unit.

At all events such units can be employed on-line, i.e. connected direct with the controller. The programs to be entered in this case are usually entered direct in the programming language of the controller, or for display purposes are read out of a memory. In on-line operation it is also possible to perform test functions, fault location and program correction.

A further important function of the programming unit is the program documentation. Special documentation software converts the programs into a form readily understood by the user (auxiliary text, I/O description, comments, etc.). A printer connected to the system is able to print out the documented programs.

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2.6 Criteria governing the Choice of PLC

Basically, any PLC can be used for control tasks in the cement industry, provided it satisfies to following requirements.

2.6.1 General

- The system is well represented and generally known in the country
- Spare parts are guaranteed obtainable at least for 10 years
- The system must be capable for expansions in order to integrate future adaptations
- The dimensions of equipment permit the replacement of existing facilities

2.6.2 Central Unit (CPU)

- When the system is extended to full capacity, the cycle time should not exceed 150 ms
- Adapted memory capacity, so that there is no shortage of storage capacity when the system is fully expanded.
- On-line programming of modifications while the process is in progress
- Reasonable set of instructions containing the following:
  arithmetic with variable decimal point, PID algorithms and functions specified by the user, modules (e.g. motor module).
- Floating point arithmetics

2.6.3 Communication

- Between PLC's with bus
- Standardised interface with simple protocol with a main-frame computer (e.g. RS 232, RS 422, RS 485)
- Standardised interface with subcontrol systems (field bus, profibus, RS 232)

2.6.4 Inputs/Outputs

- Decentralised peripherals connected with the CPU by a serial line (possibly an optical link)
- 24 V or 48 V DC single-ended inputs/outputs with common ground
- 4-20 mA analogue single-ended inputs/outputs with common ground
- Capable of extension up to 2000 inputs and outputs

2.6.5 Programming and Documentation

- Off-line programming and documentation with standard PC's
- Remote line connection of programming units
- Graphic representation, preferably by functional diagram
- Symbolic programming of addresses with at least 15 freely chosen characters
- Commands and operating instructions in the local language

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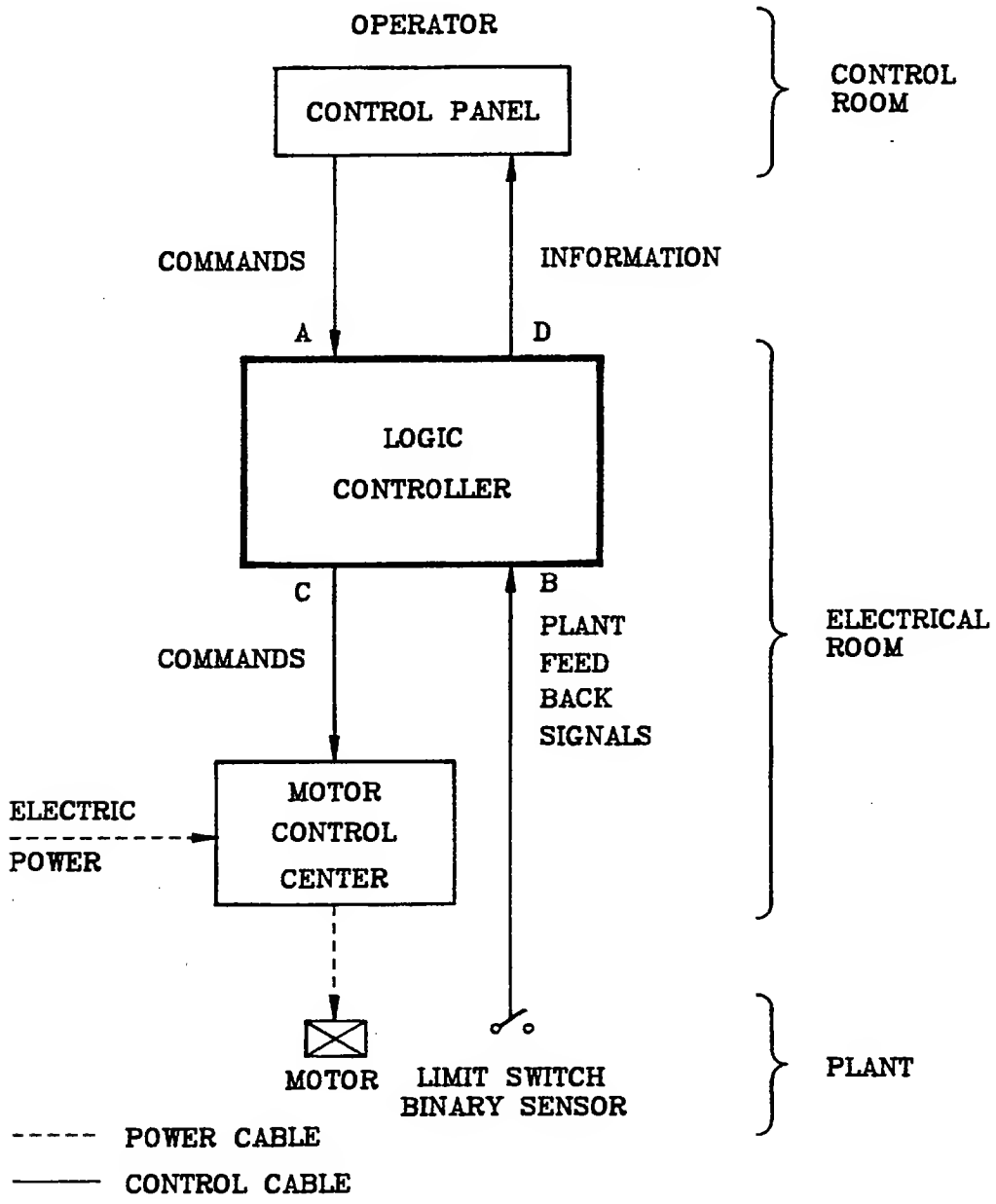
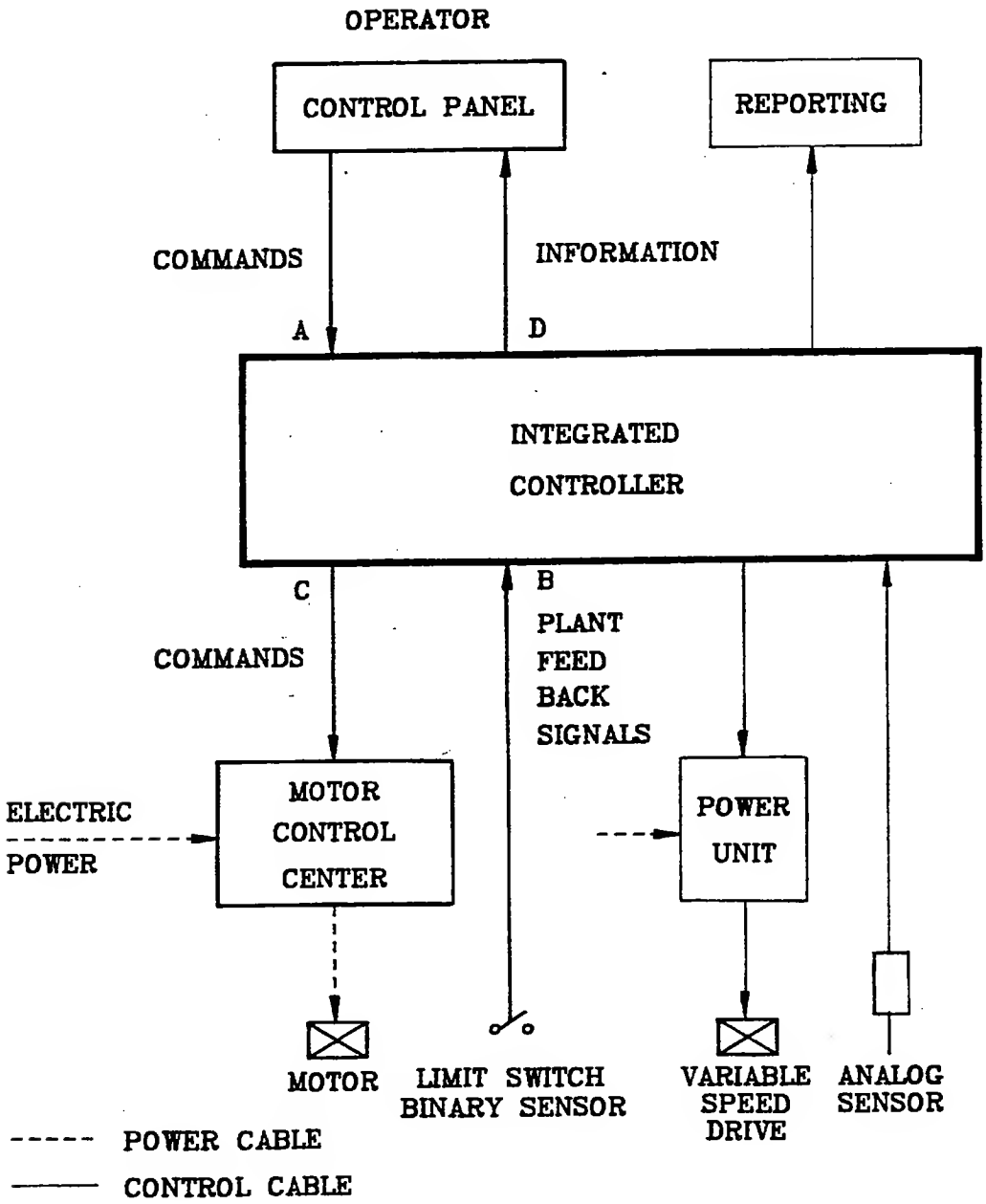
Motor control - Logic controller

Task: Sequence start and stop of motors, interlocking of motors, machine protection, alarming



Motor control - integrated

Task: Sequence start and stop of motors, interlocking of motors,machine protection, alarming, measuring, analog control, reporting



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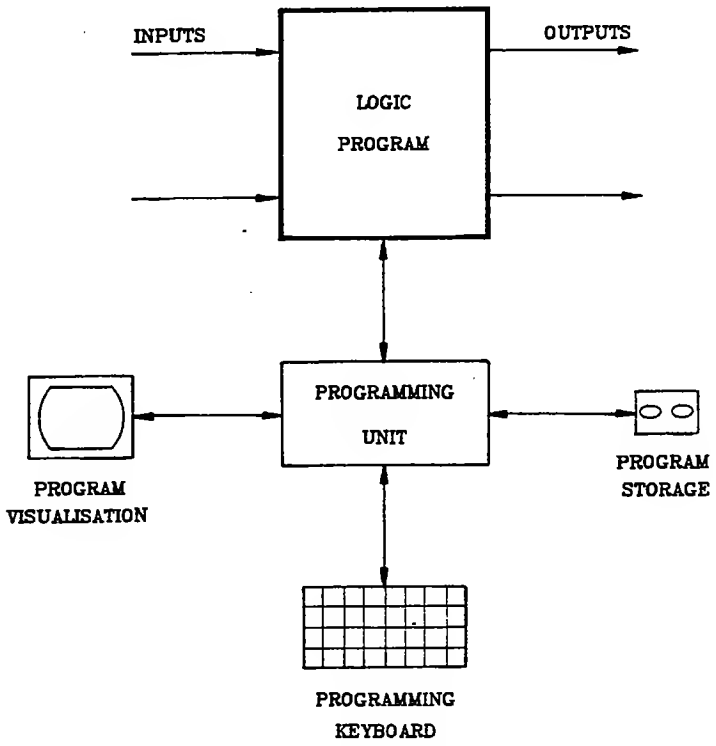
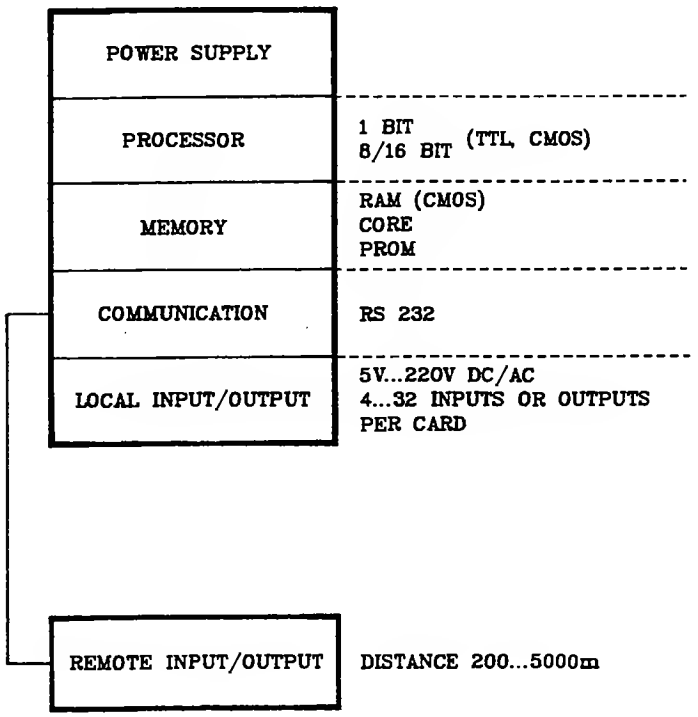
Motor control - Programmable controller: System configuration



Motor control - Programmable controller: Hardware configuration



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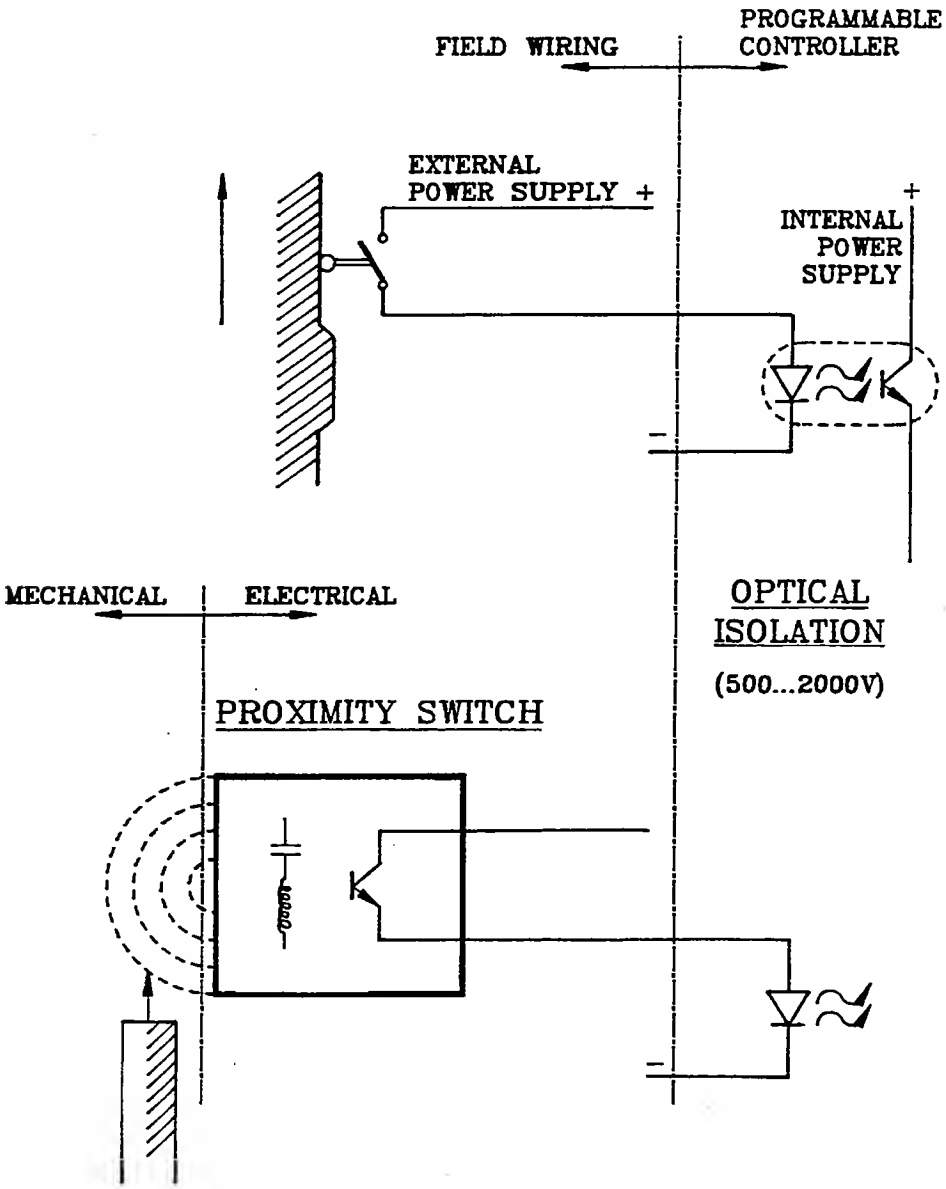
Motor Control - Programmable controller: Isolation of signals



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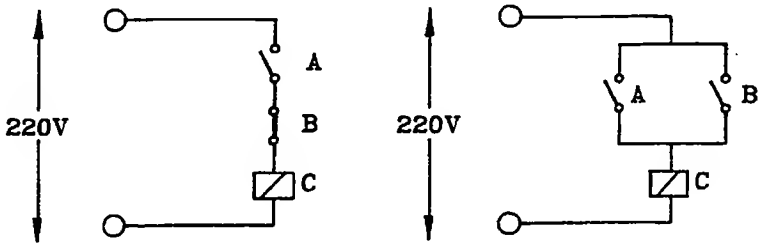
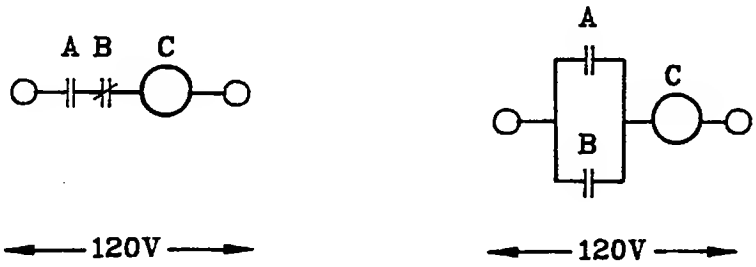
Motor Control - Programmable controller: Language, presentation

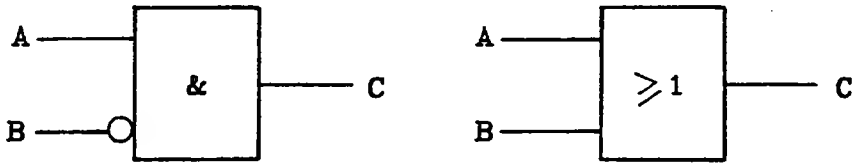
1. Relay European: AND NOT / OR



2. Relay American (Ladderdiagram): AND NOT / OR



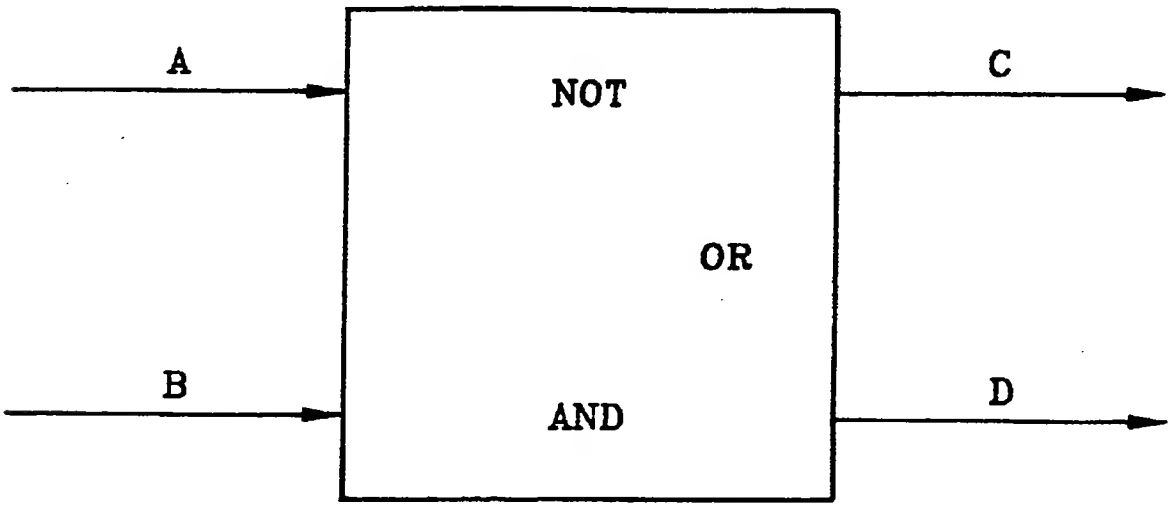
3. Logic card European: AND NOT / OR



4. Programmable logic controller: Boolean language, (or logic, or ladder)

A AND NOT B = C **A OR B = C**

Motor Control - Programmable controller: Language, Example



Example:

A: Operator command: Turn motor left

B: Plant feedback: Limit switch left reached

C: Control system command: Turn motor left

D: Operator information lamp: Device in position left

Program	**Short**
1. Input A	
2. AND NOT Input B	
3. Equals Output C	A AND NOT B = C
4. Input B	
5. Equals Output D	B = D

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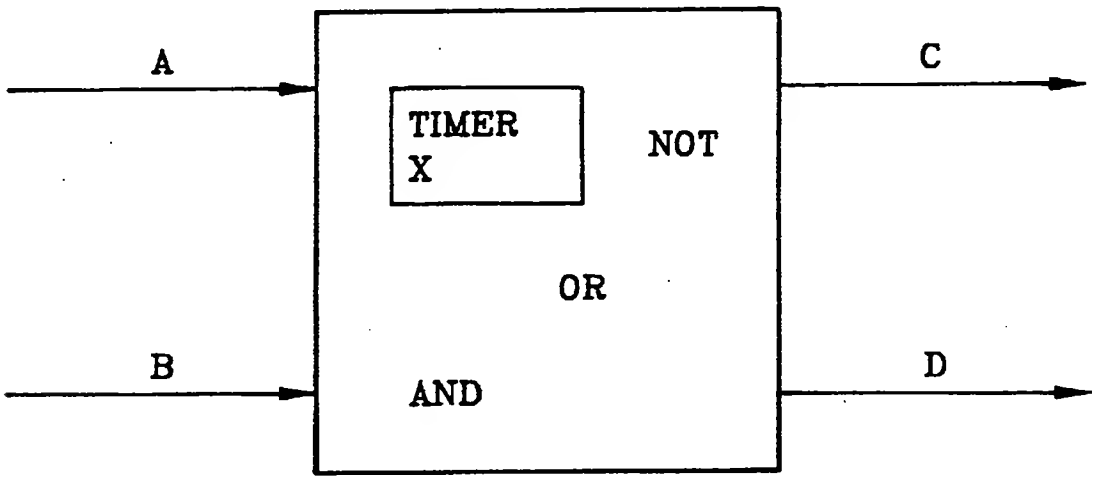
Motor Control - Programmable controller: Language, timer



Example Timer:

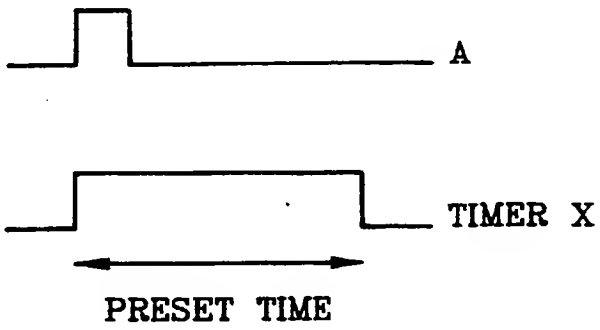
A: Operator command: Turn motor left

B: Plant feedback: Limit switch left reached

C: Control system command: Turn motor left

Program

1. Timer X = 30 seconds
2. Input A
    3. Equals timer X
4. Timer X
5. AND NOT input B
    6. Equals output C



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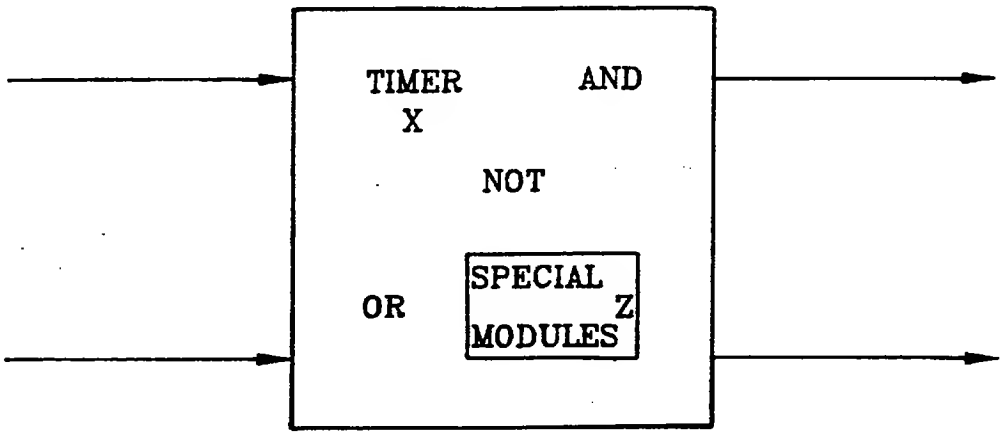
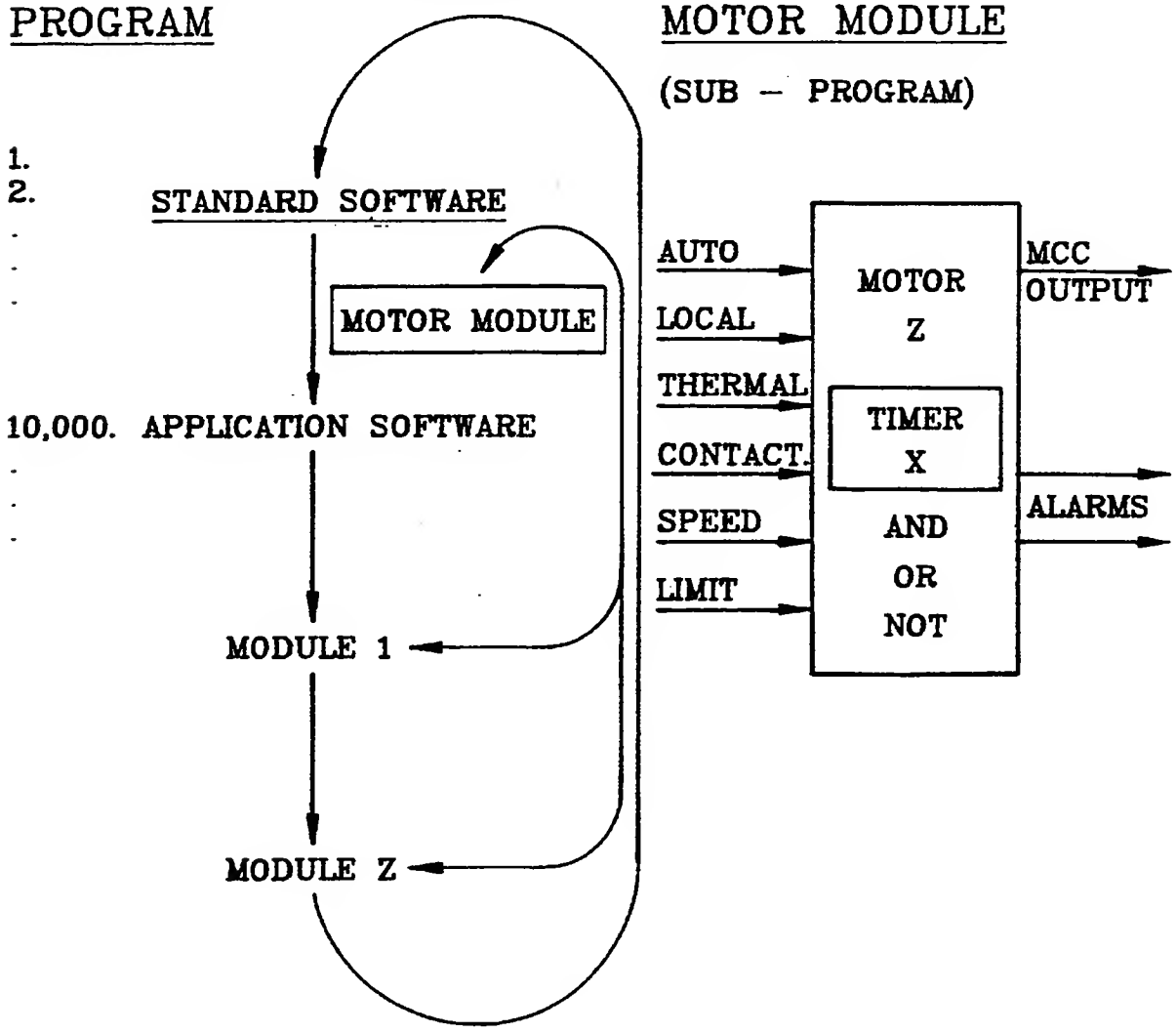
Motor control - Programmable controller: Language, structured programming



EXAMPLE MOTOR MODULE:



1 cycle = 10...300 msec
PROGRAM LENGTH = 1000...100,000 INSTRUCTIONS

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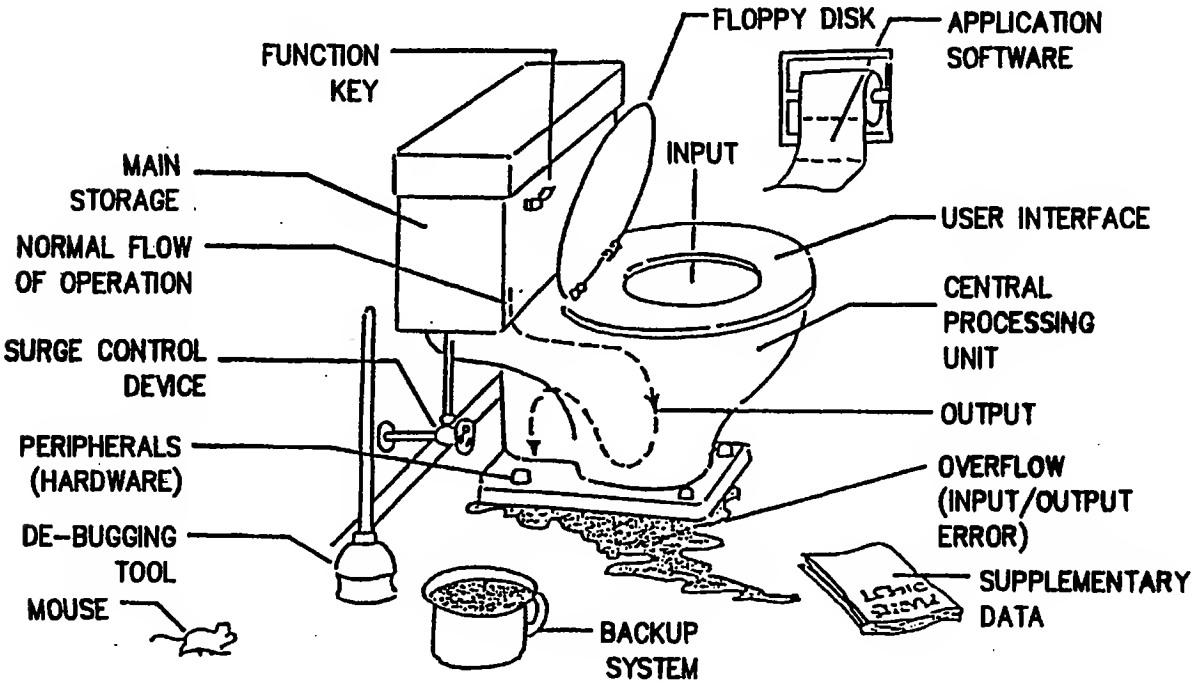
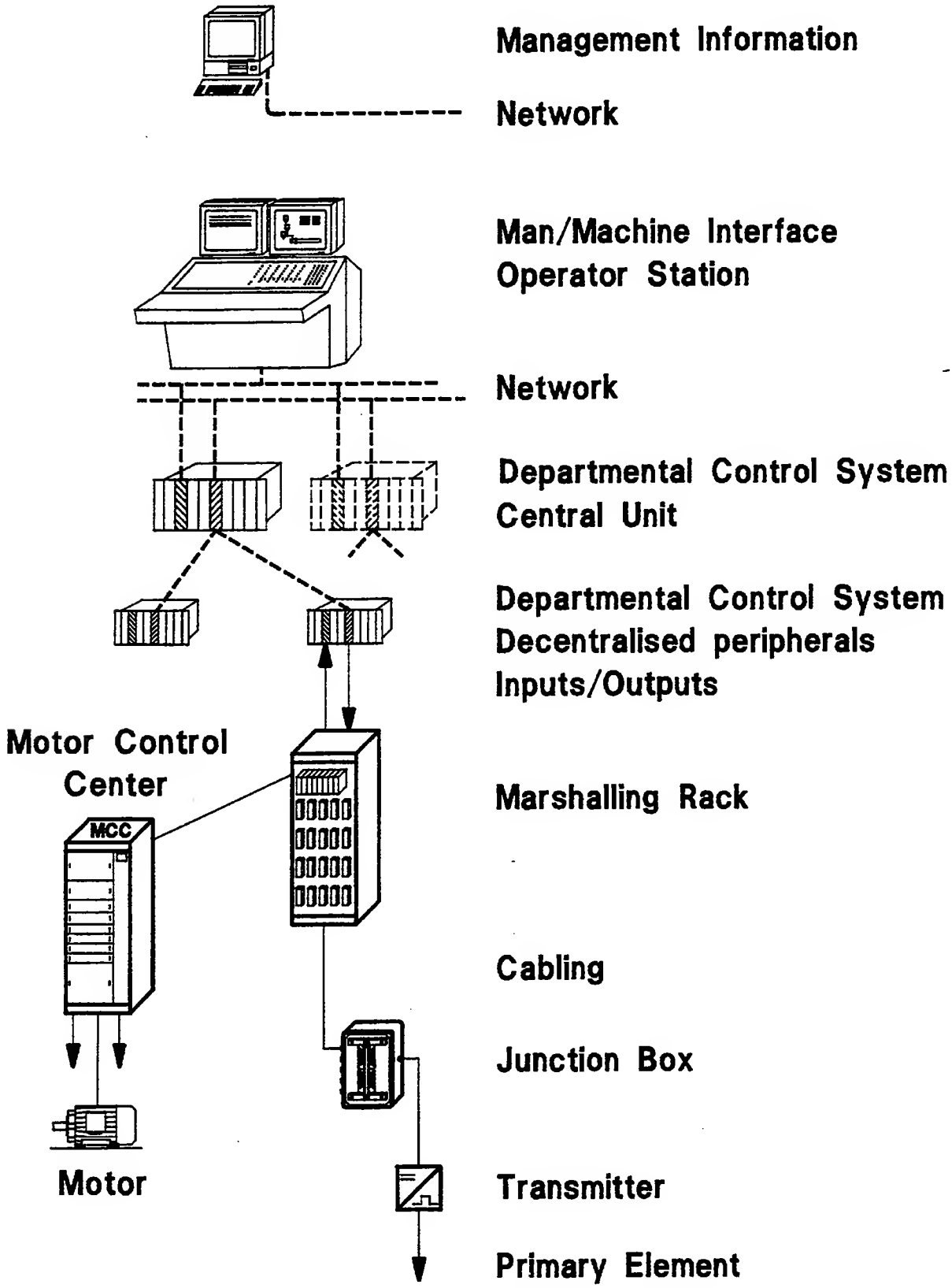
Understanding Computer Technology



Automation systems - Typical configuration



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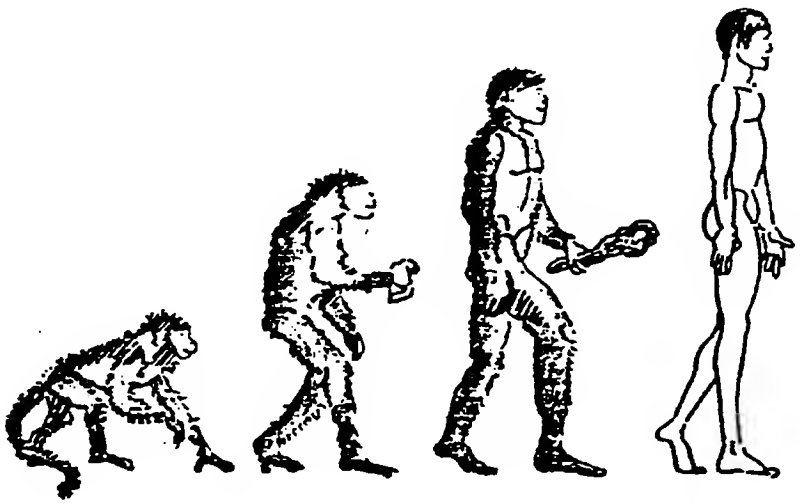
History of PLC's

1968:	Concept of a Programmable Controller
1969:	Hardware based CPU
	Logic statements, 1 Kbytes Memory, 128 i/o's
1973:	Source Code Editing
1974:	PLC with Multi-Processors
	Logic, Counter, Timer, Move Words, Arithmetic's
	12 Kbytes Memory, 1024 i/o's
1976:	Decentralised I/O-Systems
1977:	Micro Processor based PLC with Logic Co-Processors
1978:	Universal I/O-Structure
1979:	Bit Slice Processor Architecture
1980:	High efficient decentralised I/O-Structure with
	intelligent I/O-Modules and Block Transfer Instructions
1981:	Data Highway with Medium Response Time(Token Passing)
1982/83:	Micro-programmed, with multi-processor PLC's of 4th Generation
	In BASIC programmable Co-Processors
	Integrated Hard disk
1985:	Intelligent Programming Units
	Optical Links
	Use of PC's for programming and debugging
1986	Graphic/Function Programming
	More Mass-Storage
1989:	?



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1968 **M**odular **D**igital **C**ontroller

MODICON

- General Motors:
  - simple to program, easy to change
  - maintenance low → modular system
  - safe in industrial environment
  - size smaller than relay panels
  - data communication to other systems
  - cheaper than relais

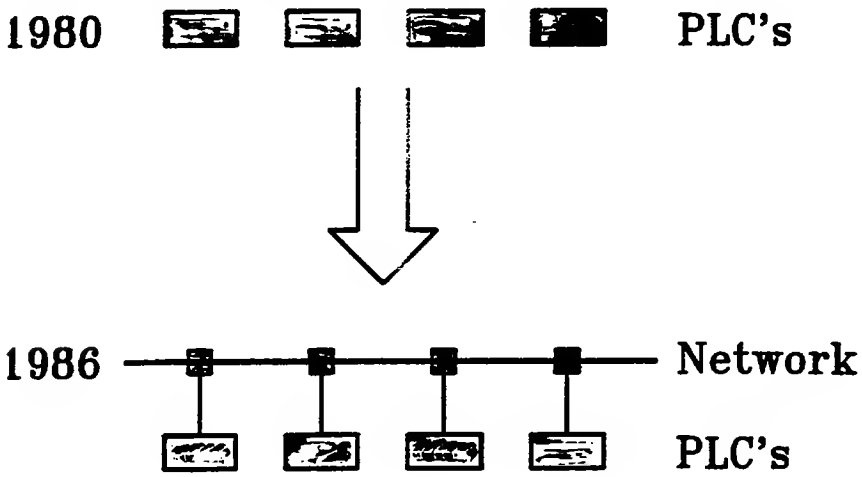
⇓

Programmable Logic Controller

Evolution of Control Systems

40 years of Relay Systems
20 years of Solid State
10 years of PLC's

from island to system



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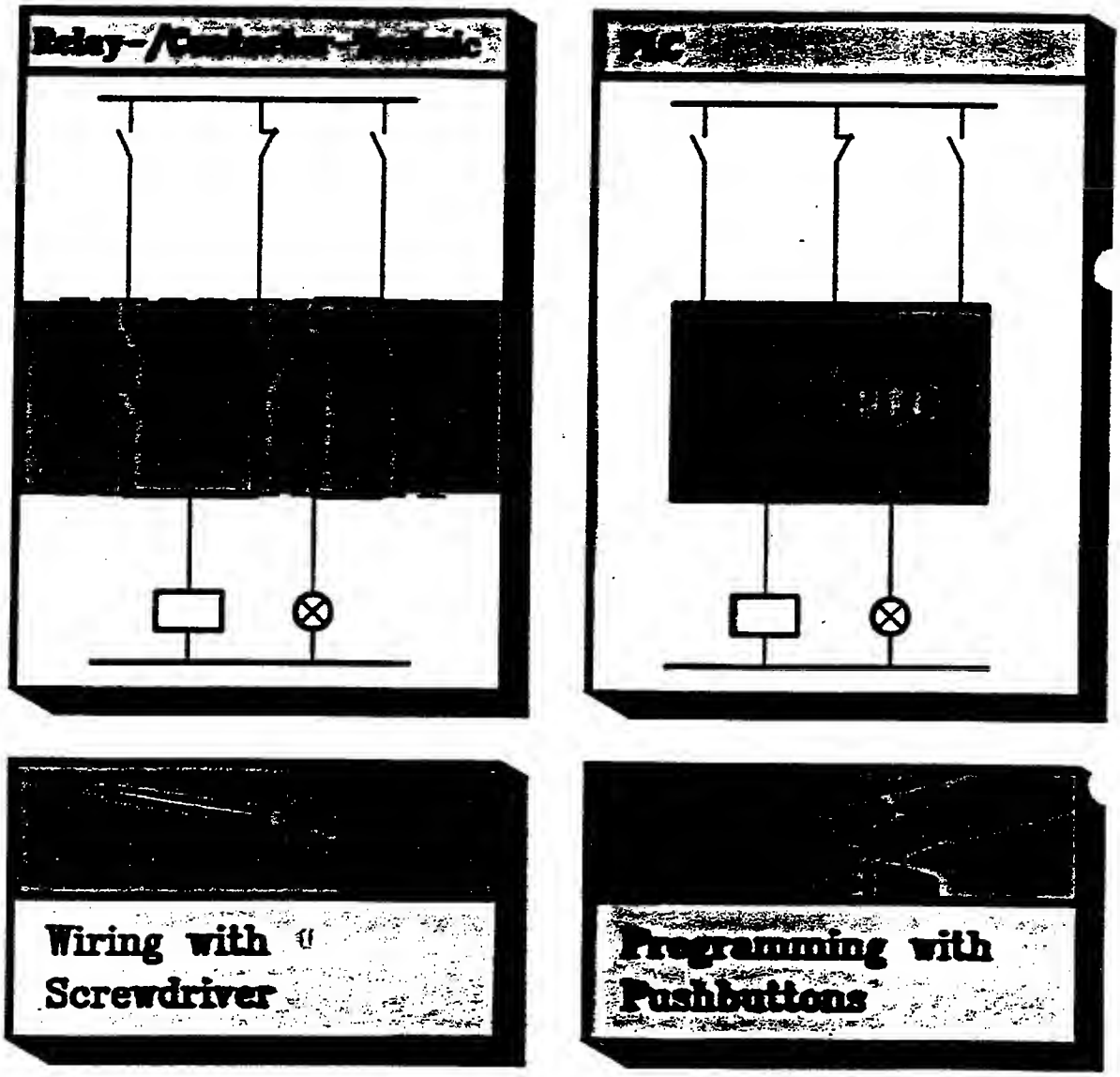
Differences between Relay and PLC Programming



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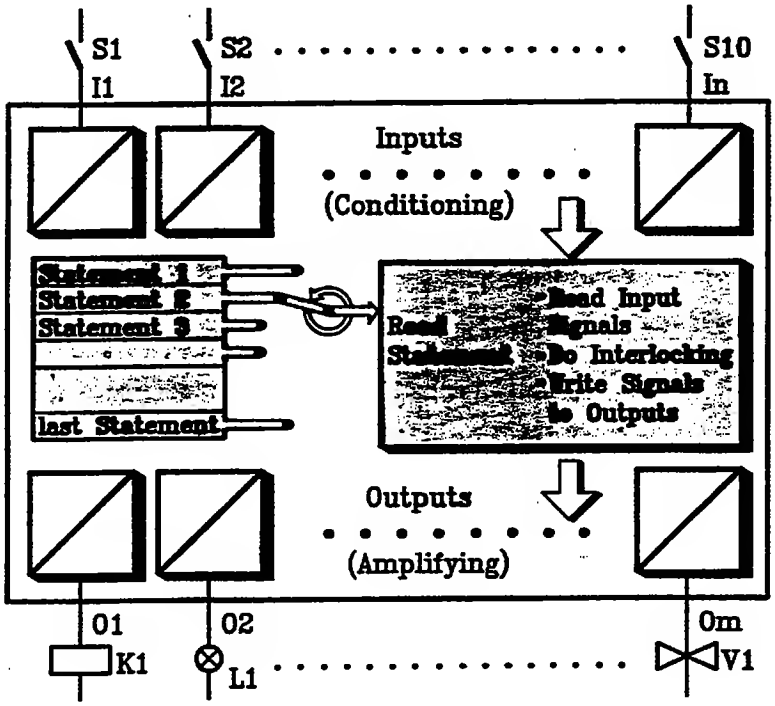
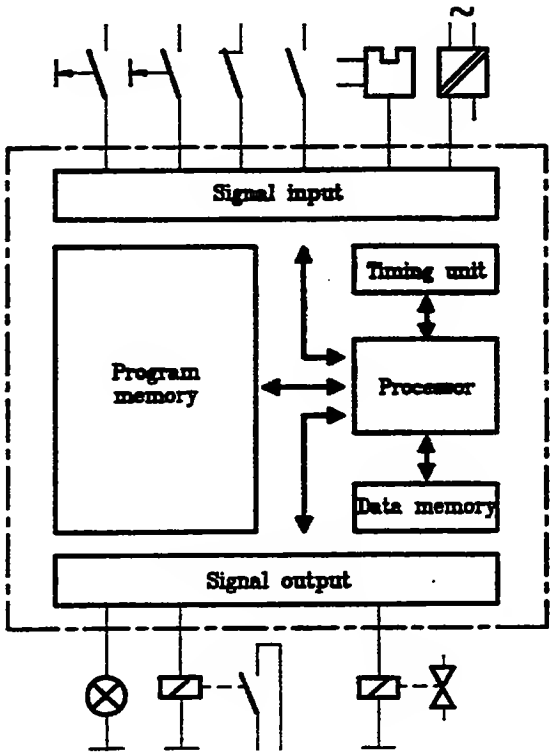
Working Principle of a PLC



Block diagram of programmable controller



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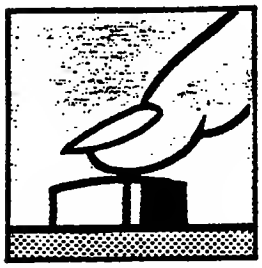
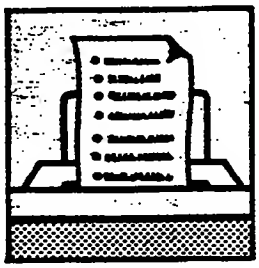
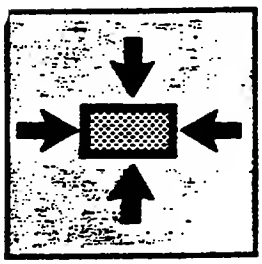
Advantages of PLC control against relay



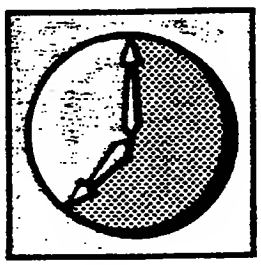
Change of control function by touching buttons



Automated and sophisticated documentation

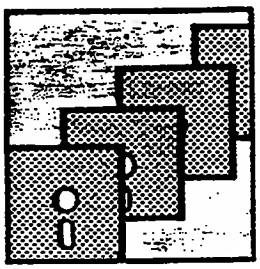


Small in size



Time saving while project execution

No wearing parts therefore maintenance free



Copy of programs by touching a button

Easy and fast for fault diagnostic

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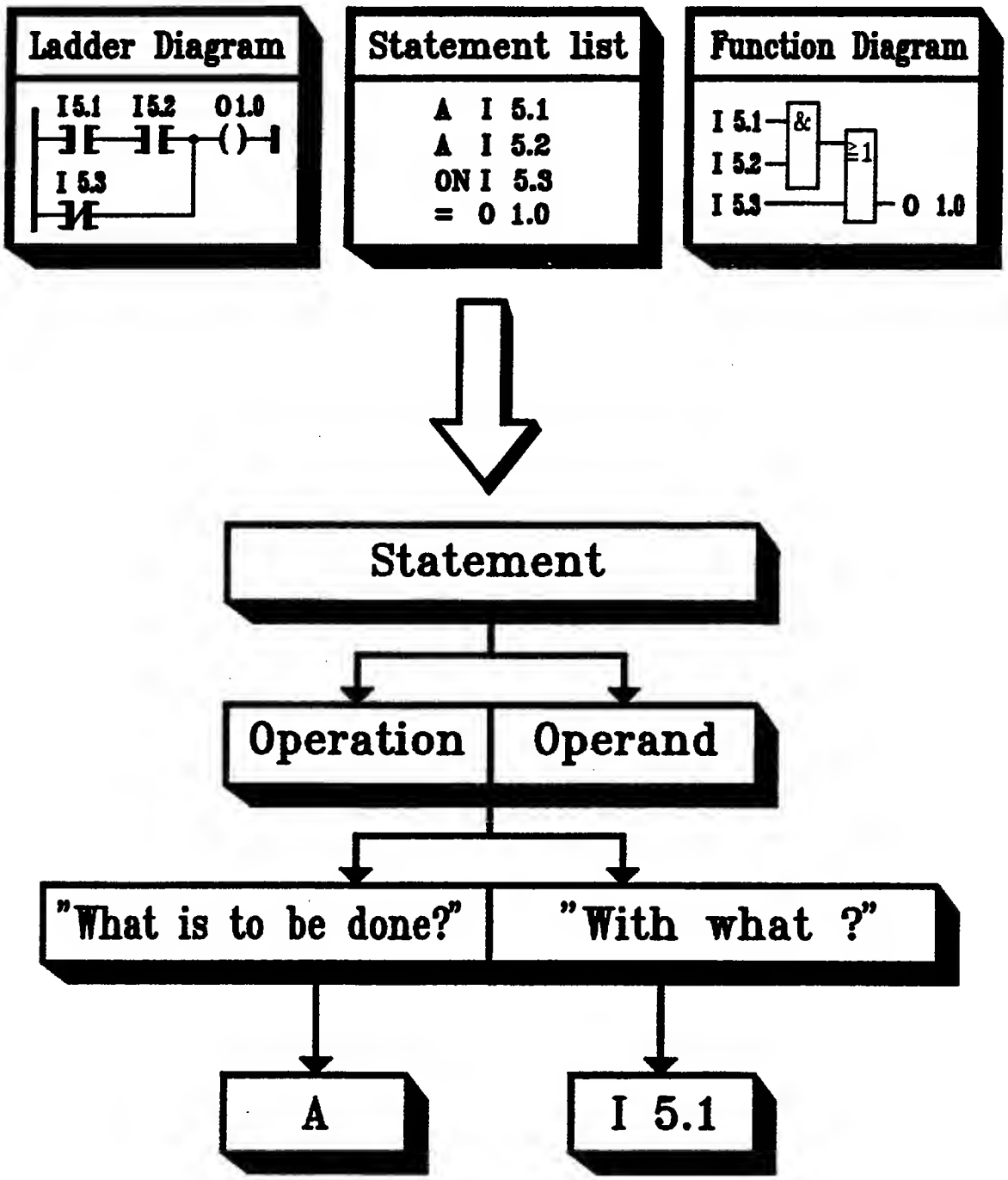
PLC-Progamming Language



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Chapter 7

Technical Information Systems (TIS)

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TECHNICAL INFORMATION SYSTEM (TIS)

Urs A. Herzog / Felix Fehr
HES 98/6348/E

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1. INTRODUCTION

Since a few years the cement industry is under continuous pressure to optimise their production in general. One reason is to lower production cost, another is to minimise the use of energy and fulfil the emission limits of new environmental protection legislation. Also cement market asks for new products and product properties, therefore production modification and expansion must be realised in short time.

To be able to act and react to this continuous challenge, management and personnel responsable of plant operation must have access to the relevant information about their production, and production equipment.

Beside the three classical production resources: manpower, capital, real estate, predictions are that information will soon be the forth.

Therefore Information Management Concepts and tools which support the user to get and analyse this information, are getting more and more important:

This paper gives an overview of modern Process Information Management, of available Information Systems and some experience of applications of such Information Systems.

2. PROCESS INFORMATION MANAGEMENT

2.1 Introduction to the Improvement Process

Market competence and changing regulations forces company and plant management to optimise continuously the cement manufacturing process to increase production, improve the equipment performance, reduce the use of energy, reduce the amount of necessary man-hours, in summary lower the overall production cost of our product the cement.

In general, optimisation and improvement is based on the following facts: (according Jim Harrington).

- **If you want to improve it, you have to control it !**
- **If you want to control it, you have to understand it !**
- **If you want understand it, you have to measure it !**

The three activities: measurement, understanding (to analyse data and to come to a conclusion) and control form together the **Improvement Loop**. Improvement is only sustainable if this improvement loop is executed continuously.

Corporate programs as BCM or MAC fully relay on this improvement loop (e.g KPI measurement, reporting, action meetings, etc.).

Nowadays (since the invention of modern computers) some of Improvement Loop tasks can be executed automatically. The most time consuming and boring task of process data acquisition, raw data handling and report compiling can be done with a computer system. Plant personnel can be used for analysing information and for decision making. – This is a more adequate challenge for a human being than coping numbers.

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2.2 Importance of information for decision making

The basis for management to make the daily operational and the medium term tactical decision, is information. Information coming from the process and process related actions as quality control, dispatch, emission monitoring and others.

If the plant or company management want to act - proactively or reactively - they need information. The more they fine tune the operation (e.g: BCM) and the more they optimise the risks (e.g: MAC), the more comprehensive, reliable, accurate and direct must the information they base there decisions on be.

A modified quotation from a Japanese Business Expert (name unknown) says it quite direct:

Information is a key success factor for business
some Managers know this
others are learning
the rest will be Victims.

2.3 Process Information which is relevant for plant performance improvement

For the daily plant operation, as well as for medium and long term plant performances monitoring and improvement the different responsibilities in a plant need the following type of information on hourly, day, week, month and year basis.:

Company / Plant Management	• What is the plant performance this day / week / month / year? • Where is the biggest improvement potential? • How are we compared to the others (benchmarking)
Production	• What is the production rate, consumption, actual stock of materials?
Process	• Does the process / equipment runs optimal? - Why not?
Maintenance	• What are the equipment runtimes, machine conditions, failures rates? - Failure reasons?
Quality	• What is the quality level of our product? Why is it in / out of specification?
Energy	• What are the specific energy consumption? - Tendencies?
Environment	• What are the emissions? Are we endangered to exceed limits?

In more general terms, the plant personnel needs a tool which provides:

- **R**elevant Information
  - at the **R**ight Time
  - at the **R**ight Place
  - in the **R**ight Form
  - to the **R**ight People

  This simple but important statement (from: Rauli Hantikainen) describes the basic requirement of adequate Information Management.

  A computer systems which can fulfil the above mentioned requirements has to be based on a integrated Database incorporating data sets from all technical related plant disciplines:

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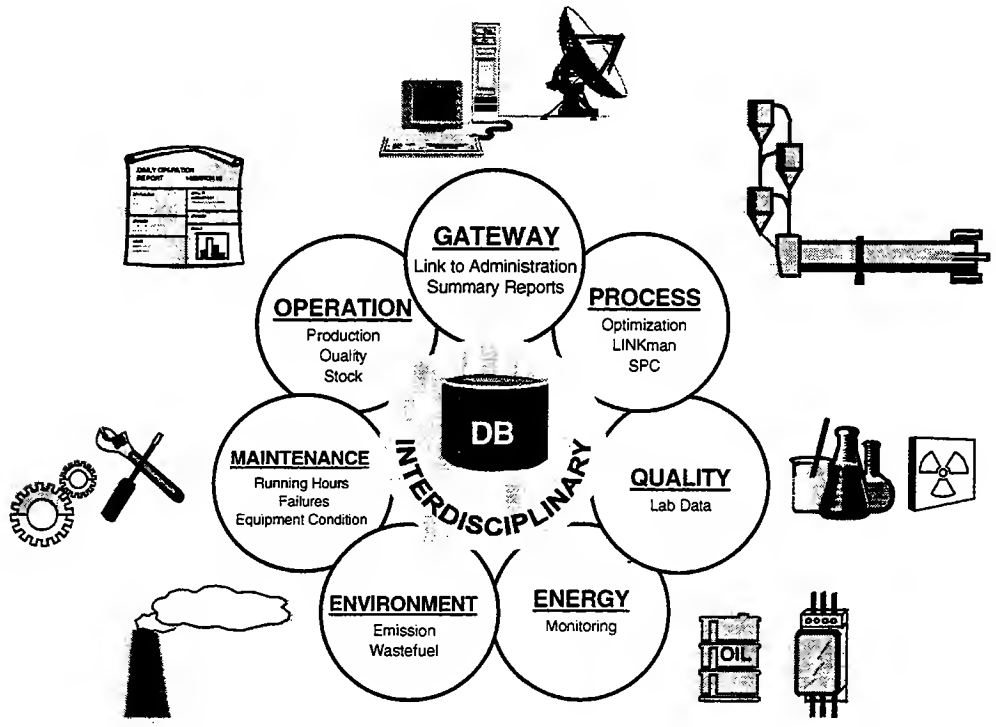
Figure 1:



Such system are called: "Technical Information System".

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3. TECHNICAL INFORMATION SYSTEMS (TIS)

In the past, most of the management's planning and optimisation was based on different data printouts, recorder charts and manually generated reports. But nowadays tools and system concept are available to get immediate and accurate information on there desk top in seconds.

3.1 Types of Automation- and Information System in a plant, Definitions

In this paper common industry term as CIM- and ERP- System are used. They are indispensable for describing and defining the architectures and system concepts that are involved in plant wide Information Management system solutions. Their definition is found just below:

- CIM = Computer-Integrated Manufacturing systems:

A series of computer-based systems, both business and technical, integrated into a single conceptual solution. A CIM system is comprised of components and elements of both ERP and TIS systems (see next), as well as the different automation systems like process control, high level control, quality control (lab), etc.

- ERP = Enterprise Resource Planning system:

A integrated, networked computer system providing (primarily) business functions and some manufacturing functions. An ERP system mostly provides also the information link between the company headquarters (HQ) and the plant. In most "Holderbank" Group Companies the product SAP R/3 is implemented (or planned) as a ERP system.

- TIS = Technical Information System:

A integrated, networked, real-time computer systems providing (primarily) manufacturing functions. Its real-time data base technology merge all plant automation data with data from the non-technical (or business) systems within a Plant. By capturing "live" information about the manufacturing process as, sensor measurement, set-points, run-times, throughput, yields, etc., a TIS can measure constraints and identify bottlenecks to better manage and control manufacturing processes (see also decryption in chapter 3.2).

(In some industries the term MES, Manufacturing Execution System, is uses also).

The sketch below shows the structure and interactions of the different mentioned systems.

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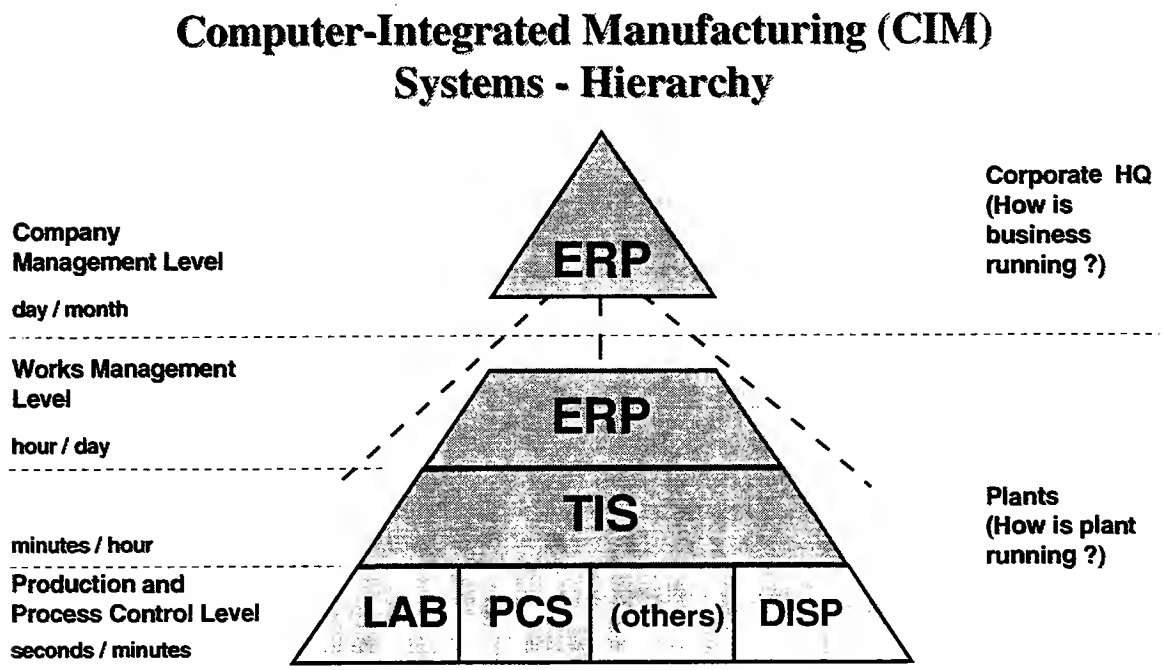
Figure 2:



Legend: CIM = Computer-Integrated Manufacturing (system); ERP = Enterprise Resource Planning; TIS = Technical Information System; LAB = Laboratory Information System; PCS = Process Control System; (others) as EMS = Emission Monitoring System and HLC = High Level Control Systems; DISP = Automatic Dispatch System

3.2 Principle of a Technical Information System (TIS)

3.2.1 General

In the CIM pyramid, a TIS is locate between the Company / Works Management Level (ERP) and Production / Process Control Level (Plant Automation). The interaction between those levels is as follows:

- The tactical and strategic decision are made in the higher levels:
  - The computer systems on these levels must provide to management business data and compiled and summarised plant information (process, production, equipment and quality) (time range: days, month, years).
- Operation of the plant is done from the lower level systems.
  - The computer systems on these levels must provide to plant responables immediate accurate real-time process and process related information. (time range: seconds to hours).
- A TIS System does the automatic data acquisition and data pre-processing. It provides tools for data evaluation, create reports, graphs and tables, for status and performance analysis. It also serves as a data interface between Operation and Managing Level of a plant and company.

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3.2.2 Tasks (requirements) of a TIS

Data Acquisition

Data from the process and process related tasks, e.g as material handling, product shipment and quality determination is the basis for compiling the above mentioned information. In a up-to-date cement plant most of this data are already automatically measured by a computer based Process Control System (PCS) and other Automation systems (High Level Control, Lab, Dispatch, Emission, etc.). In a standard set-up of a 3000 to 5000 tones per day plant, the PCS measures and handles roughly 600 analog sensor, 600 counters and integrators and up to 15'000 alarm and event messages. Data acquisition, data processing and visualisation is done in real time (milliseconds to seconds). Historical data is normally presented to the operator in graphic trend charts and data historian is stored for playback for up to a week, sometime month. The PCS capability is very limited for tasks, as long term data storage and retrieval, history archiving, data consolidation and report generation. Furthermore data from the lab analyser and dispatch system are normally not interfaced to the PCS. Data acquisition scan time from data of all above mentioned automation system is between 10 to 60 seconds.

Data Pre-Processing, Storage and Archiving

All the automatic read in data, is filtered and pre-processed according data type (e.g. integration of a kW signal to kWh, conversion of silo distance measurements to silo level, combination of a cement type signal and a material flow signal to a amount of finish good produced, calculation of specific fuel consumption out of heat content, flow of fuel, kiln feed and clinker factor etc.).

The different type of scanned and calculated values are then stored in integrated, real-time Database. The data storage structure is optimised for a huge amount of data to be read and retrieved in very short time. Archiving and retrieving to and from a tape or disk unit must be possible under on-line conditions. Raw data lifetime on harddisk is normally 6 to 12 month, on tape between 1 to 10 years (in some special cases as emission data for the EPA, archive data lifetime, must be guaranteed for up to 30 years).

Especially a fast and user-friendly data retrieving engine is important. Otherwise the data is buried in a Data Graveyard with no use at all. A short estimate of the total amount of data on such a system emphasise the importance of this statement:

- Scan time = 1 minute, analog measurements = 1200 tags, amount of alarm and event messages (132 char.) per minute = 1 data lifetime (on disk) = 1 year.
- (4 Bytes per number, 1 Byte per character, 4 Bytes per message)
- per minute: ==> (1200 * 4) + (132 * 1) + 4 = 5000 Bytes
- per day: ==> 60 * 24 * 5000 = 7.2 Mbytes
- per year: ==> 365 * 7.2 MB = **2.6 GB (Data Storage Capacity)**

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Data Evaluation

To analyse this big amount of data it is indispensable to applies specific methods and tools to transfer the raw data to useful information (remember the difficulties to analyse the pyro-process with data on multiple, endless paper charts strips with no physical scale, date, and remarks on it).

A state-of-the-art TIS provides most of the following Data Analysis and Reporting tools:

- Plant Overview Display
  - Gives plant management an immediate overview over actual plant- and equipment-operation.
- Daily / week / month / year Manufacturing Report
  - Summary Report with information about the process, production, equipment status, material stock, quality, shipment etc.. Medium and long-term performance monitoring.
- Operation Log Reports:
  - Short term production and process performance monitoring
- Trend graphs
  - Actual - and historical process status and performances
- Alarm List and Alarm Statistic
  - Shows actual and historical equipment failures and gives maintenance personnel an overview of equipment problems.
- List of running hours, production numbers and process values
  - Enables plant personnel to plan production and maintenance schedules; For special situations also Ad-Hoc analysis can be done and special Reports (Emissions for EPA, ATR for "Holderbank", etc.) can be created.
- Statistical Analysis as Charts, Correlation, Pareto
  - For Quality control, process optimisation and maintenance support.

Manual Data Entry and Data Modification Capability

For the calculation in the pre-processing (see above) the TIS needs manual entered plant constant. Examples of such plant constants are: clinkerfactor, raw material humidity, head content of fuels, etc. A TIS must provide user-friendly functions to enable plant personnel to adjust the constants in a easy way.

Furthermore, a TIS must provide functions to modify and adjust calculated and integrated report values. In contrast to booking numbers in a transaction based ERP business system, every physical sensor measurement has a measurement error (independent how often the sensor has been calibrated). Integration of sensor values result in accumulation of the error. For example in a 2 million tonnes per year cement plant, a 2% error in the produced cement belt weigher system, results in a divergence of 40'000 tonnes of cement. TIS applications showed that automatic generated reports must be checked for plausibility and adjusted accordingly before they can be approved for further use (e.g. as input data for a SAP, ERP system).

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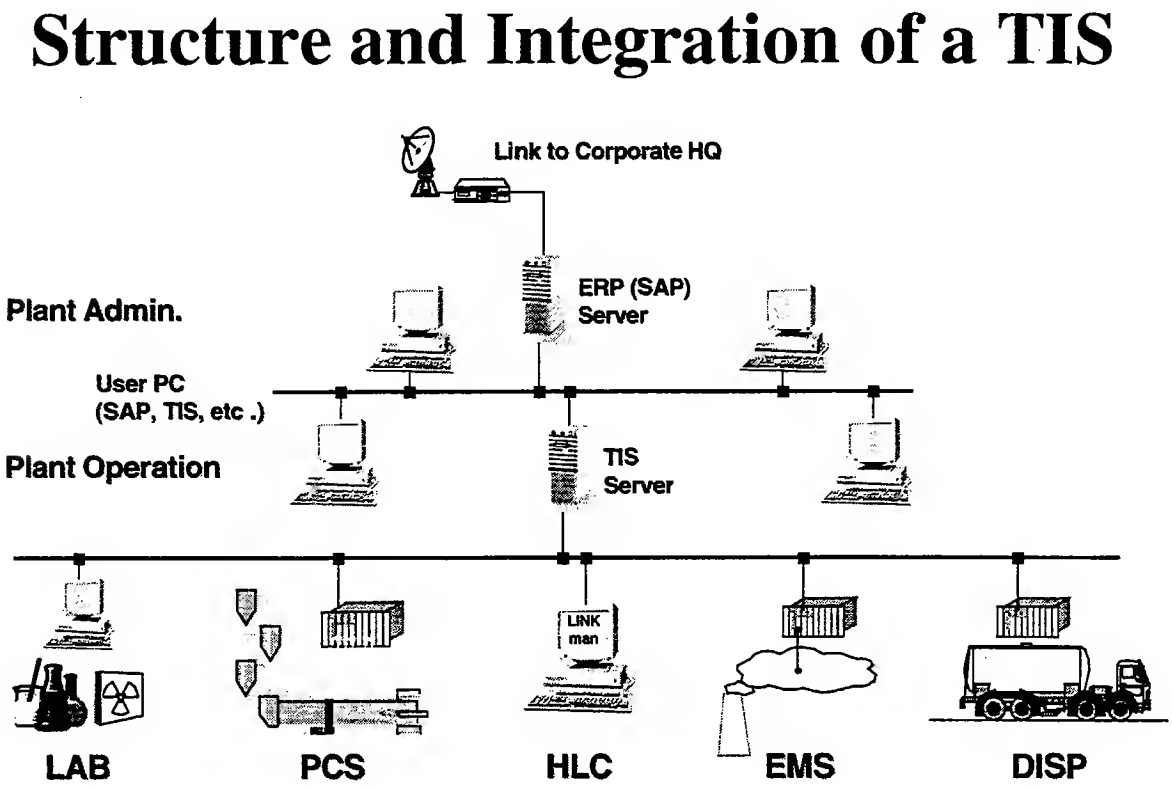
Interfaces to Automation- and ERP System

As shown in chapter 3.1, a TIS System is located in between the ERP business system of company / works management level and the automation system of the process / production control level. This means a TIS reads his input data from one or multiple automation system as Process Control System, Lab System, Emission Monitoring System etc.. Compiled reports can be sent to the upper level ERP System for further treatment. A TIS supplier must provide and support highly reliable interfaces to different brand of automation- and ERP systems.

3.2.3 Structure and Integration of a TIS

The general structure of a TIS and the integration in the plant information management concept looks as follows:

Figure 3:



With this approach, the process related data flows in a structured flow from bottom to top. All systems are networked, manual data entry is minimised, data transfer speed and data quality are maximised.

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What to avoid

In some plants ("Holderbank" and Non-"Holderbank") the automation- and control systems as well as different business computer systems were installed over several years not applying an overall concept.

This "natural grown" computer system agglomeration are typical patchwork solutions with individual computer islands (not using standards).

Such systems are highly complex, difficult to document and maintain and result high operation and support costs. The flow of data is limited, because too many systems and interfaces are necessary. Often manual data transfer is used (data printout of one computer system, data entry typing in the other computer). Such approaches are not user friendly, show slow data transfer, low quality of data and redundant data sets and results in user frustration.

3.3 TIS Systems and Suppliers

The results of different Industry Market Scan (1993 - 1997), executed in co-operation with "Holderbank" Group Companies in various countries, are summarised below. The list shows suppliers, product names and country of origin:

Supplier	Product	Origin
ABB	CIMS	Switzerland
Siemens	CEMAT-MIS	Germany
AspenTech	CIM/21, InfoPlus.21	USA
FLS / Fuller	Plant Guide	USA / Denmark
OSI	PI System	USA
Honeywell	Uniformance	USA

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4. TIS APPLICATIONS AND EXPERIENCES

4.1 TIS System installed and planned in "Holderbank" plants

4.1.1 System installed and running

Company / Plant	Country	TIS Supplier / System	Installed
AB GmbH / Lägerdorf	Germany	Siemens / CEMAT-MIS	1994
BCU / Untervaz	Switzerland	ABB / CIMS	1995
HOLNAM / Holly Hill, Clarksville, Dundee, Artesia	USA	AspenTech / CIM21	1995/96
Alpha / Dudfield, Ulco	South Africa	ABB / CIMS	1997/98
HOLNAM / all 15 plants (cement and slag)	USA	AspenTech / InfoPlus21	1997/98

4.1.2 System planned to be installed (Project approved)

Company / Plant	Country	TIS Supplier / System	planned
HCB / Siggenthal	Switzerland	ABB / CIMS	1998/99
SCL / Chekka	Lebanon	under Evaluation	1998
QCL / Gladstone	Australia	under Evaluation	1998/99

4.2 Some Specific Applications and Experiences, Benefits

4.2.1 Alsen Breitenburg Zement - und Kalkwerke GmbH, Lägerdorf, Germany

(Main use of TIS: Process Analysis, Quality Monitoring, MAC report data)

Cement plant with 2 kilns, with a capacity of approximately 1.6 mio t/y cement.

The company sells more than 34 different types of products and has to keep track of all the sales and quality of the products. Furthermore the new kiln 11 is designed to be feed with numerous alternative raw materials and fuels.

TIS System implemented: **Siemens CEMAT-MIS**

- Technical Concept
  - Siemens CEMAT-MIS Technical Information System installed in 1994
  - The system uses a Server based on a PC (Intel Pentium) running Siemens proprietary DBMS and PC Clients running MS Excel for reporting and data analysis.
  - Trend display and analysing tool (extended analysis possible on integrated Excel tool)
  - Use standard production, operation reports and custom defined daily production report (realised on Excel).
  - Reads process and production data directly from the Process Control System (PCS = Siemens CEMAT Coros LSB / S5). Reads approx. 2000 analog values and process more all alarms and event messages, scan rate = 1 minute).
  - Quality data from the automatic, roboterised lab system (Polysius POLAB) are feed via interface directly to the CEMAT-MIS (event driven).
  - Data from the Emission Monitoring system are send to the CEMAT-MIS via interface.

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- The CEMAT-MIS reads dispatch and sales data once per day from the company mainframe via ASCII file transfer.
- All systems are connected via an Ethernet LAN (H1 and TCP/IP protocol).
- Integrates production, operation, quality, emission and consumption monitoring and reporting.
- More to 16 user PC's.
- Reports on the plant files server (Novell) can be transfered via modem lines at the terminal and grinding plant locations.

♦ Experiences / Benefits
- The CEMAT-MIS is a very reliable and adequate performing system (more than two years of experiences).
- The excellent trending and reporting features provided a tool to analyse and optimise the process easy (for example the ball mill charges). Advantage: TIS calculate the specific energy consumption for each type of cement individually. With this cement type dependent energy consumption trends can be monitored which is only possible with automatic data acquisition and pre-processing via PCS and TIS.
- The CEMAT-MIS was directly used for the commissioning of the new kiln 11. The long-term data storage and trending function provided a excellent tool to speed up commissioning. It even prevented the plant to pay the cost for a damaged EP Filter, because with the data from the CEMAT-MIS the engineers were capable to find the root-cause and the exact time when the damage happened. So repair cost could be turned over to the suppliers insurance.
- Tailor made reports for the MAC Initiative provide automatically, on daily basis the necessary production- and equipment efficiency- numbers. This data were used to calculate the KPI (Key Process Indicators) and are the basis for failure analysis. The biggest benefit from the system is fast and automatic report generation every morning (sustainable). Data accuracy with this system is much better than manual data processing and saves up to 3 man-hours per day.
- Complex process analysis realised with correlation charts provide new perceptions, which help to increase production equipment efficiency.
- System is very well accepted and used by plant personnel.

♦ Further Proceeding and Projects
- In 1998 a SAP R/3 ERP System will replace the now used mainframe system.
- An interface for data exchange between the SAP R/3 and the Siemens CEMAT-MIS systems is foreseen. The planed use of this interface is to transfer form the TIS failure reasons and equipment running hours in the SAP Plant Maintenance Module (PM). A study will check the possibility to use the SAP Production Planning Module (PP) with production numbers from the TIS.

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4.2.2 Buendner Cement AG, Untervaz, Switzerland

(Main use of TIS: Emission Reporting, Overall Manufacturing Report, Energy Reporting).

Cement plant with 2 kilns with a capacity of 1 mio t/y cement.

TIS System implemented: **ABB CIMS**

- Technical Concept
  - ABB CIMS Technical Information System installed in 1994/95.
  - Reads process and production data directly from the Process Control System (PCS), realised with an Allen Bradley PLC 5 and PC based HMI system (reads approx. 600 analog values and process up to 15'000 alarms and event messages, scan rate = 1 minute).
  - Integrates production-, operation, consumption and emission monitoring and reporting.
  - Trend display and analysing tool (extended analysis possible on integrated EXCEL tool).
  - CIMS is a Server / Client solution based on DEC Alpha server running a Oracle DBMS and PC Clients running MS Access and Excel for reporting and data analysis.
  - Use standard production and operation reports and custom defined emission report (realised on Access).
  - CIMS can consolidate (and compress) data to hour, shift, day, month and year values and stores data for on-line access up to 1 year.
- Experiences / Benefits
  - Improved and faster monitoring and reporting of emission data to EPA.
  - Tool to analyse operating and emission data which improved the use of HWDF burning.
  - Reduced drastically the man-hours (up to 2 man-hours daily) needed for manual data entry and manual analysis to generate emission and production reports.
  - A sophisticated Overall Manufacturing Report provide on daily and monthly basis a summary of key process, production, consumption, efficiencies, stock and quality data. Saves production management up to 0.5 man-hour daily.
- Further Proceeding and Projects
  - Expand CIMS for enhanced production reporting and detailed electrical consumption reporting.
  - An interface for data exchange between the SAP R/3 and the ABB CIMS systems is foreseen (flow of material, equipment runhours and condition).
  - Adapted reports for support for MAC Initiative.

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4.2.3 Holnam: Holly Hill, Clarksville, Dundee, Artesia; USA

(Main use of TIS: Emission Reporting, Process Analysis).

Four Cement plants which burn HWDF (Hazardous Waste Derived Fuel). Legislation force them to monitor emission (based on gas analyser and materiel input model calculation).

TIS System implemented: **AspenTech CIM21** (former company: ISI)

- Technical Concept
  - AspenTech CIM21 Technical Information System installed in 1995 and 1996
  - Reads process and production data directly from the Process Control System (PCS), realised with a Modicon PLC and Gensym G2 based HMI systems (reads approx. 200 - 330 analog values per kiln, scan rate = 1 minute).
  - Integrates emission monitoring and reporting.
  - Trend display and analysing tool (extended analysis possible on EXCEL tool).
  - CIM21 runs on UNIX based HP workstation server. The Database is proprietary. PC (under X-Window) can be used as user interface.
- Experiences / Benefits
  - Provide the mandatory (according EPA) emission monitoring, reporting and data archiving.
  - HWDF burning would not be possible without this TIS systems.
  - The system is also used for process data analysis (mainly graphic trend) because trending features of the used PCS are not sufficient.
- Further Proceeding and Projects
  - Upgrade the existing CIM21 system with the new InfoPlus21 (NT based Client /Server System).
  - Install in the remaining 11 plant a InfoPlus21. Use mainly as process data historian and emission monitoring tool.
  - The company wide "Manufacturing Data Integration" (MDI) project will integrate these TIS in the ERP Datawarehouse (Holnam proprietary Data Management System based on Oracle).

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5. TYPICAL PROJECT SCHEDULE AND SCOPE OF SUPPLY

5.1 Project Schedules (typical)

The introduction of a TIS system needs some pre-project investigations resulting in a detailed specification. With this, an evaluation (bidding process) can be executed to find the most appropriate TIS. Normally a TIS is implemented in steps, similar to a SAP, realising the functions with the highest priorities first. The following schedule gives some indications of the whole procedure.

Phase	Actions
0: Pre-condition	The plant must be equipped with a state-of-the-art computerbased Process Control System (PCS).
1: Study	Investigation study to determine plant requirements, elaboration of a concept. Investigate integration in plant /company computer infrastructure. Check of interface solutions to all automation systems and to the ERP system.
2: Project Planning	Creation of a Specification and elaboration of a budget and an implementation schedule.
3: Tendering and Evaluation, Offer	Creation of a tender document. Execution of a system evaluation and selection of a System / Supplier (in some case evaluation can be skipped if a company standard exist). Ask for offer (using specification and schedule).
4: Implementation of Step 1	Implementation and commissioning of TIS basis System. Check of performance, adaption of functions to meet plant requirement, (if necessary).
5: Implementation of Step 2	.....

5.2 Scope of supply (typical)

The scope of supply of a TIS application may vary from plant to plant, as the requirements are different.

Nevertheless there are basic requirements in a typical cement plant that can be satisfied (usually in a first implementation phase). That is why the scope of supply is roughly divided into Basic System and Options. The experience shows, that a cost estimation cannot be given at this place, because functionality, interfaces and numbers of users differ from project to project. But it is important to know, that project costs are higher than system costs, because time and cost intensive tasks (study, specification, evaluation, see schedule in 5.1) must be done prior to the system implementation.

The following sketch gives an overview of different packages of a TIS basis system (1...4) with options (5...7).

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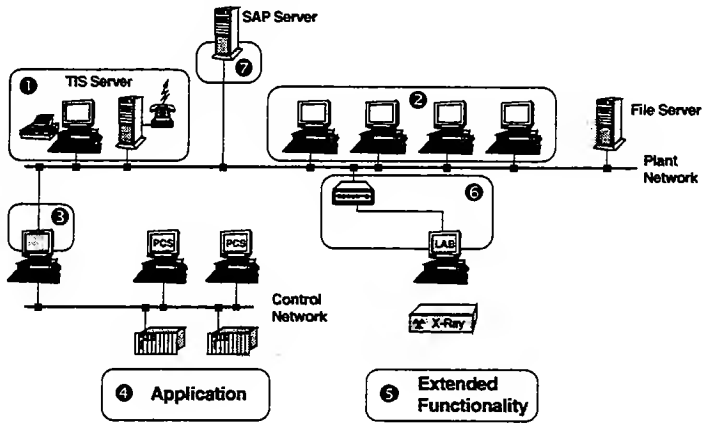
Figure 4:

TIS Scope of Supply (typical)



TIS Basis System (for reference see number in sketch)

1) Server Hardware (HW) an Software (SW), TIS basis SW including DataBase
2) 4 User PC SW license for basis data evaluation
3) Interface driver SW for data acquisition from a Process Control System (PCS)
4) Application applying standard reports and evaluation methods, including system installation and user training

TIS System Options (for reference see number in sketch)

5) Extended functionality as Maintenance Support Functions, SPC, plant specific Manufacturing Reports, etc.
6) Interface to Lab or Dispatch (weighing) system
7) Interface to SAP System, including data exchange concept
   a) Hardware is part of the plant computer infrastructure
   b) not included is set-up and configuration on interfaced system

Assistance from HMC/HES

HMC offers its assistance and experience for TIS Implementation in a plant or group:

- audit
- elaboration of (detailed) specifications and tender documents
- evaluation of offers from different suppliers
- general project assistance
- detailed engineering of reports (optional)
- training for plant personel (optional)
- performing aceptance tests

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6. RESULTS AND CONCLUSION

6.1 Results, Benefits

The Experience and results from different Process Information Management Projects in the cement and other basic industries show the following benefits:

- **Fast and accurate information**
  - Fast and accurate information enables the plant management to see tendencies and to react and direct measures before limits were exceeded. Because reports were calculated automatically, response time is hours or days, **not month.**
- **Open information exchange**
  - All managers which use any kind of integrated Information Management System (not necessary a TIS) confirm that such tools enables open information exchange which improves teamwork drastically and minimises mistrust. This because every user has access to the same information. People share information and work closer together.
    In projects were an interface between the ERP System (e.g. SAP R/3) and the TIS are planned, personnel responsible for the process and administrative personnel form an interdisciplinary team, were both sides start to understand also the problems and the requirements of "the other side".
- **Indispensable for sustainable optimisation**
  - High sophisticated Optimising System as LINKman High Level Control need to be fine tuned and adapted to changing process conditions. Only continuous monitoring of the performance and process conditions with specific analysis tools, as Correlation's, prevent from decreasing system performance.
    In one plant the LINKman runtime can be maintained continuously over 95% with the help of daily performance feedback, which allows immediate reaction to arising problems.
    Permanent monitoring of key parameters and adapted analysis methods like Statistical Process Control (SPC) provides vital information about equipment status and equipment failures. Maintenance Improvement procedures (like the one in MAC) relay on such type of feedback data.
- **Fulfil legislation requirements**
  - Legislation forces us to monitor, analyse and report certain critical values. (e.g emission, use of waste fuel). In USA (BIF) and Switzerland (TA Luft) monitoring, reporting and data archiving of emissions and waste fuel with a TIS, were accept by the local EPA's.
  - Continuous Quality monitoring of shipped cement (e.g.: Germany: Cement Norm VDZ, USA: Mill Certificate) can be realised with the help of a TIS.
- **Saves man-hour**
  - In all plant using a TIS, plant management claim man-hour savings. But the even bigger advantage is, that a TIS frees the engineers from formal work (like data entry, manual data processing) and allows him to use more time on the data analysis.
  - Studies of TIS applications in different basic industries show:
    * Reduced data entry time: 75%
    (reports are compiled in 3 minutes, and checked in 5 minutes)
    * Reduced Paperwork: up 50%

6.2 Conclusion

Considering the importance of relevant information, the actual trends in Information Technology, the indications we have from Technology Watches of non-cement and cement

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industries (our competitors) and the experiences from TIS and SAP applications result in the following conclusions:

- A competitive optimised plant has to be based on computerbased, integrated Information Management System (according CIM concept).

Process Information Management with a TIS is a central pillar of such an "Integrated Plant". Individual users are empowered at the desktop. Process Data feedback and intelligent use of this information is indispensable for a continuous improvement process and will help the plant to be competitive.

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- Ivo Keller, Urs Bleisch, Michel Moser, Thorsten Fuchs; for discussing and providing information about SAP projects concept, plans and experiences.

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Chapter 8

Case Studies

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Engineering - Case Studies

Case studies are not included here.

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Chapter 9

Introduction to "Holderbank" Corporate Engineering (CE)

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Introduction to "Holderbank" Corporate Engineering (CE)

This presentation is not included here.

Handouts will be provided during the presentations by CE.

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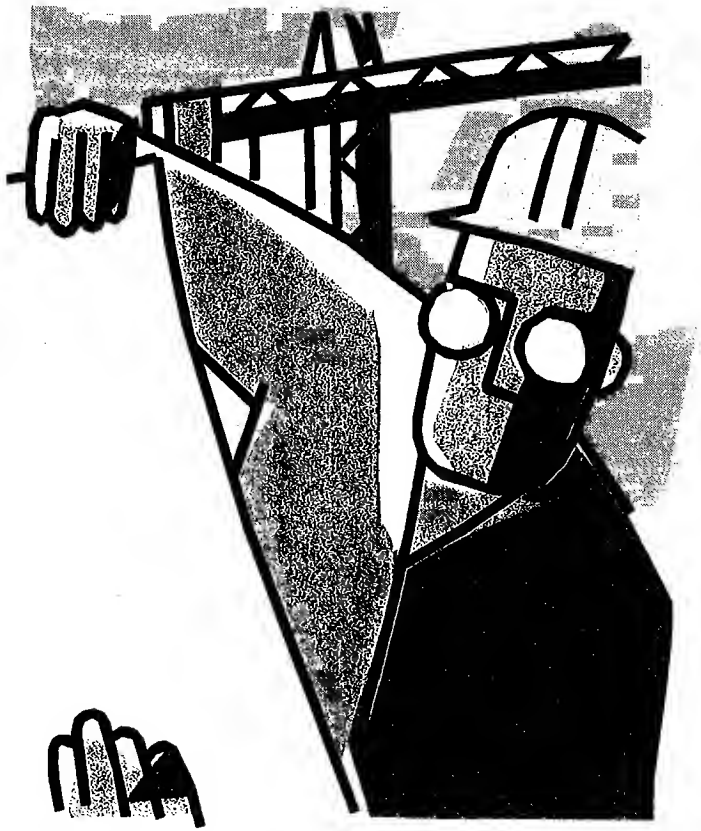
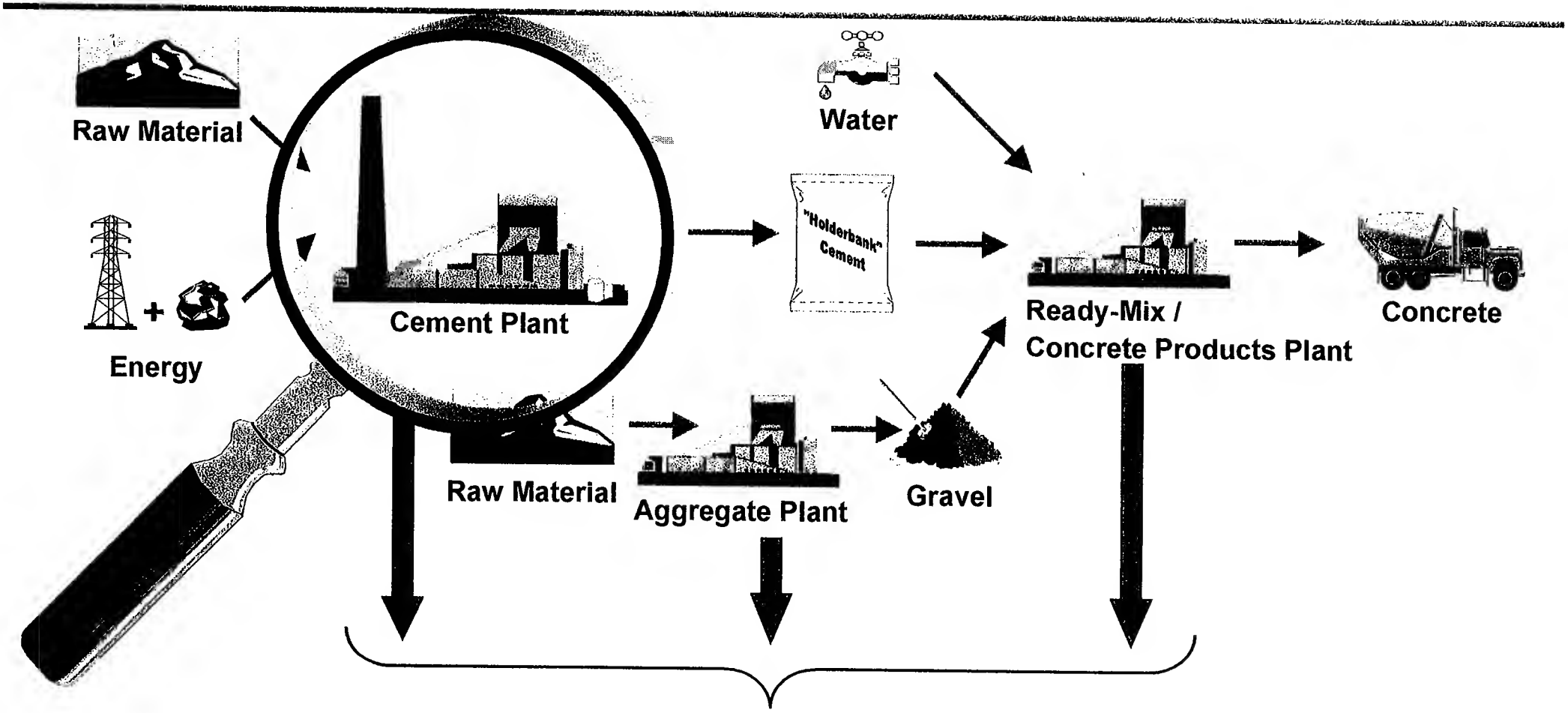
"Holderbank" Corporate Engineering

Duties and Activities



"HOLDERBANK"

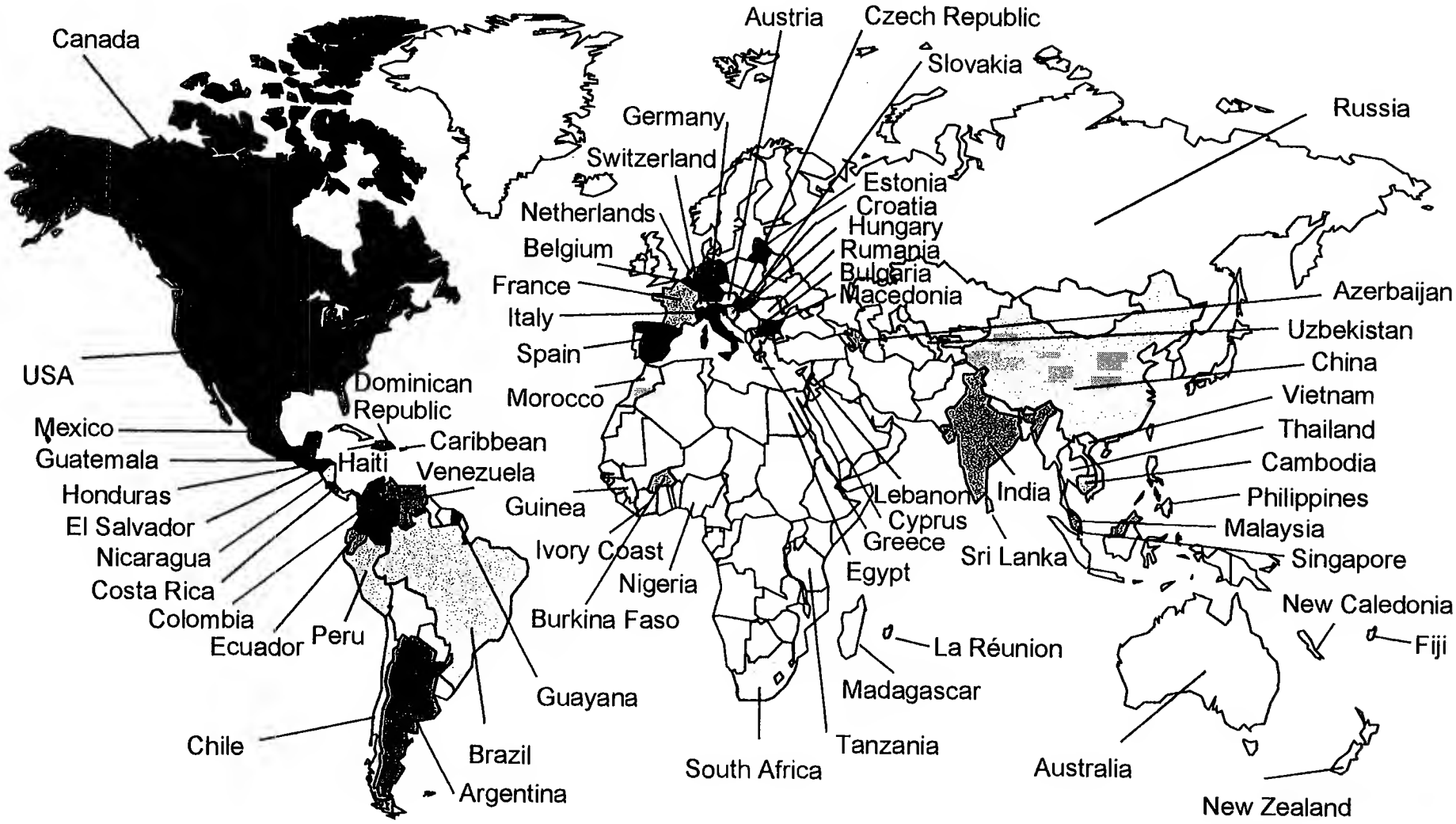
From Raw Material to Concrete



N°s of Plants 1998: 108 Cement Plants (138 Kilns); 245 Aggregates Plants; 687 Ready-Mix/Concrete Plants
Production 1998: 68.4 mio t Cement and Clinker; 83.9 mio t Aggregates; 20.8 Mio m^3 Concrete

"HOLDERBANK"

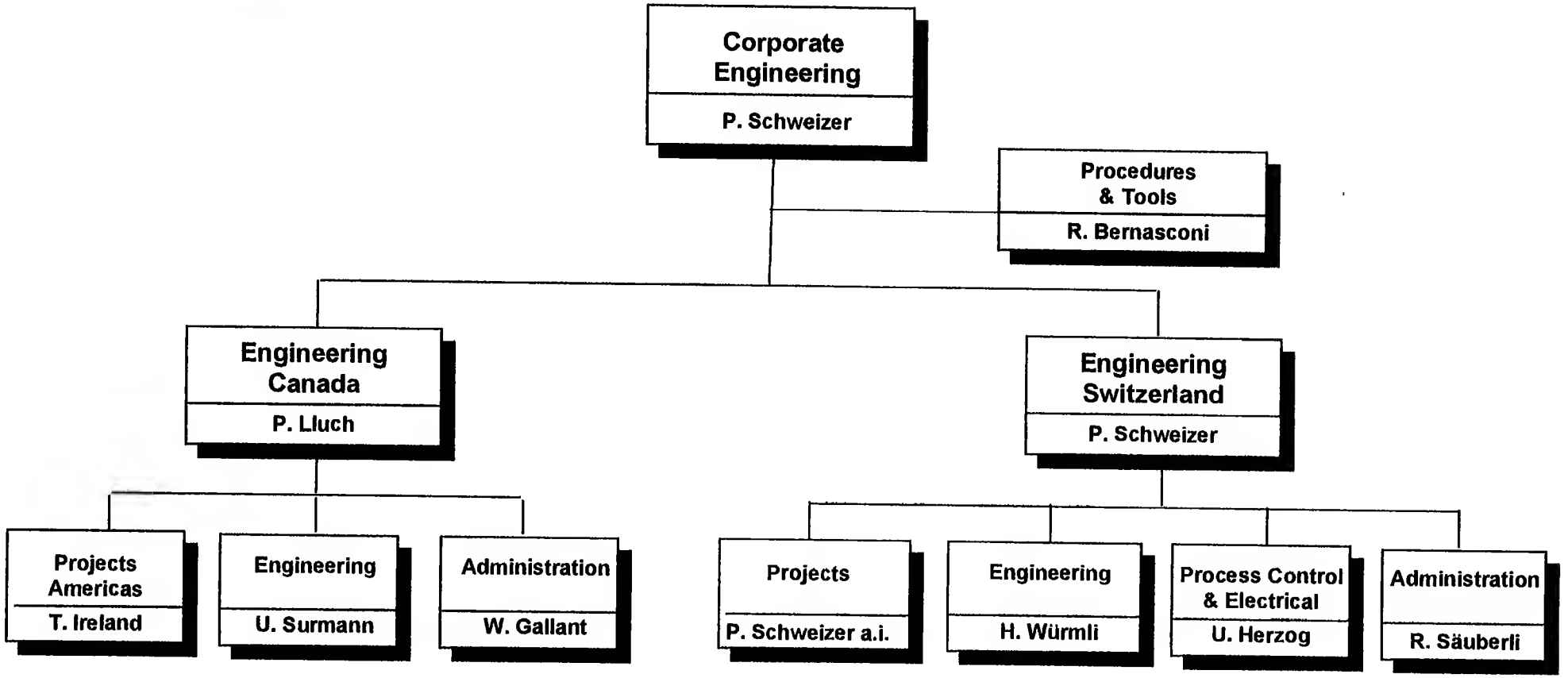
"Holderbank" - in the World*



* Group companies and affiliates

"HOLDERBANK"

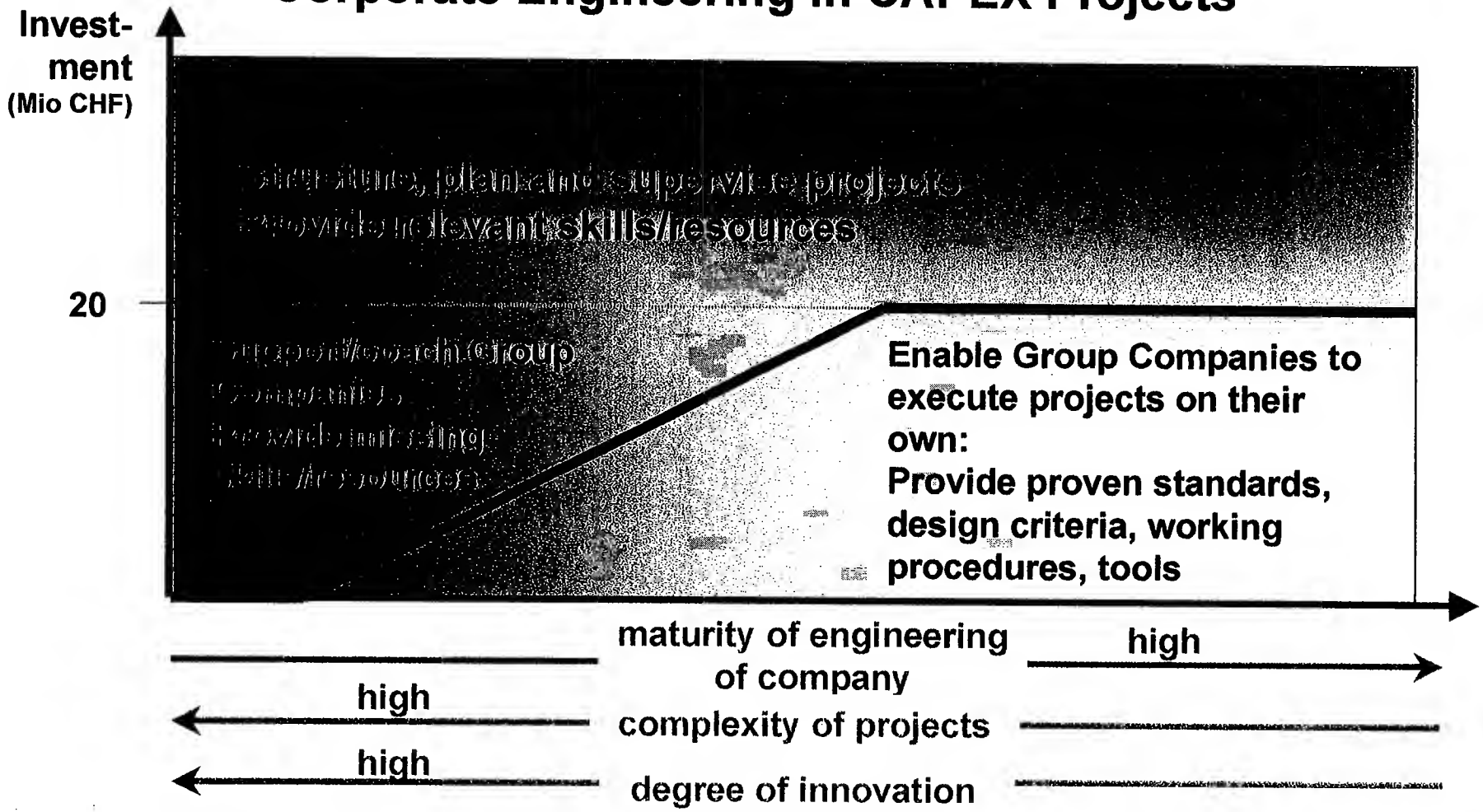
Organization Chart "Holderbank" Corporate Engineering



"HOLDERBANK"

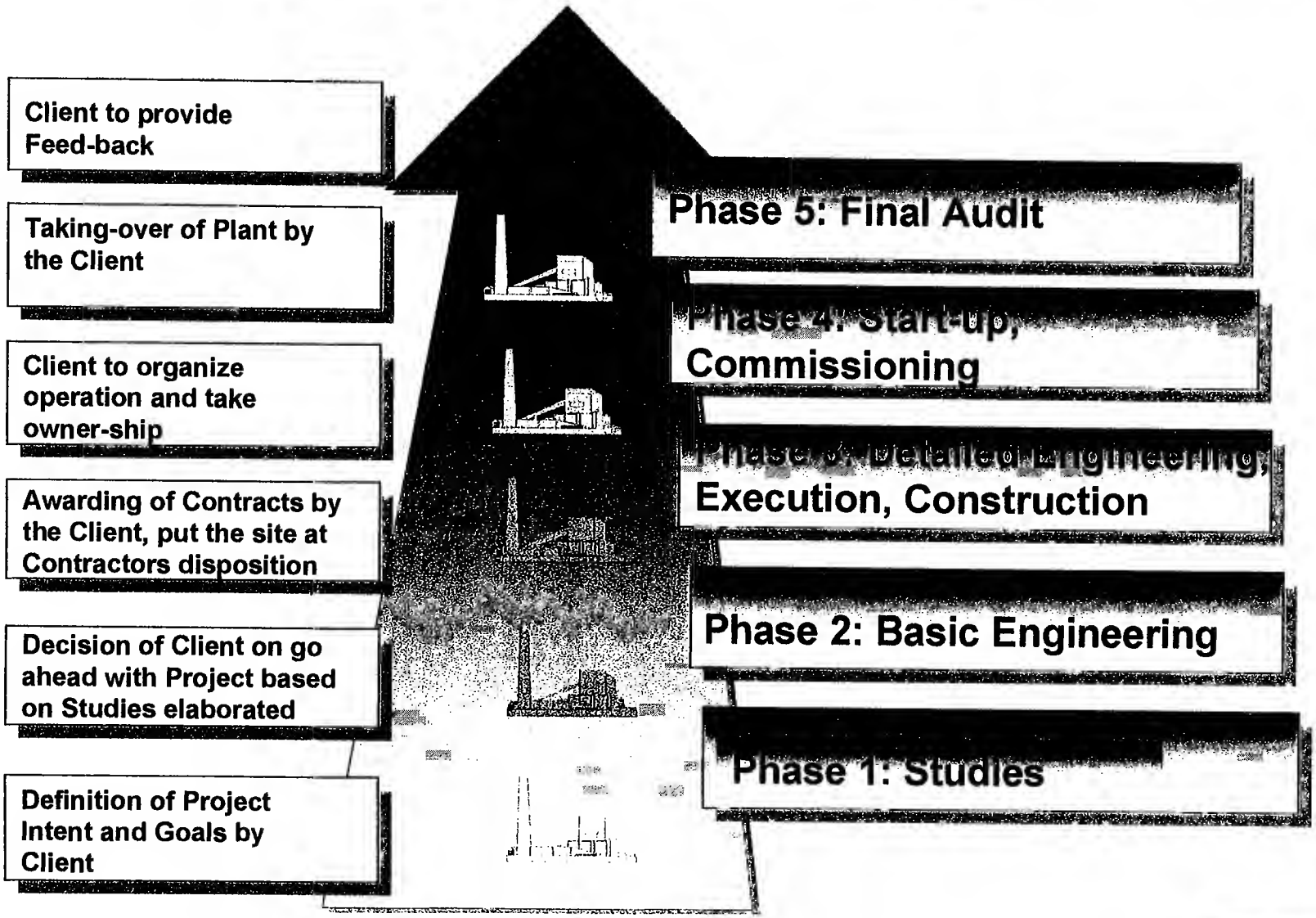
Annex to Mission of Corporate Engineering

Responsibilities and Tasks of Corporate Engineering in CAPEX Projects



"HOLDERBANK"

"Holderbank" Corporate Engineering



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Studies

- Feasibility
  - Market
  - Environment
  - Raw Materials
  - Plant Location
  - Technical Concept
  - Investment/Operating Cost Estimates
  - Economic Analyses
- Audits and Asset Evaluations
- Masterplans

Basic Engineering

- Raw Materials
- Technical Concept
- Plant Design
- Project Organization
- Technical Specifications
- Contracting Plan
- Cost Estimates
- Tender Inquiries
- Tender Evaluation
- Contract Negotiations
- SDC Audits

Plant Master Planning

DRAFT

Guide for Establishing the Plant Master Plan

Summary:

The documental hand defines the objectives of a Plant Master Plan (PMP), positions the Plant Master Plan in the Business Planning process and highlights most important aspects of the elaboration of a Plant Master Plan. In order to give practical support to all those involved in the elaboration of Plant Master Plans a generic table of contents for Plant Master Plans is included.

CB/HES99/4004/E

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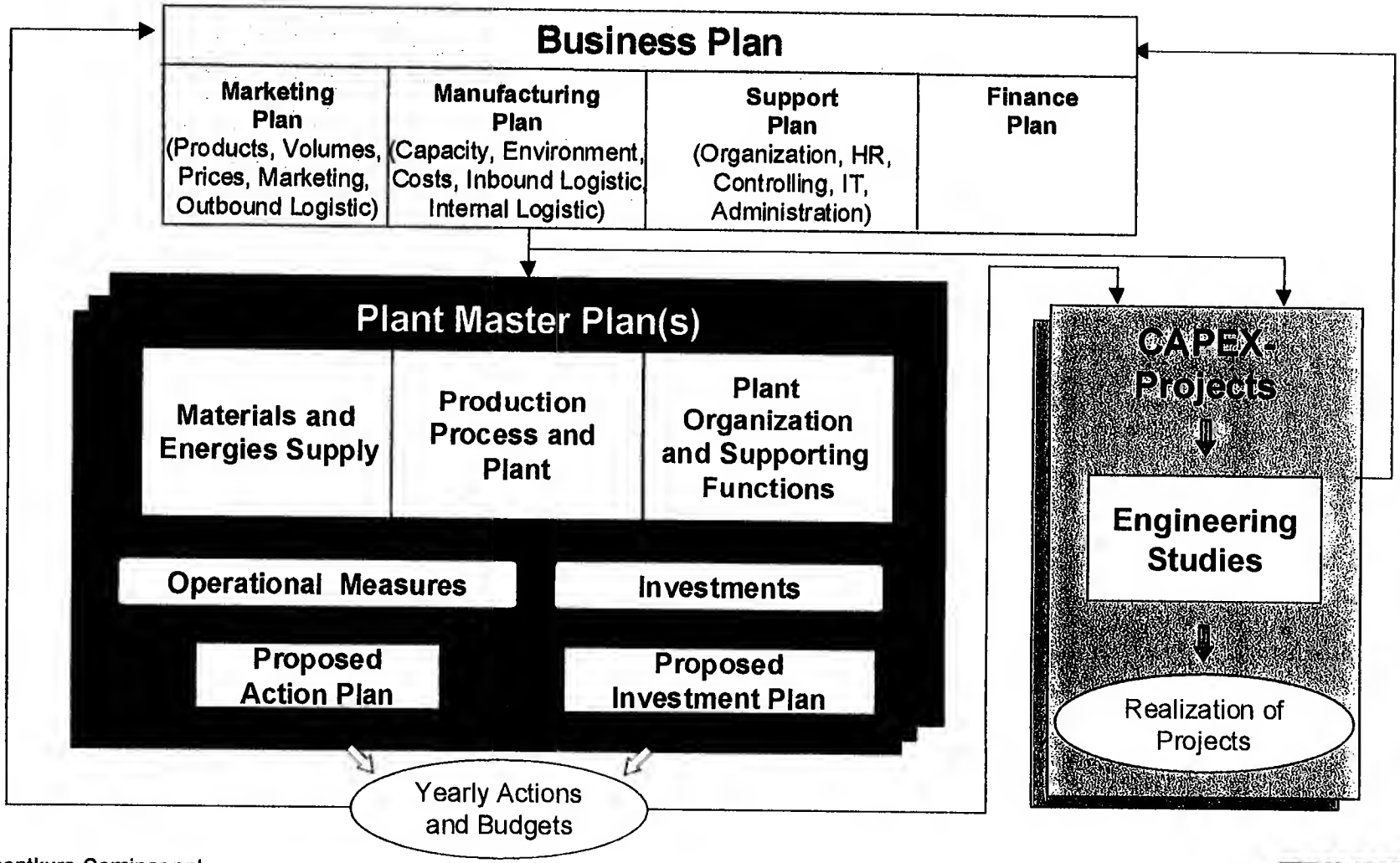
Plant Master Planning

Link between Business Plan and Plant Master Plan



Feasibility Studies

Where it all starts .....

Cost Estimates

Investment Cost Presentation

(ICP)

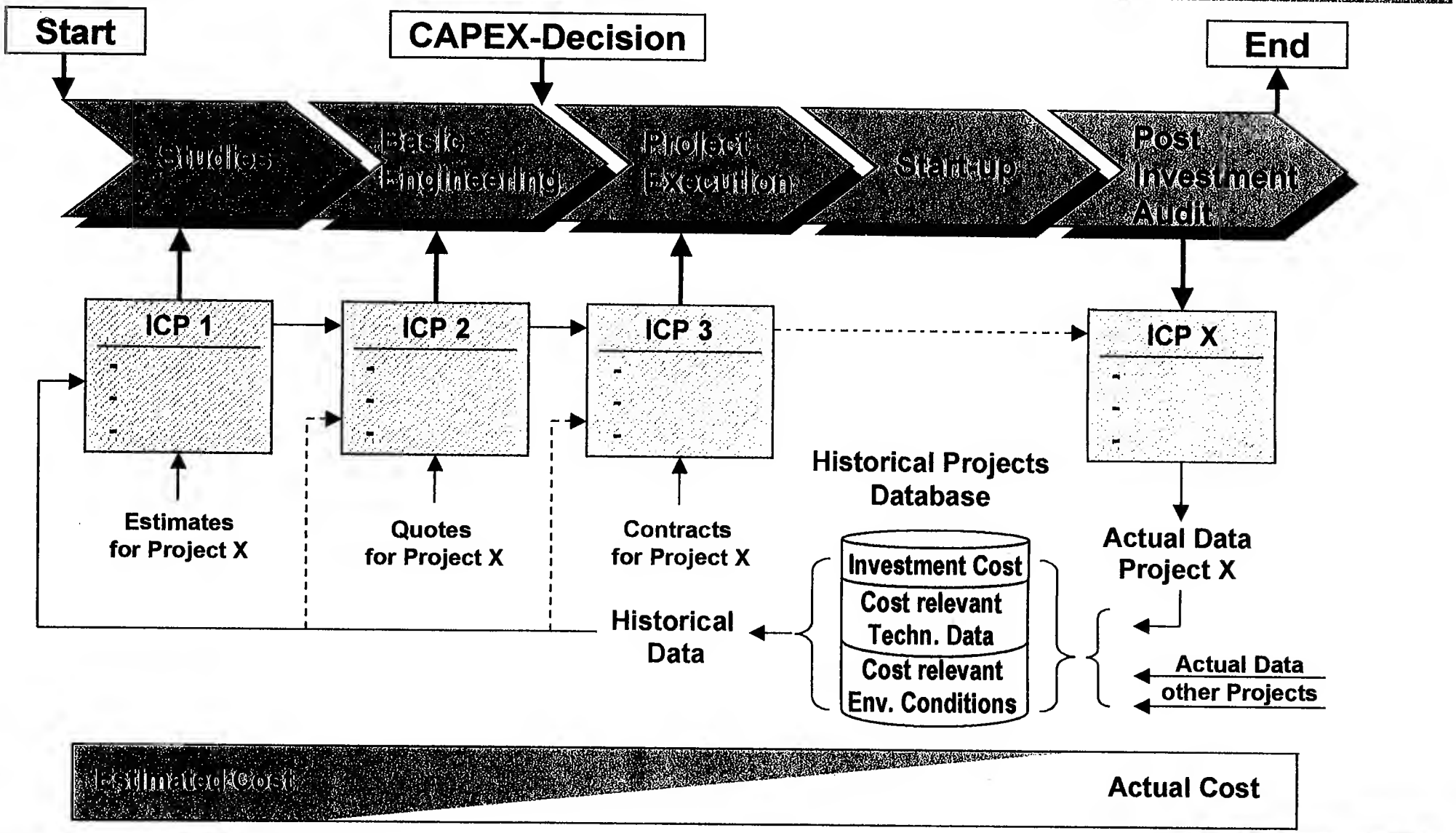
A supporting document to the "Guide to Effective and Efficient Execution of Capital Expenditure Projects"

Executive Summary:

The document at hand presents "Investment Cost Presentation" (ICP), a system establishing structure for and approach to compilation and presentation of project investment cost figures. It also allows meaningful interpretation and fosters comparability of these figures. Project related investment cost data are presented in a set of standardized forms together with cost relevant technical and environmental conditions data. A set of six key specific investment cost figures are made available on the three Investment cost levels "Plant Proper", "Project" and "Total Investment" for project cost judgment and further use in project comparisons

CE/HES97/4003r/E
Revision March 2000

Cost Estimates



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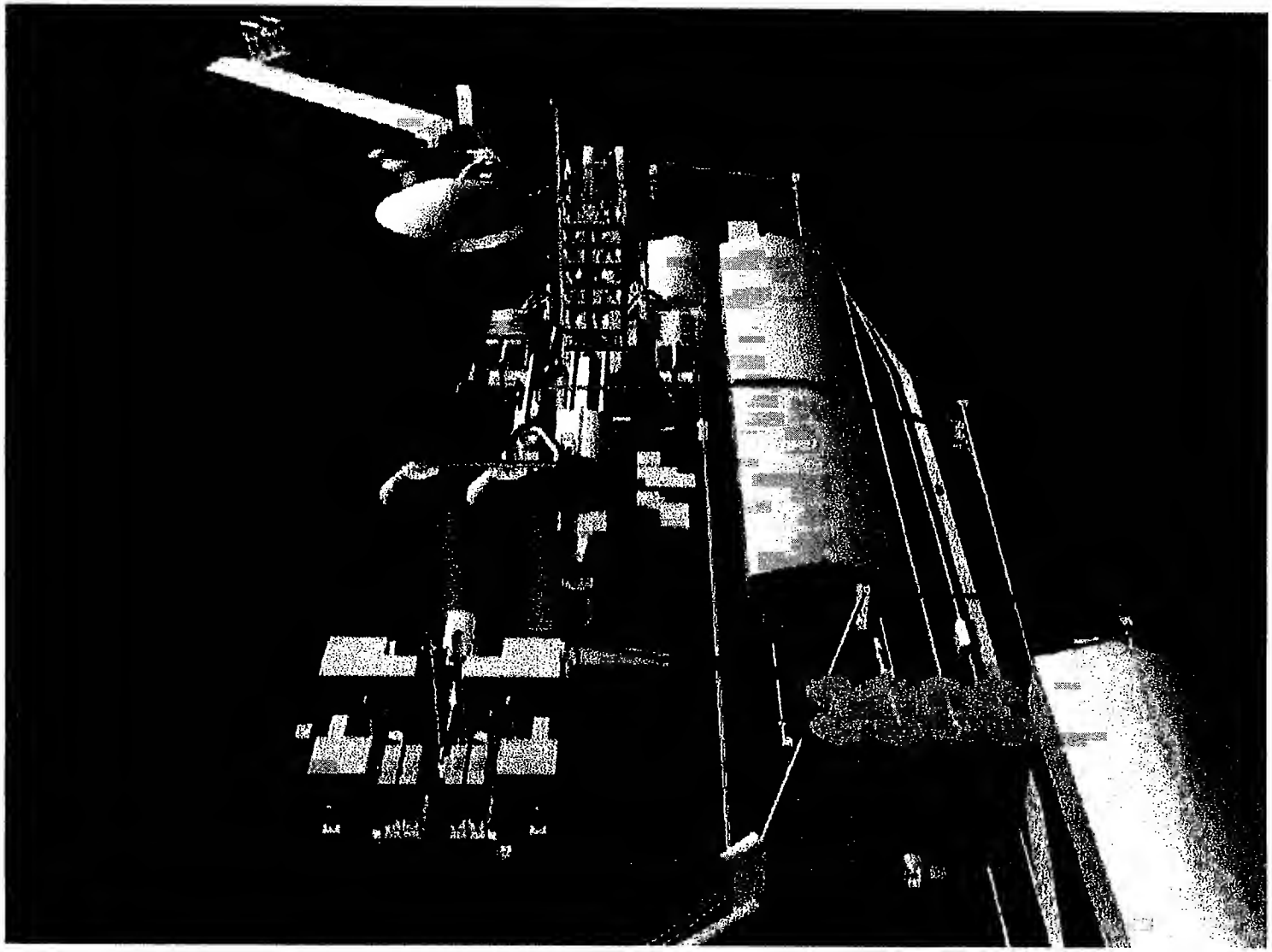
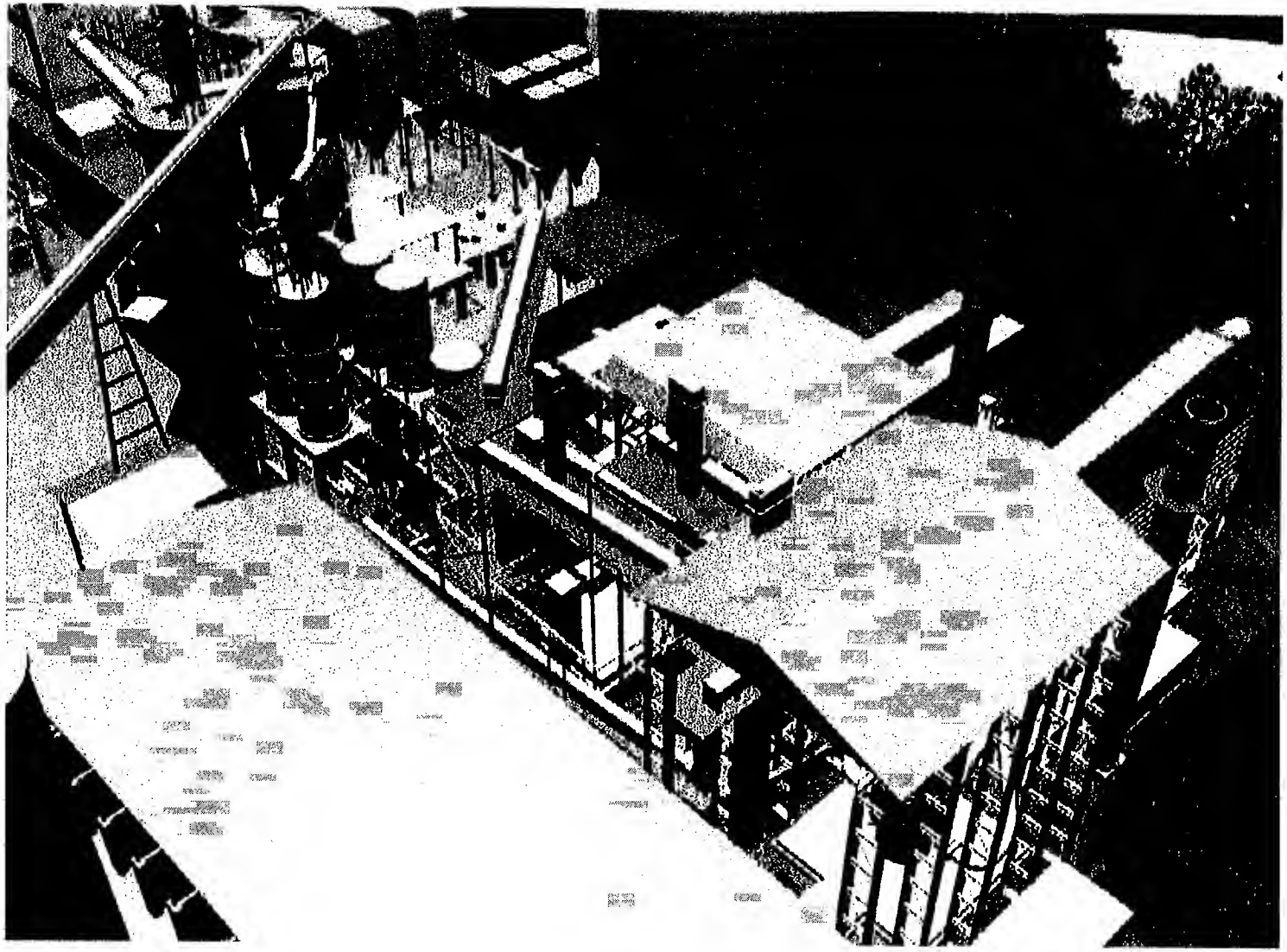
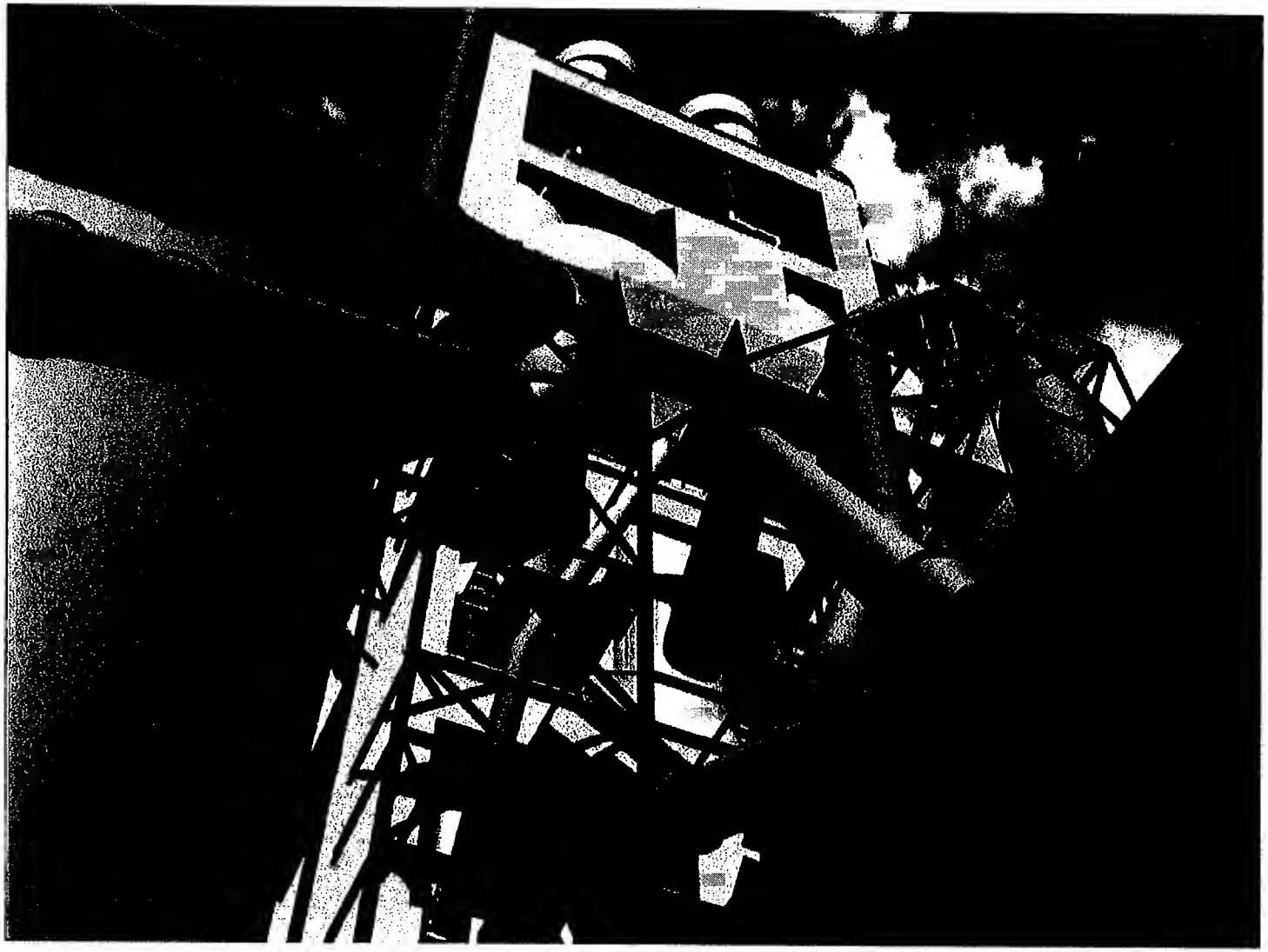
Plant Design



Plant Design



Plant Design



"Holderbank" Corporate Engineering

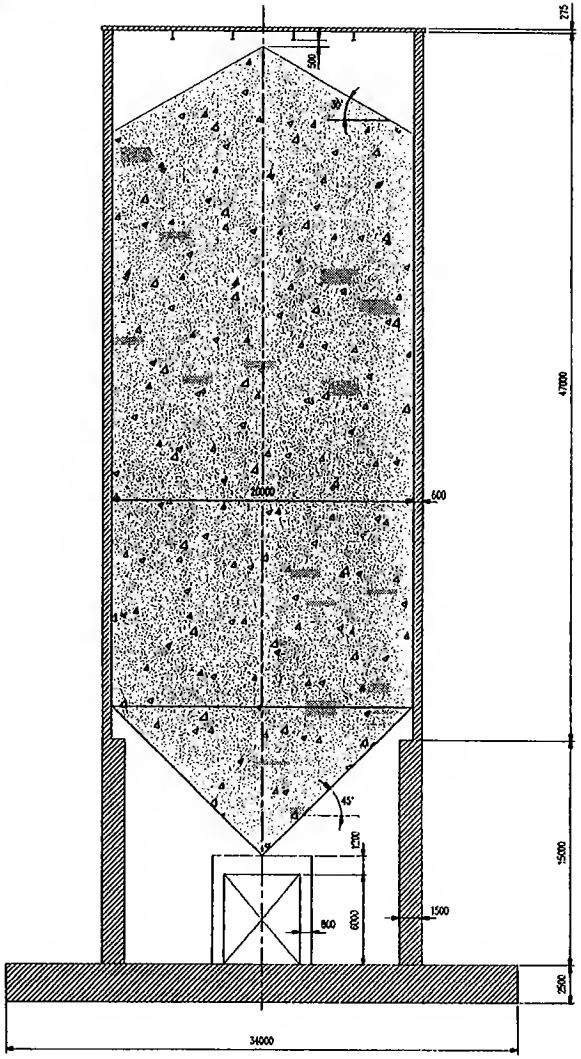
Project Execution

- Contracting Activities
- Detail Engineering
  - Mechanical
  - Elect. + Process Contr.
  - Civil + Structural
  - Infrastructure
- Construction/Erection
  - Site Supervision
  - Site Coordination
- Budget + Time Control
- Quality Control
- Plant Organization, Personnel, Training

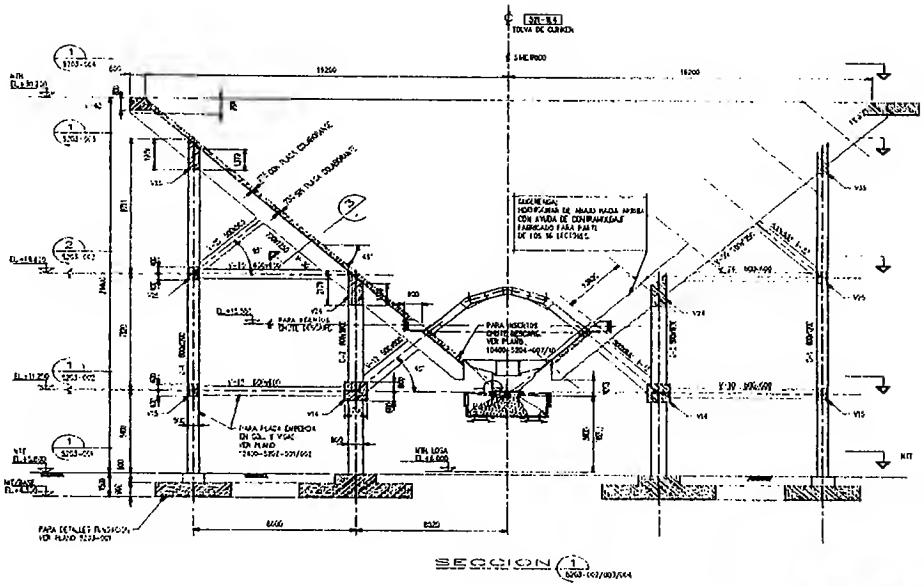
Start-up/ Commissioning

- Integration of Operation Personnel
- Coordination and Supervision
- System Testing
- Performance Tests
- Initiation of Corrective Measures
- Acceptance Certificates

Detail Engineering



Concept matching local conditions and minimizing cost



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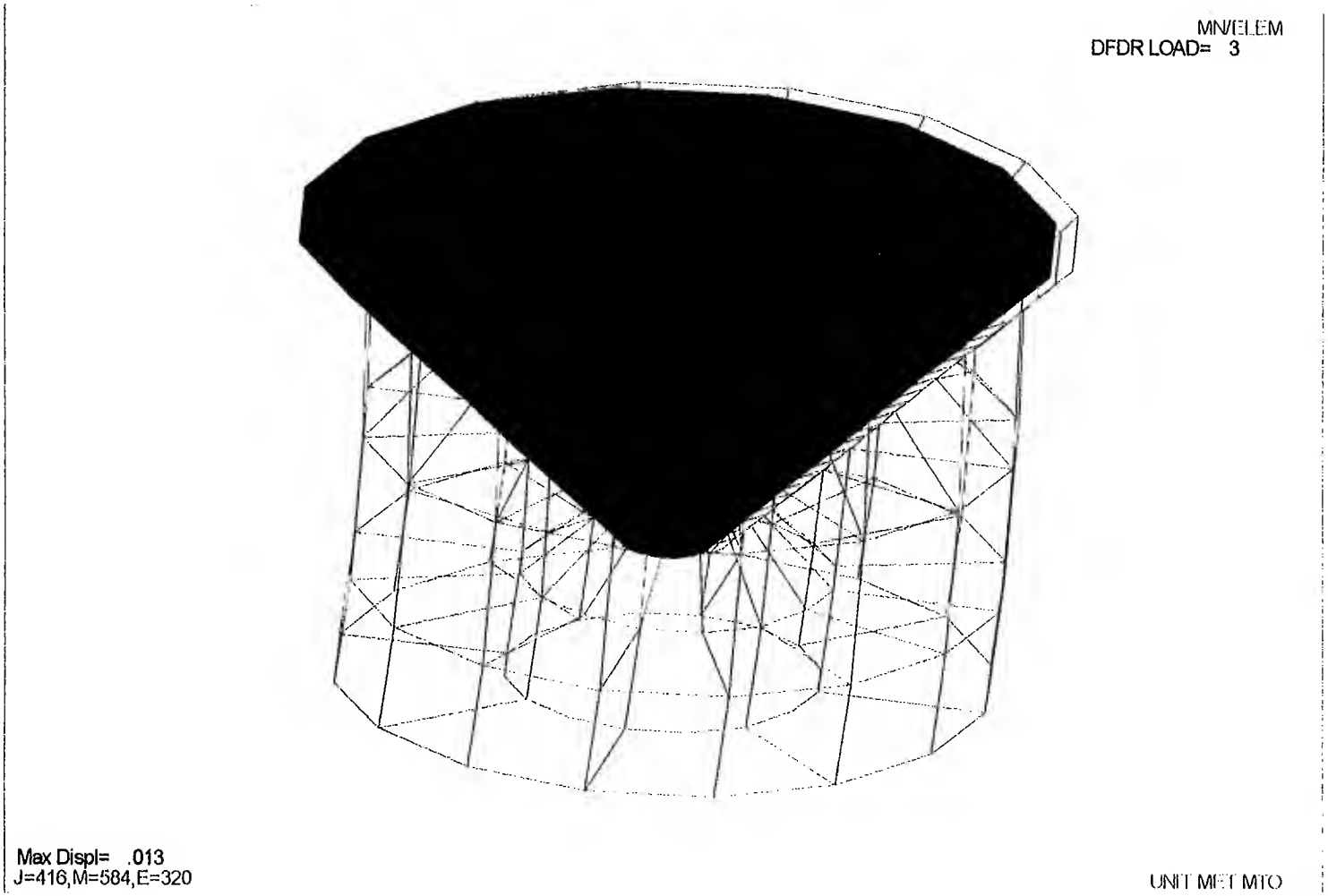
Detail Engineering



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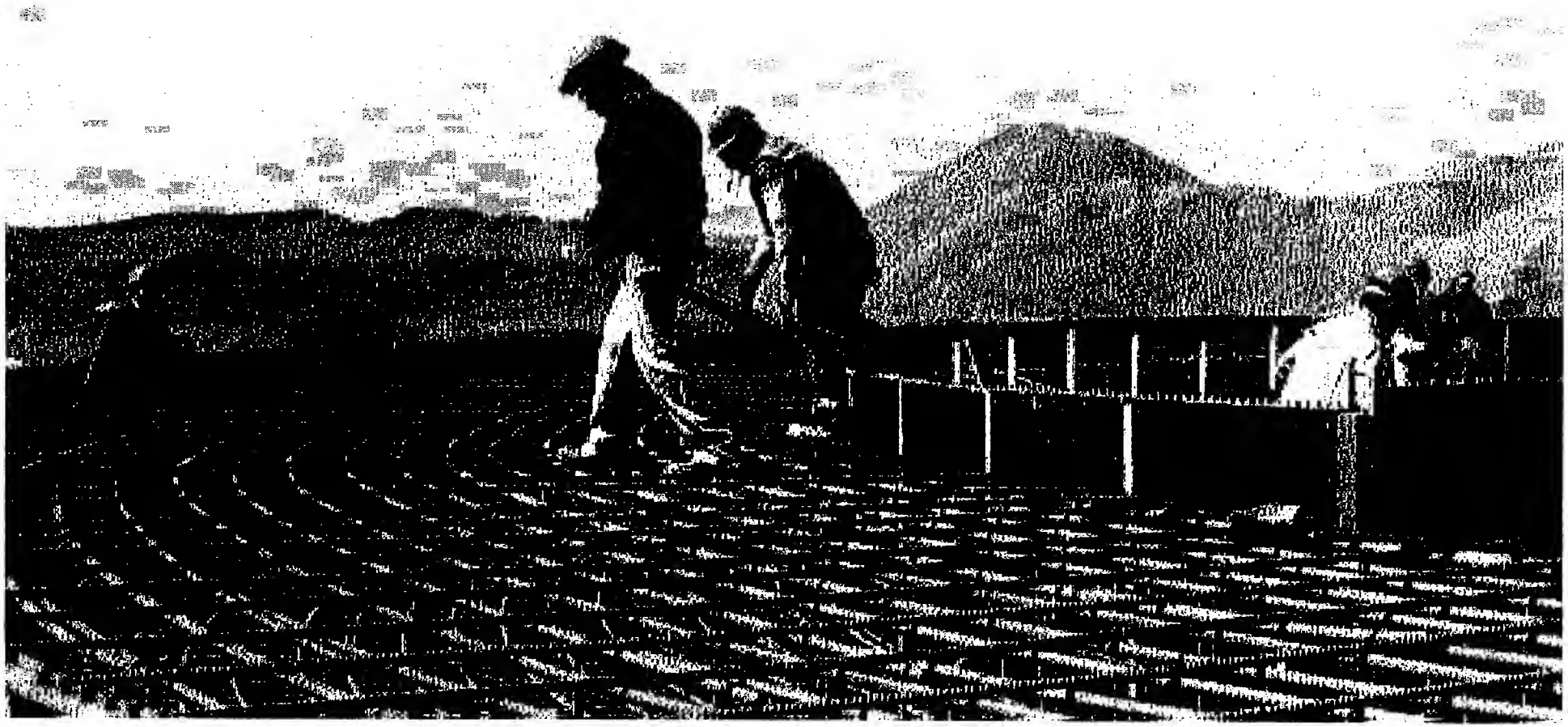
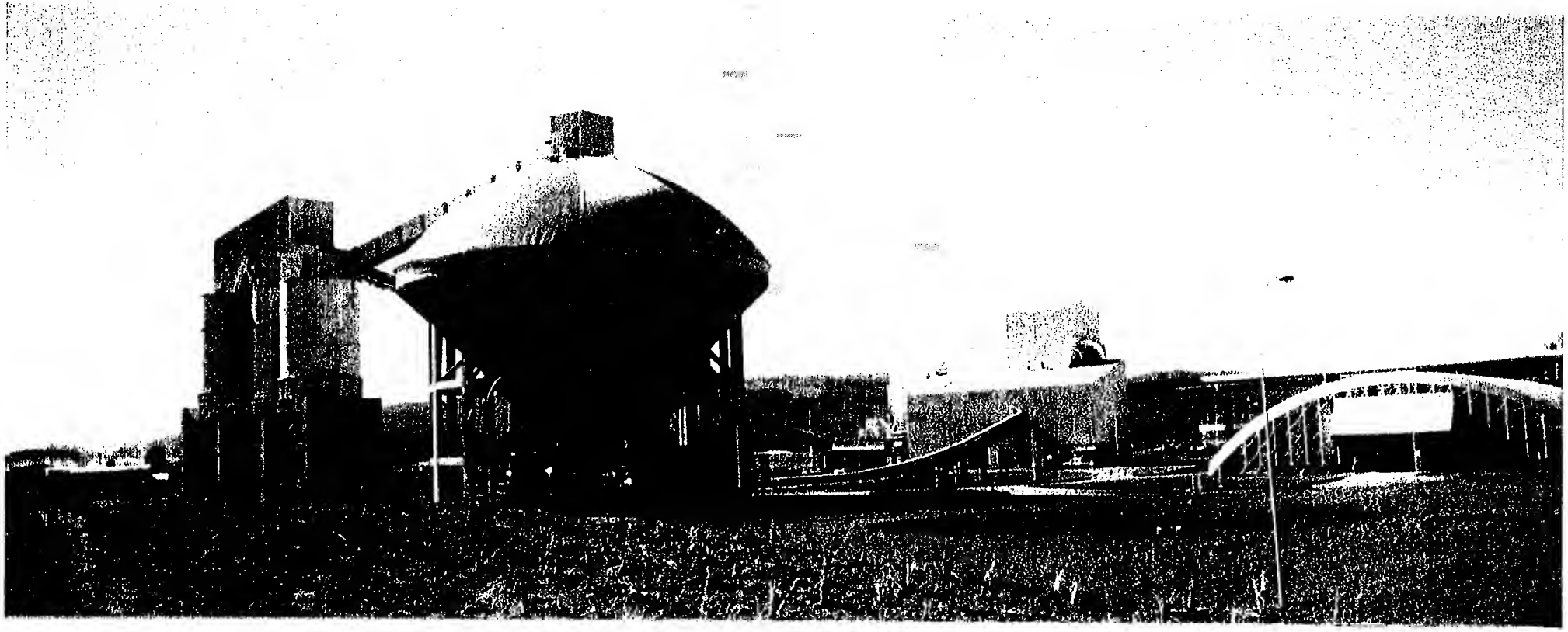
Detail Engineering

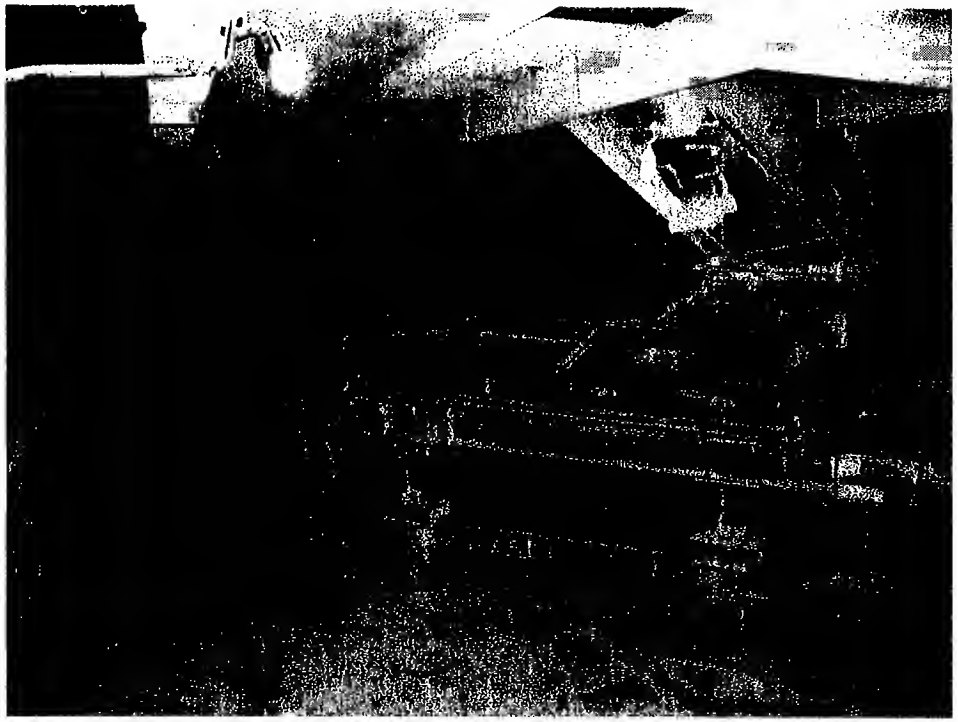


Detail Engineering

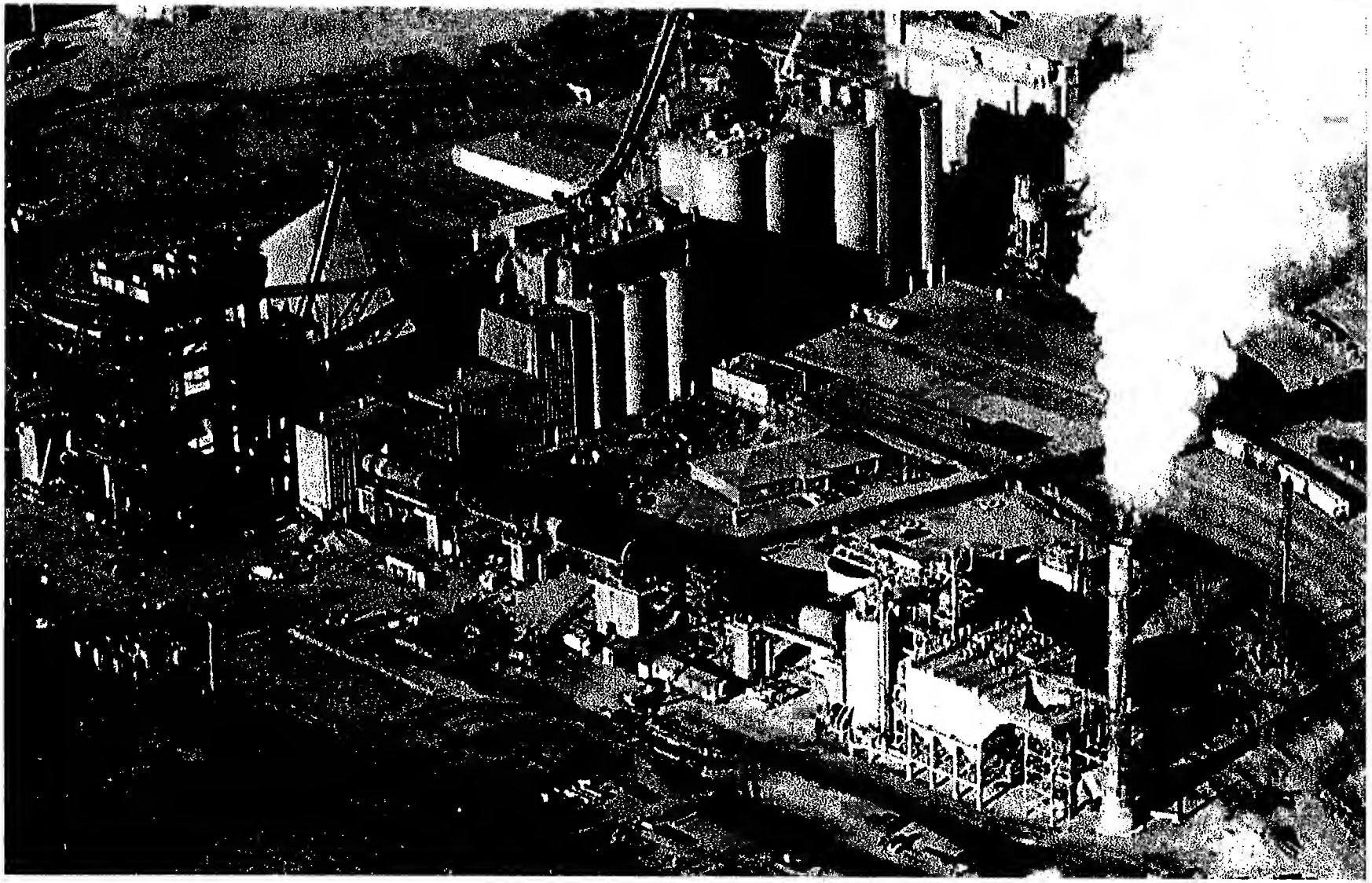


Detail Engineering

Detail Engineering

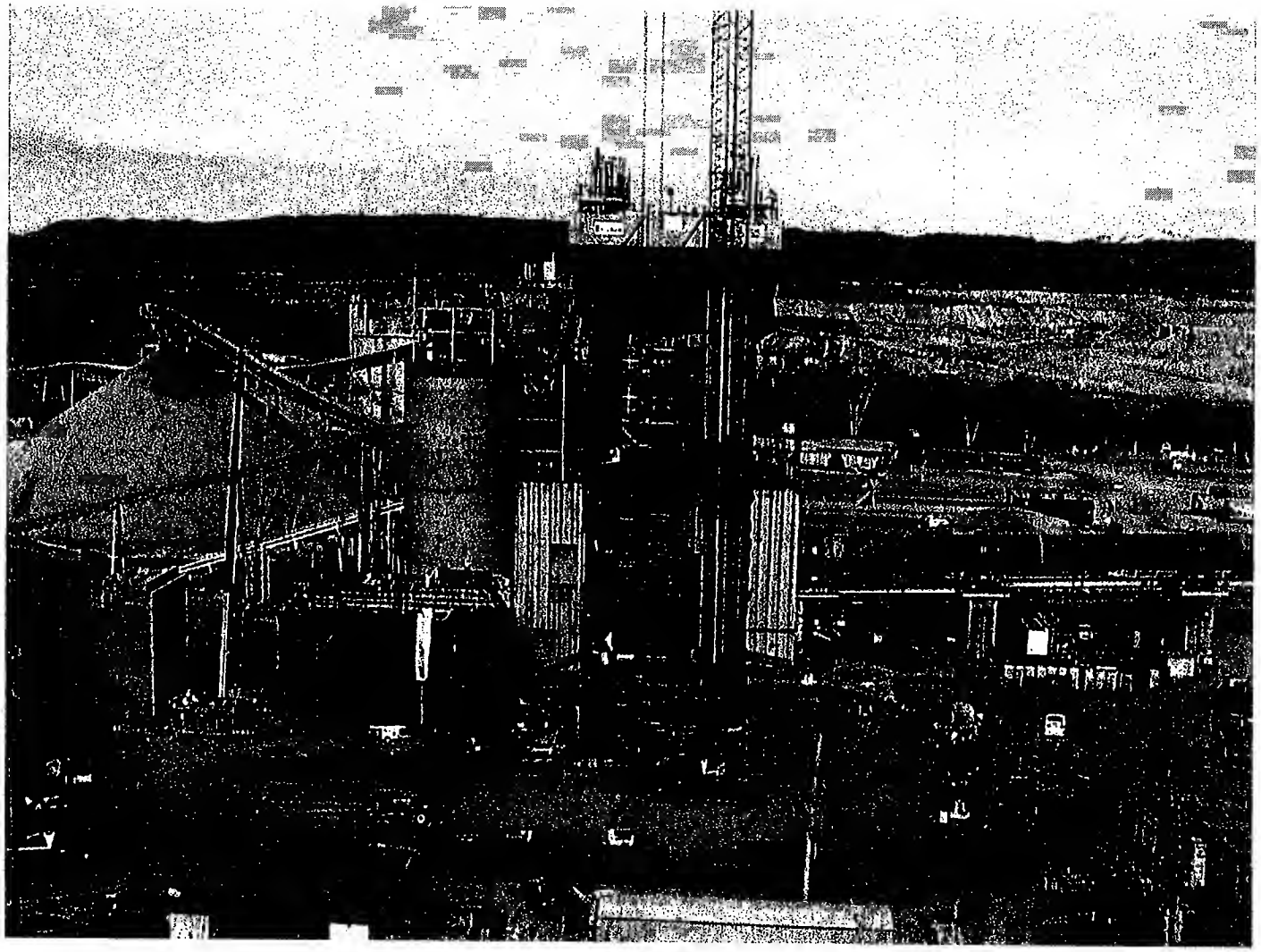


Construction / Erection

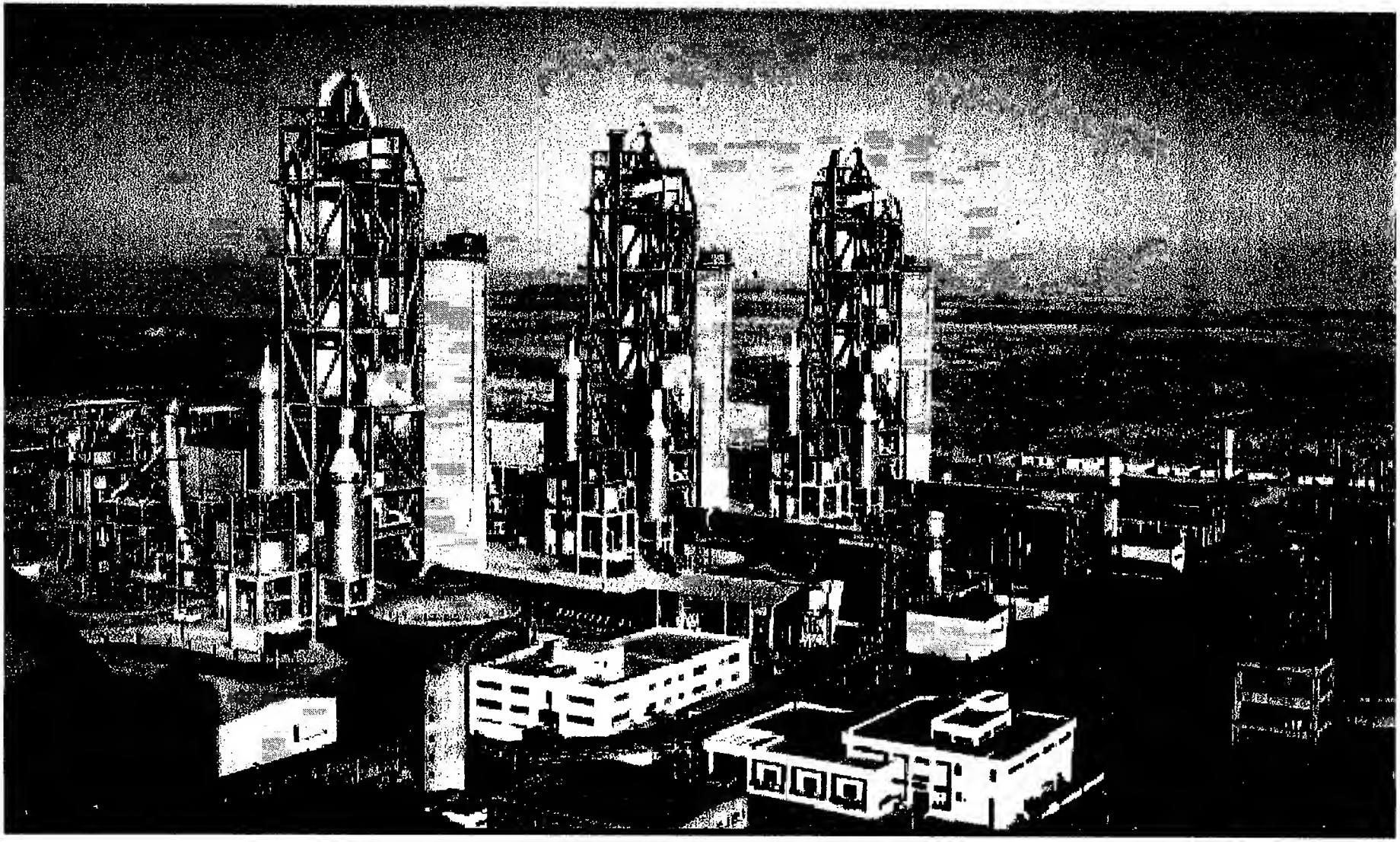


Construction / Erection

Slipforming
Preheater
Tower



Site Supervision



"Holderbank" Corporate Engineering

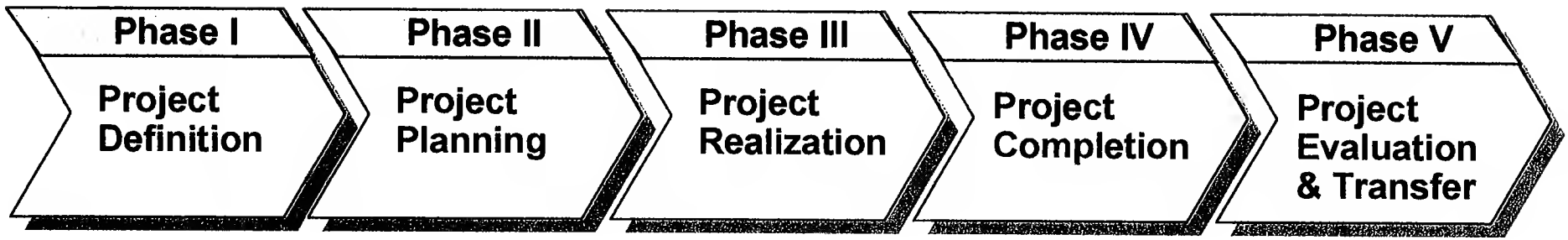
Final Audit

- Evaluate results with the goal to
  - Feedback learnings
  - Evaluate performance with Contractors/Suppliers
  - Improve
- Tasks/Audits
  - Technical Concept
  - Economic Projections
  - Feedback of Operations and Maintenance during first year

Accompanying Task

- Project Management
  - Planning
  - Coordination
  - Budget/Time Control
  - Quality Control
  -

Final Audit



lessons learned from other projects

lessons learned for new projects

Project Management

Budget / Time Control

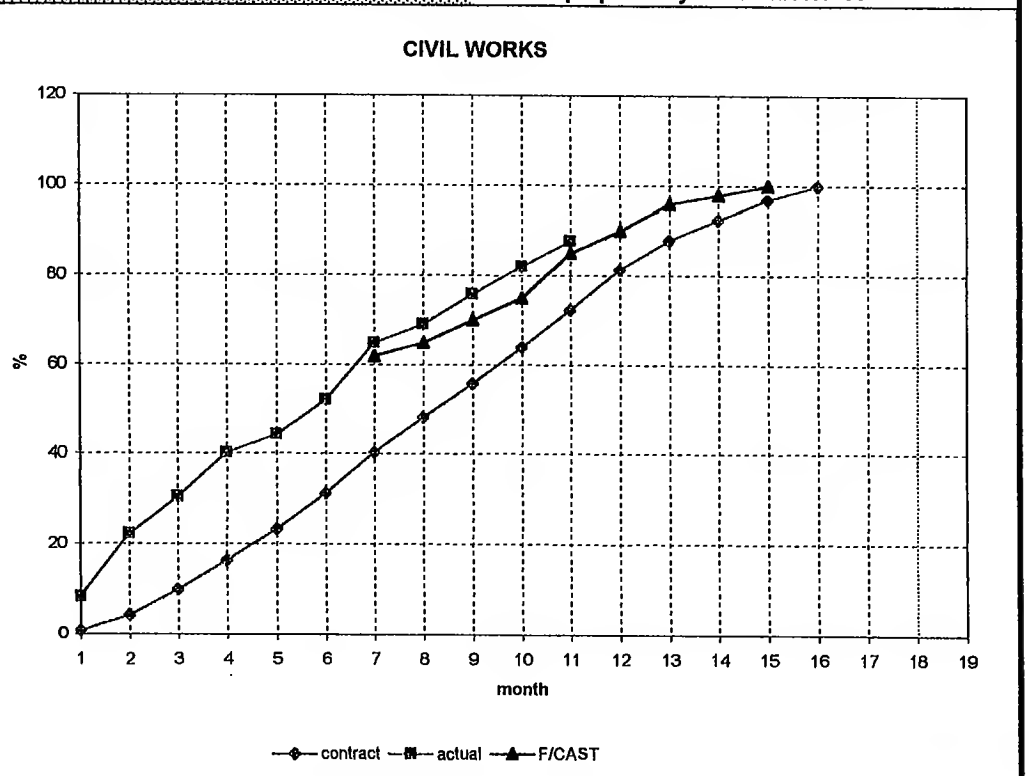
ECC 3 — **CIVIL WORKS PROGRESS** — HOLDERBANK — PROGRESS REPORT MARCH 2000

EGYPTIAN CEMENT COMPANY

CIVIL WORKS PROGRESS & CURVE FORECAST — forecast prepared by E. K. 22.10.1999

	MONTH	No.	CUMUL. RATE contract	CUMUL. RATE actual	PMT. F/CAST
	May	1	1	8	
	June	2	4.5	23	
1	July	3	10	31	
9	August	4	16.5	40	
9	September	5	24	45	
9	October	6	32	52	
	November	7	41	65	62
	December	8	49	69	65
	January	9	56	76	70
	February	10	64	82	75
2	March	11	73	88	85
0	April	12	82		90
0	May	13	88		96
0	June	14	93		98
	July	15	97		100
	August	16	100		

*NOTE! progress now measured in tonnes only



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Chapter 10

General Project Procedures

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General Project Procedures

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1. INTRODUCTION

Nowadays, the implementation of extensive projects, either within the company or in collaboration with others, tends to an increasing complexity and interdependence between various parameters.

In this connection great demands are made upon managing bodies and far-reaching responsibilities are delegated.

This chapter shall give a short overview about project engineering in general in order to serve as a guideline for project implementation.

2. DEFINITION OF PROJECT ENGINEERING

2.1 General

Project engineering has to be understood as a collective name for all planning, supervising, coordinating and controlling measures to be taken in order to find a feasible solution for a project, whereby the procedure to works out the solution, the required staff and means, their engagement and co-ordination is more important than the solution itself.

In order to achieve an efficient project engineering procedure it is important to distinguish between **functional** and **institutional aspects:**

The main features of the **functional aspects** are the planning and control task which are necessary to set and keep a project in motion, containing in particular the following topics:

- Project planning
- Project control
- Project information system

The **institutional aspects** concern formation of working teams, control and decision making bodies as well as their tasks, competences and interdependences. These topics are established within the

- Project organization.

The attached graph (Fig. 2/1) shows the interconnections between project engineering and a typical proceeding pattern.

It is obvious that the problem solving cycles which are composed of

- Situation analysis
- Objective fixation
- Synthesis (patternwise assembling of individual solutions)
- Analysis (of the generated solution concepts)
- Evaluation
- Decision

come up for each stage of the project proceeding pattern.

Furthermore, it shows the progress from the rough to the detailed degree of elaboration following stage by stage and the decreasing probability of discontinuance.

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Coming along with the progress of the project the degree of knowledge about the system/project increases, whereas the degree of acceptable unawareness about the system/project decreases.

(See Fig. 2/2)

Figure 2/1 Proceeding Pattern, Project-Engineering

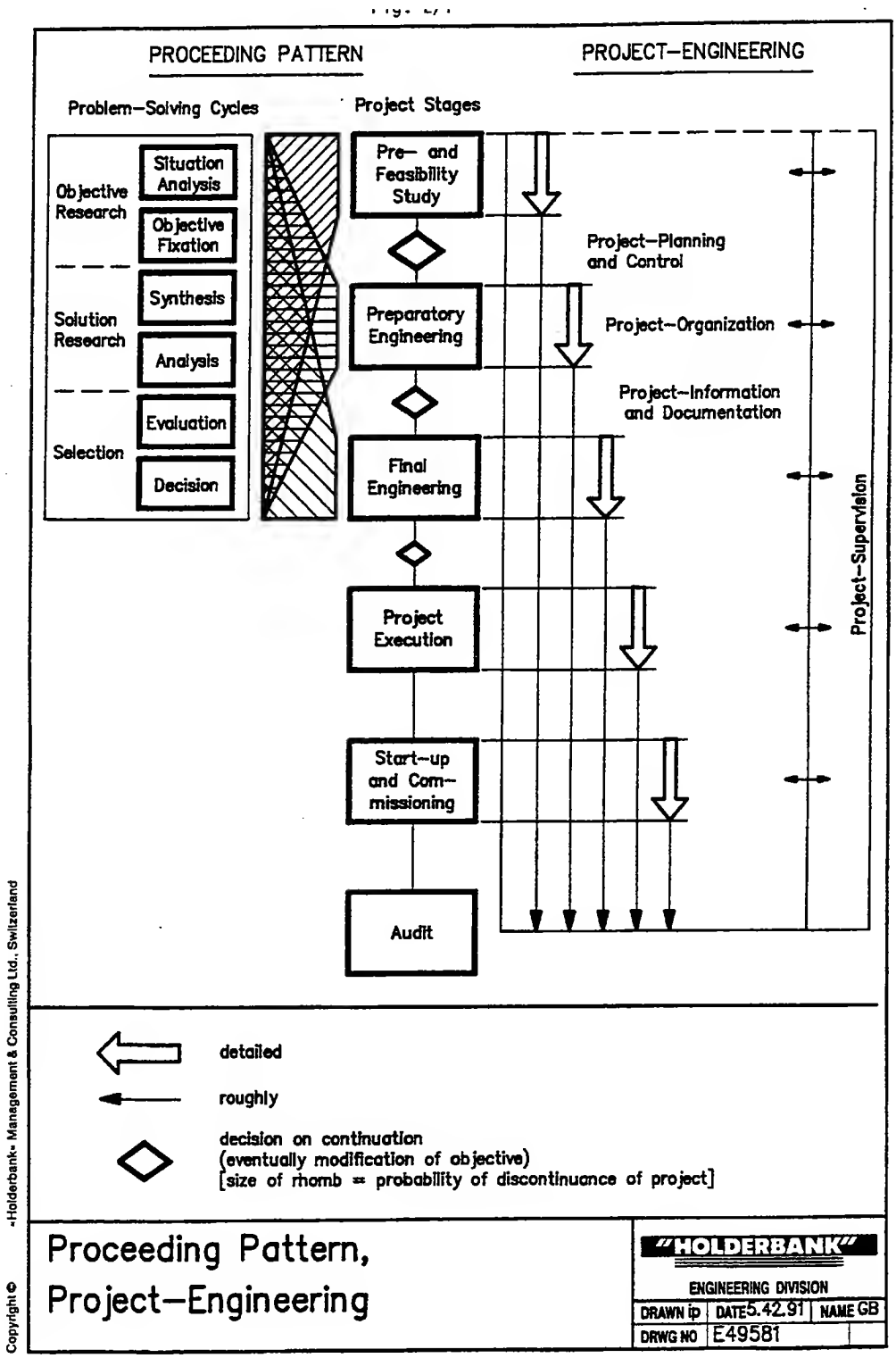

Figure 2/2 Degree of knowledge in relation to the progress of a project

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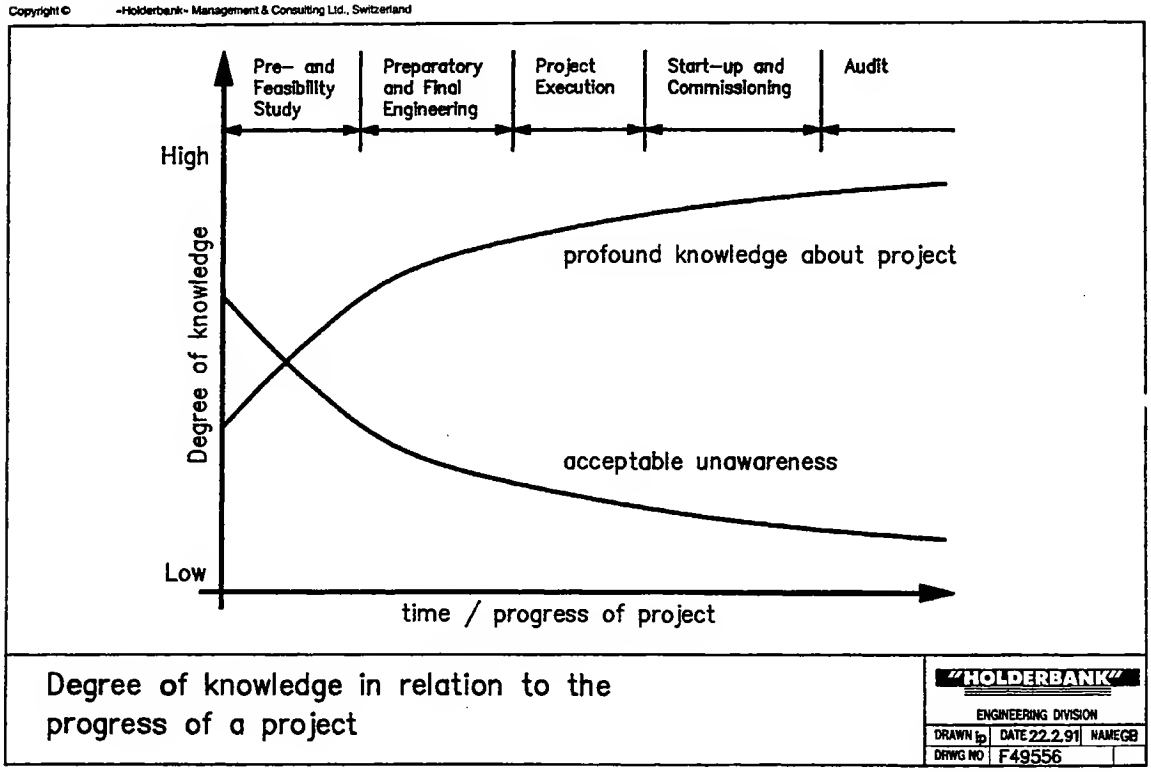

Degree of knowledge in relation to the progress of a project

"HOLDERBANK"
ENGINEERING DIVISION
DRAWN bp | DATE 22.2.91 | NAME GB
DRWG NO F49556

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2.2 Functional Aspects

2.2.1 Project Planning

The project planning comprises mainly a determination of intermediate objectives, planning of an implementation organization, cost estimation, time schedule and of a project information and documentation system.

The existence of plans enables an efficient communication among the people involved in the project and a significant alignment of their engagement for achieving the project goals.

The above-mentioned aspects of planning can be represented in a succession, whereby the observance of the file listed below is recommended as suitable for most cases.

a) Planning intermediate objectives
A clear definition what the results shall be of each project stage.

b) Definition of individual tasks
The better the definitions of individual tasks, the easier to co-ordinate the realization of a project (concepts, engineering).

c) Termination strategy
The investigation of the interdependencies between individual tasks shall lead to a clearly structured termination strategy, evident for all people involved.

d) Requirements estimation
Estimation of the required personnel, time and means in regard to quality and quantity for each defined individual task.

e) Organization planning
See project organization para 2.2.4.

f) Cost estimation
Based on the requirements estimation (d) the cost for the project realization shall be estimated (including additional expenses for training of personnel, consulting etc.).
In order to avoid unrealistic estimations it is advisable to talk to the designated responsible person.

g) Target - dates (milestones)
So as to have a proper progress control and thus a more efficient project control it is recommended to set target-dates (milestones).

h) Budget
A budget comprises all costs which are required for the implementation of a task and have been approved by an authorized body.

i) Project information system
See para 2.2.3.

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2.2.2 Project Control

Project control means that necessary measures are taken to specify and correct actions in order to carry through the decisions taken during the project planning phase.

The following measures could be a part of project control:

- Assignment of tasks, competences and responsibilities as far as this hasn't been done during the project organization phase.
- Introduction, motivation and protection of collaborators.
- Supervision of the problem solving progress (surveying and interpretation of relevant values).
- Taking measures if plan deviations occur (including correction of plans and goals).
- Co-ordination of
  - different project stages
  - client and project team
  - different working teams within the project team

2.2.3 Project Information System

If people shall be able to act in a co-ordinated pattern they must have sufficient information at their disposal.

Therefore, a well-organized information system is required in order to provide the right information to the right person at the right time.

One can distinguish in two main types of information:

- **orientation information** within the project team such as decision minutes (concerning goals, concepts, strategies), problem-related information with direct impact on the task of concerned people, instructing information etc.
- **control information** which characterize deviations from the planned implementation procedure.

In this connection it is an important task of the project engineering to coordinate the activities of the various working teams.

The required reports during the project progress have the purpose of putting on record the elaborated knowledge and to provide it for subsequent project stages and projects.

All members of the project team and people concerned must be given insight into these reports in order to avoid overlapping task and engagement as well as loss of time.

A general idea to the creation of such an information system is attached (Fig. 2/3).

The main positions which give or receive information are itemized. The connection lines represent the ways of information exchange which can be realized either visual or in writing (depending on the importance and/or the extent of the information).

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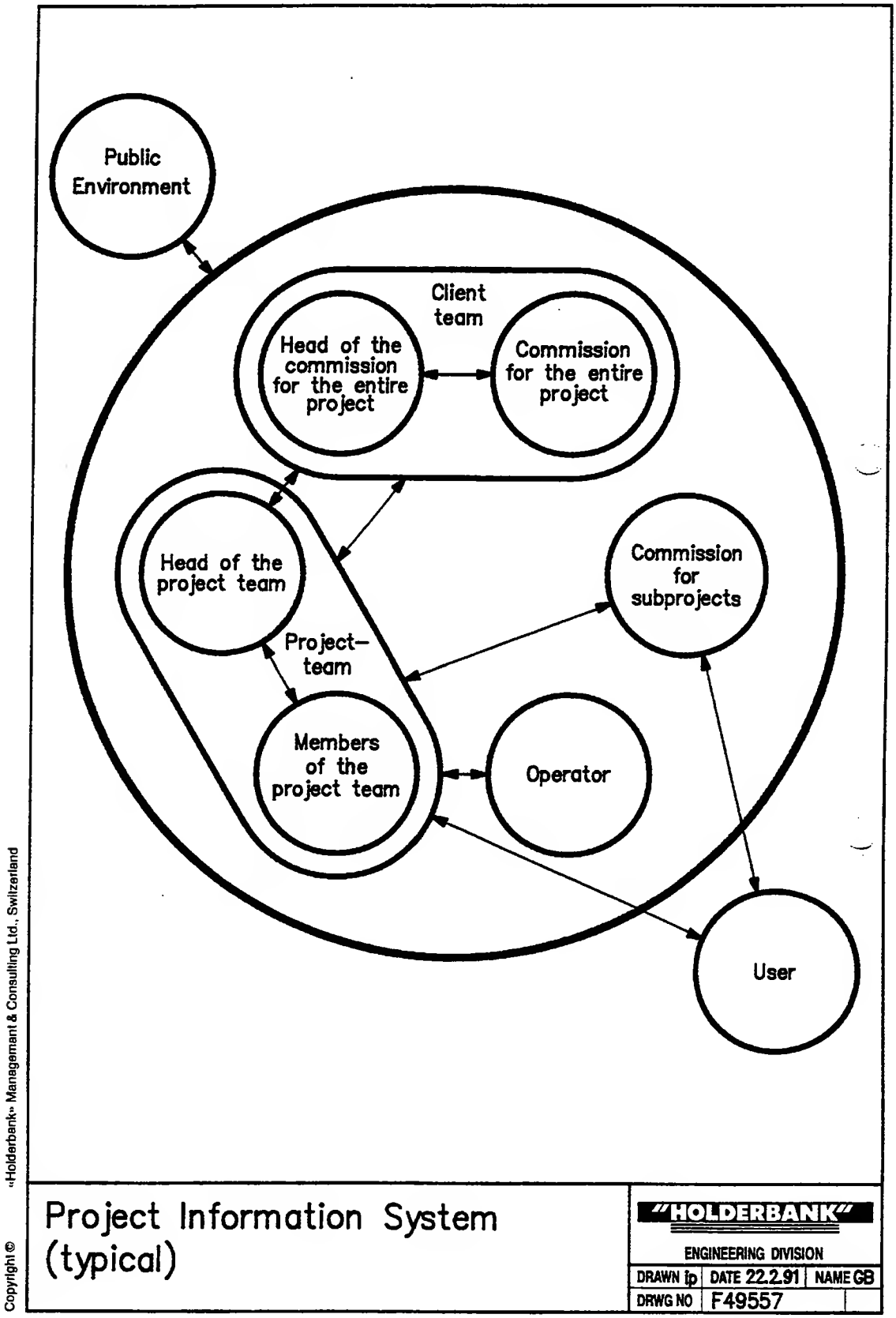
Figure 2/3 Project Information System (typical)



2.3 Institutional Aspects

2.3.1 Project Organization

The organization of a project specifies the sort and arrangement of working teams, control and decision making bodies as well as their tasks, competences and interdependences.

The institutional aspect of project engineering is characterized by comparing problem-guided working teams (partly or completely releasing of collaborators out of their original position within the company's organization depending on the importance and priority of a project) and their organizational institution. The task of such an institution is a systematic implementation of a project in accordance with their delegated competences, the provided means and relevant conditions. The composition of these working teams (institution) can change from project stage to project stage according to the respective requirements. After the project implementation is carried out, the problem-guided project organization is splitted up and its members return to their original positions within the company's organization.

The head of the project team is responsible for the technical, temporal and financial aspects and has to suggest suitable measures to be discussed and carried out in order to gain the set targets.

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3. PROJECT PROCEDURE

3.1 General

This part refers to the procedure of project execution (Fig. 3/1) for any major capital investment projects starting with the prefeasibility studies and leading through to the audit.

PRE-INVESTMENT PHASE

Identification of Potential Projects

Corporate Planning, Plant Masterplan } Closely linked planning activities which enables the company to identify future needs, financial investment and other aspects for maintaining optimal plant operation.

Initiating of Specific Projects

Stage 1: Feasibility Studies

Stage 2: Basic and Preparatory Engineering

Stage 3: Procurement

INVESTMENT PHASE

Execution of Project

Stage 4: Final Engineering

Stage 5: Supervision and Control of Project Execution

Stage 6: Plant Organization, Personnel

Stage 7: Start-Up and Commissioning

OPERATIONAL PHASE

Operation of the Equipment

Stage 8: Audit

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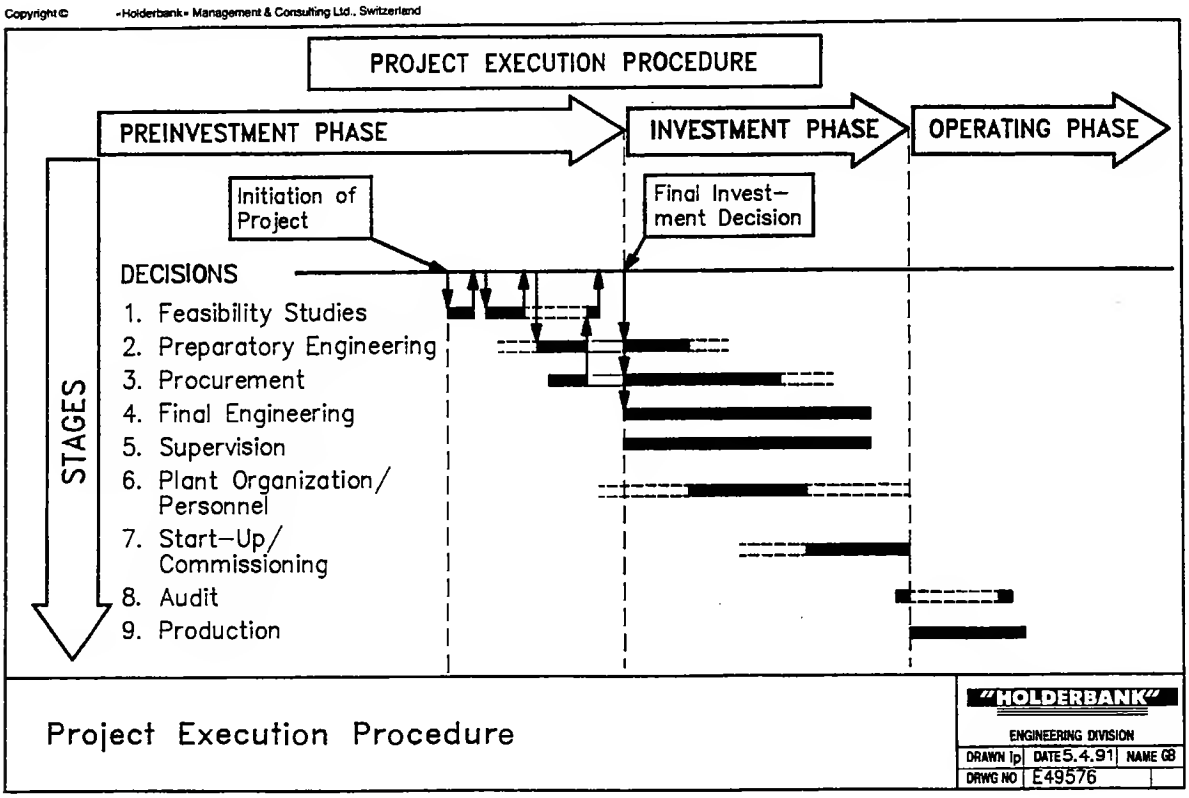
Figure 3/1 Project Execution Procedure



3.2 Stage 1: Feasibility Studies

The first step to be taken in order to make an investment decision for a well defined project is to establish a technical, economic, legal and often also political base.

Once a more detailed evaluation of a specific investment opportunity (potential project) is justified, the company will have to establish a temporary organizational structure and budget for this purpose. In case where the tasks exceed the capacity and expertise of the company, external consultants and experts should be called in to render assistance.

This project organization, preferably located outside the operating units of the company, has to perform all activities within well defined cost-time- and quality limits to facilitate the necessary major decision by the company which might eventually lead to the final investment decision.

The elaboration of a techno-economic feasibility study is rather costly and time consuming. Therefore, before assigning funds for such a study, a preliminary assessment of the potential project is often made in a prefeasibility study which shall show whether or not it is justified to proceed with the detailed feasibility study. The prefeasibility study is a short version of the feasibility study.

A feasibility study shall elaborate on and evaluate alternative solutions which ban be applied under the prevailing conditions of the projects.

The activities of such a feasibility study concentrate on the following main topics:

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Cement market :	review and evaluation of characteristics and trends
Environment :	impact on project of socio-economic environment
Raw materials :	reserves, raw maxis, recultivation strategy
Plant location :	considering local conditions, availability of labour and infrastructure, land acquisition, fiscal and legal regulation
Process and technical plant concept :	process, mechanical/electrical equipment, utilities, civil works, operation and maintenance
Infrastructure and housing facilities :	transport, power, water, housing facilities
Organization and staff requirements :	organizational structure, manpower
Investment cost :	fixed investment and pre-operation capital expenditures
Production expenses :	fixed and variable production expenses, personnel expenses
Implementation time schedule :	activity schedule, target dates, interfaces, completion
Financial projections and project profitability :	investigation of the economic feasibility of the project, determination of funds required to finance the project, determination of project profitability

The duration of a feasibility study depends mainly on the time required for raw material investigations.

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3.3 Stage 2: Basic and Preparatory Engineering

According to Fig. 3/2 and 3/3 the stage of basic and preparatory engineering is located within the liquid phase of the project progress, i.e. the remaining solutions, preselected and based on the feasibility study should be investigated carefully in order to reduce the number of alternatives and finally to determine the project ideas in such detail as required to produce drawings, schemes, flow sheets and specifications.

The purpose of the basic and preparatory engineering is to complete the engineering work as for as required for calling for tenders based on comprehensive and detailed tender documents (including specifications) for all supplies and services, broken down in accordance with the contracting plan. In particular the specifications have to be prepared as detailed as possible in order to obtain comparable offers.

Furthermore, lots of vagueness which demands for further inquiries can be avoided by clearly defined tender documents and specifications.

In a well-managed project no subsequent changes and revisions should occur, since these are time-consuming and can cost lot of money, especially after the procurement.

In respect of contract negotiations, the work-packages should be identified and responsibilities assigned for their execution including liability, guarantees and insurance.

Contract proposals can be prepared and a preselection of suppliers and contractors can already be made.

Concerning cost estimate(s) it has to be mentioned that establishment of detailed and final investment cost estimates which, after approval, may be used as the binding investment budget for project execution should be carried out at this stage of the project.

Accuracy of cost estimate(s) generally still range between 95% and 100%.

Project organization and administrative structures must now be established, methods and procedures for cost-, time- and quality control selected, the activities of all parties involved coordinated, corrective measures initiated if deviations from set targets occur and regular project progress reports prepared.

All typical organization pattern during the planning period is given on figure 3/4.

Obtaining outstanding permissions from national and local authorities as well as outlining of project financing requirements and scheme are further tasks to be carried out.

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Figure 3/2 Degree of freedom versus project realization progress

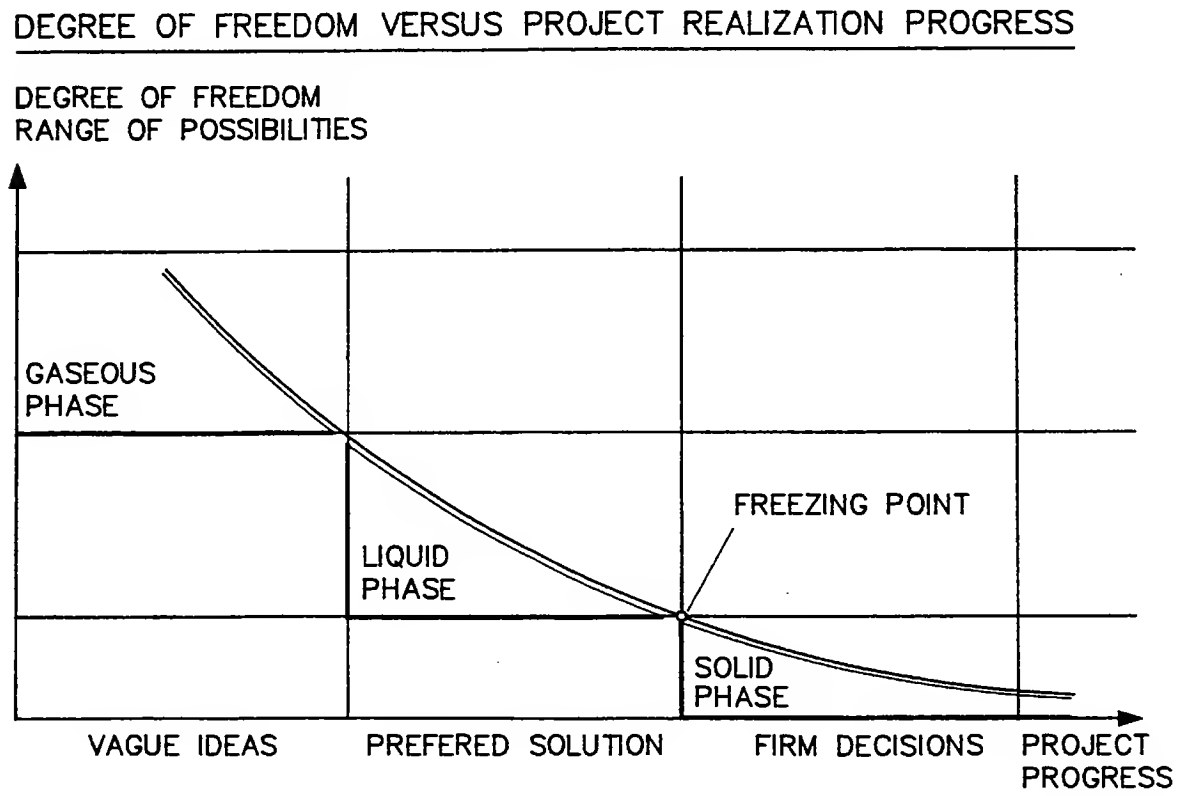

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Figure 3/3 Constitutions phases and activities of project planning and execution

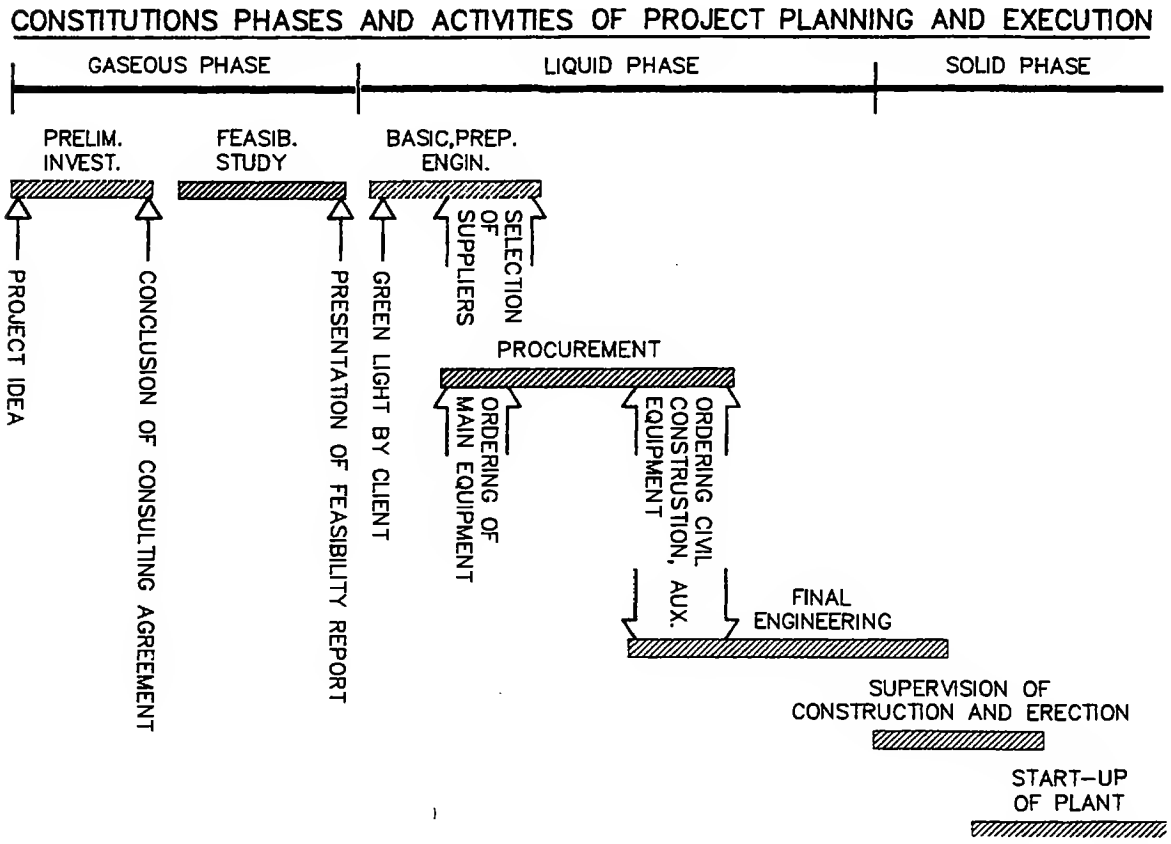

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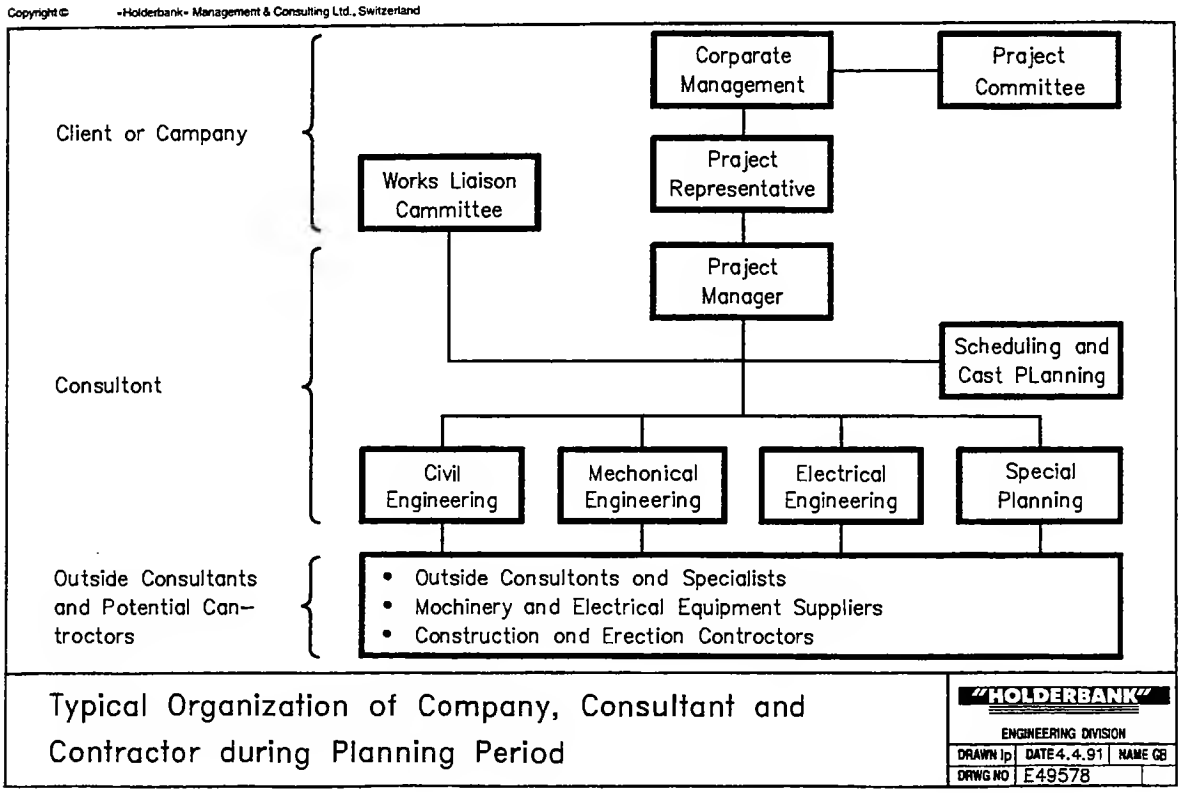
Figure 3/4 Typical organization of company, consultant and contractor during planning period



3.4 Stage 3: Procurement

The purpose of the procurement is the achievement of readiness for placing orders.

First the purchasing procedure has to be initiated by issuing of the Tender Documents.

After receiving of the tenders the evaluation of the offers and accordingly the preparation of purchase recommendation has to be made.

Contract negotiations will follow and not to forget import licenses for equipment and permit for payments in foreign currency has to be obtained if necessary.

Adjustment of final investment cost estimate, will take place if necessary, after binding tender prices are available for part or total of supplies and services. Accordingly a rerun of the investment return and funding projection based on above final investment cost estimate and actual financing conditions have to be performed.

3.5 Stage 4: Final Engineering

During the final engineering all detailed engineering work should be completed in order to allow for timely manufacture and erection of equipment and installations including modifications and/or new civil works (if required).

Company involvement varies significantly with the established contracting plan, i.e. minor whenever a general contractor is applied (like turnkey) but substantial where the company does its own detail design and lets small contract packages.

The final engineering comprises tasks as follows:

- Securing approval of investment budget
- Securing authority to place orders
- Preparation of specifications and drawings (including final equipment arrangements) for the manufacturers equipment
- Preparation of detail drawings for construction/modification of civil work (if necessary), erection and installation
- Preparation of all documents and data schedules to coordinate above activities (including project manual)

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3.6 Stage 5: Supervision and Control of Project Execution

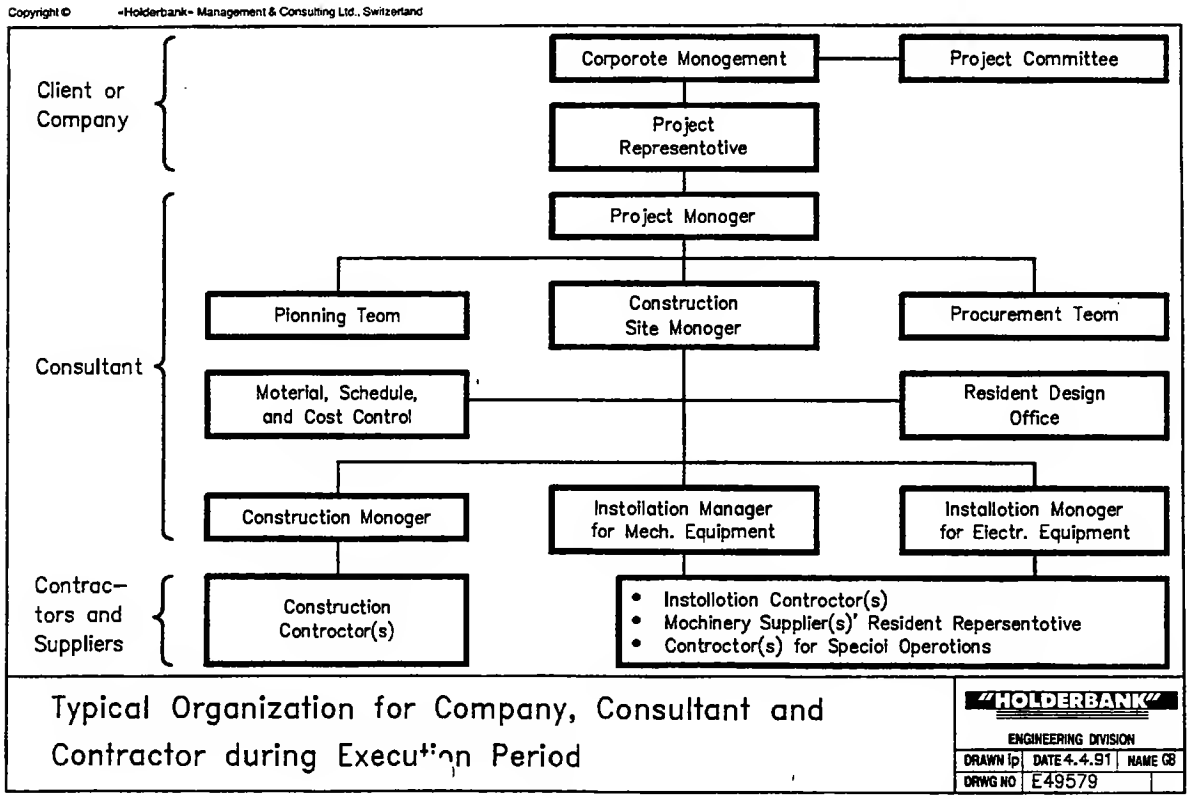
An overall project monitoring and control system for quick identification of problems and deviations from set targets to enable fast and adequate corrective actions must be implemented. Hence, a pertinent organization structure is required, comprising client, consulting and contractor (see Fig. 3/5)

3.6.1 Approval

All contractors final documents, data, specifications and drawings have to be reviewed and approved that they reflect the detailed technical concept as specified and indicated in the contractual documents. Further works must not be released prior to approval!

Checking and approving of drawings and calculations is an important and time-consuming task and it should for that reason go without saying that sufficient time must be allowed for it in any network program. As it is not rare that drawings or calculations must be refused, time should be allowed for a repetition of design work and another checking without jeopardizing the milestones of the project time schedule. In order to economize on time for this approval procedure, it is of utmost importance that all parties involved are really co-operating efficiently.

Figure 3/5 Typical organization for company, consultant and contractor during execution period



3.6.2 Budget and Progress Control

For various reasons the budget is one of the most important set-points for the project manager.

What budget control system ever is applied (there are various software programs on the market), it must inform the project manager about the following data:

- Original budget
- Commitments of the company in connection with this project up to date
- Work executed/deliveries received
- Payments up to date
- Expected/most probable final cost
- Revised budget, if applicable

It is essential to control not only the suppliers (should be clearly defined by the contracts) but also additional internal costs which can arise, e.g. by supplementary installation of ventilation or air-conditioning equipment, working hours of employees etc.

The delays between commitment, delivery and payment are a matter of company policy, respectively contract conditions which can normally not be influenced by the project management.

In order to keep in track with the most probable budget figure and the expected completion date it is indispensable to apply a commensurate control system, nowadays usually carried out by one out of various software programs which are on the market (e.g. Timeline, CAPM) (Fig. 3/6 and 3/7).

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Figure 3/6 Project progress

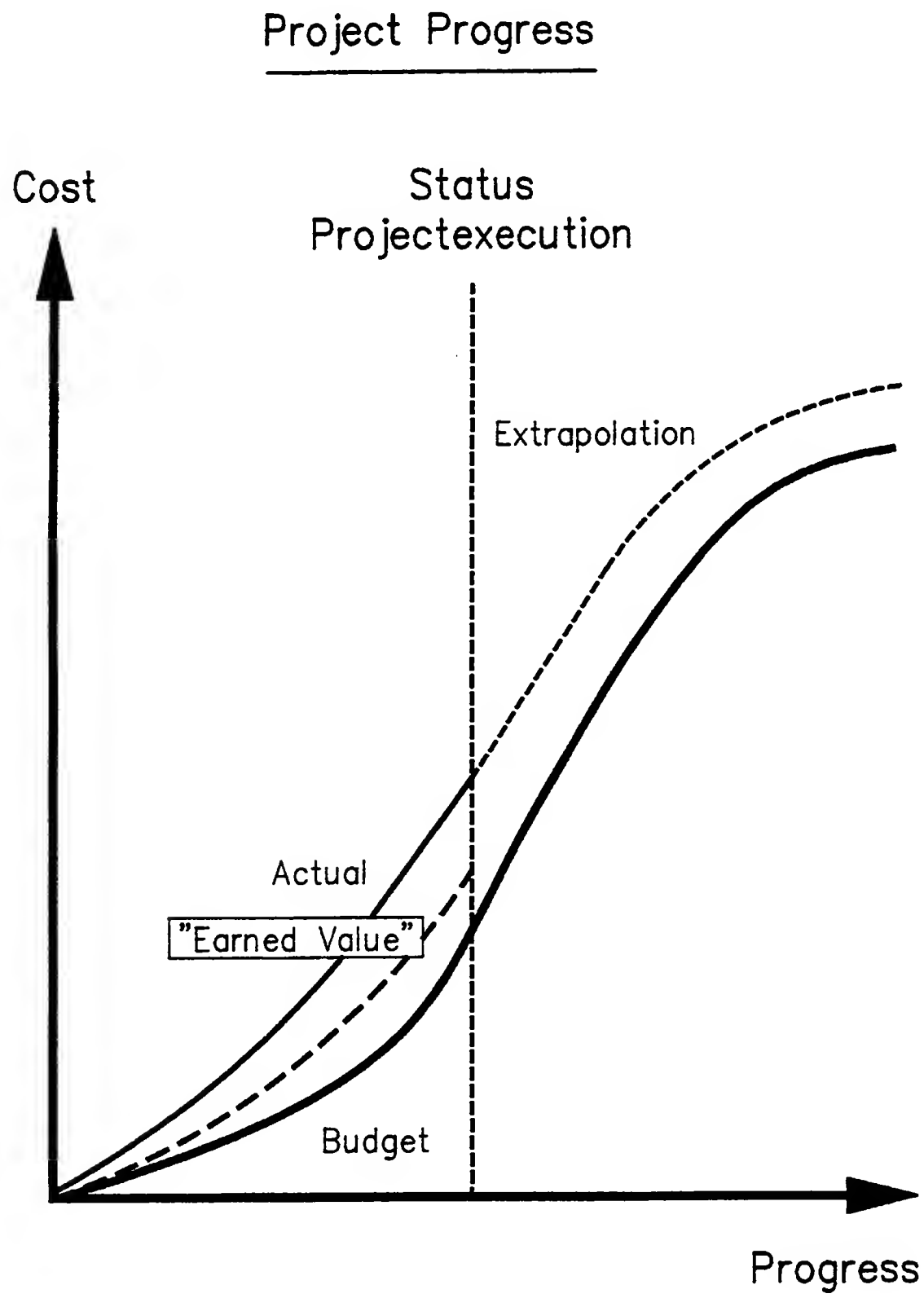
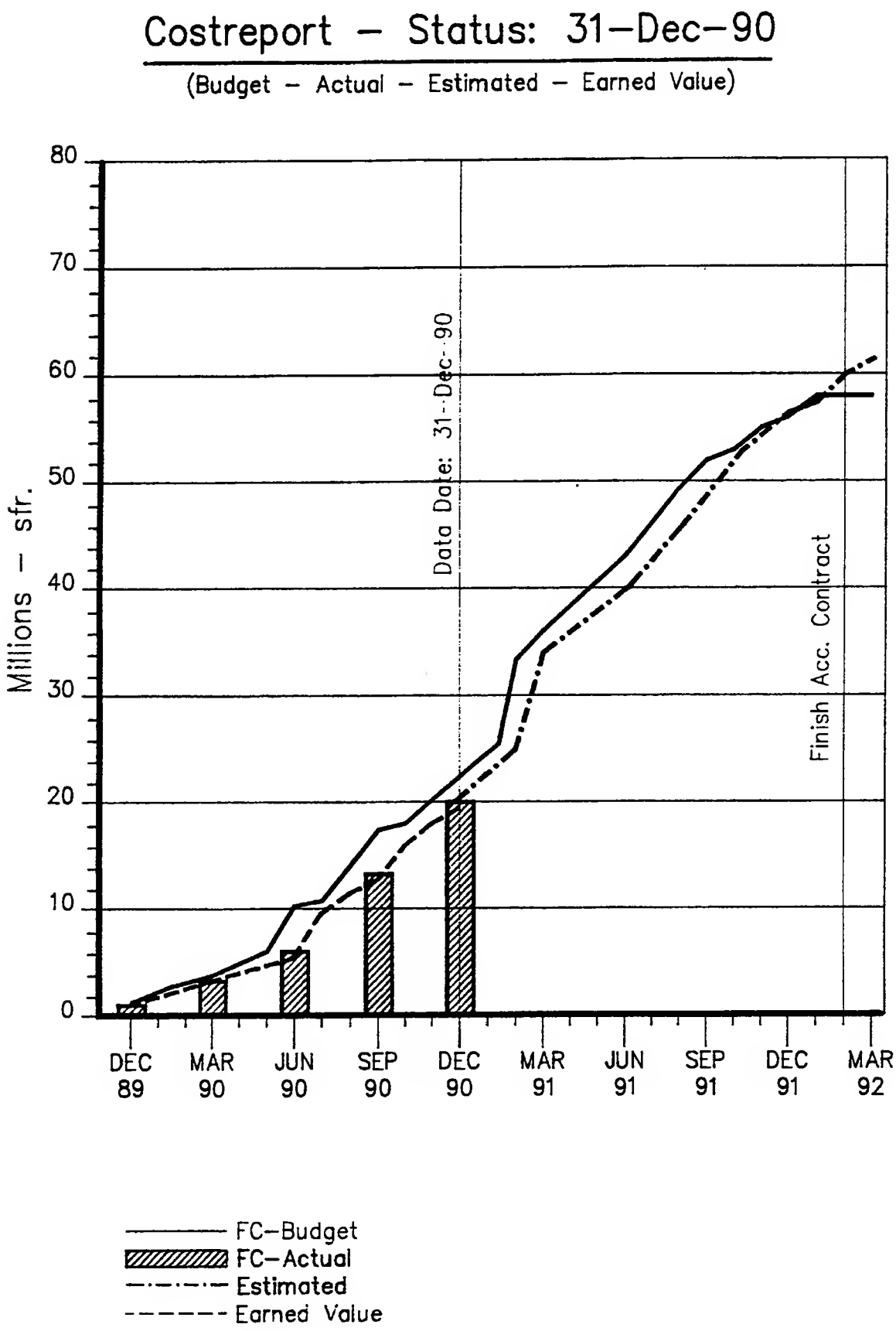


Figure 3/7 Cost report - Status: 31-Dec-90



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3.6.3 Quality Control

Quality (and quantity) control means in the very widest sense to assure a good value of the works and suppliers.

Quality control comprises activities like the following:

- Checking and approving of drawings, delivery specifications, installation plans, diagrams, etc.
- Inspections at manufacturer's shops in order to witness material tests, test-runs, etc.
- Checking of execution of all works (civil), mechanical erection, electrical installation, etc.) according to approved plans, approved specifications and the applicable standards
- Checking of all quantities used for civil construction work (quantity surveyor)
- Checking of progress of work at site, etc.

In order to really be able to perform this controlling function, it is necessary that any contract allows the project management to withhold payments if the quality of material or workmanship is not according to the satisfaction of the project management.

In this context, the delivery schedule and the payment schedule need mentioning, because they must be balanced in such a way that the project manager really has the possibility to manipulate the supplier/contractor by not releasing payments.

3.7 Stage 6: Plant Organization, Personnel

It is of importance to establish an adequate organization an personnel infrastructure (Fig. 3/8) already at an early date to operate and maintain the plant as well as assurance of raw materials and utilities for start-up and normal operation.

The main tasks are as follows:

- Design and implementation of organizational structures for plant operation to be ready at start-up
- Preparation of plant manual
- Ensuring availability of raw materials correctives, additives and consumables, spare parts
- Selection and training (if required) of personnel and introduction to project before start-up
- Providing operating and maintenance manuals and instructions

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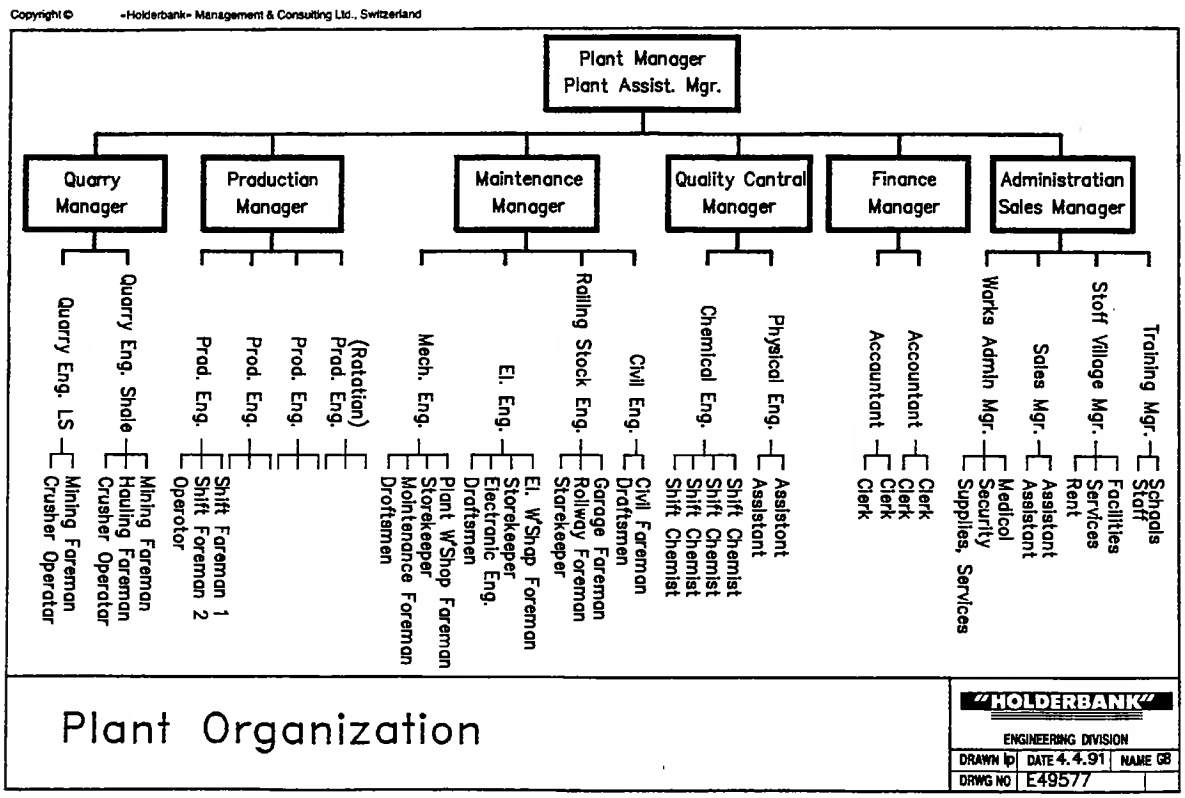
Figure 3/8 Plant organization



3.8 Stage 7: Start-up and Commissioning

Once the construction/modification of buildings and the erection and installation of equipment is approaching its completion, commissioning of equipment commences.

Before commissioning can start the whole equipment and installation has to be checked against completion and function, i.e. idle, part- and full-load test runs have to be carried out as well as repairs and adjustments if required.

Furthermore, a performance test has to be executed (conduct and supervision) and deficiency lists to be compiled as a part of taking-over certificates.

However, it has to be ensured that contractual obligations by suppliers are met with the objective to achieve a continuous routine operation at full rated output immediately after provisional taking-over of the respective plant section(s).

Issuance of pertinent certificates where appropriate and determination of penalties for unfulfilled performance guarantees shall take place.

A due documentation has to be adjusted or composed and issued.

3.9 Stage 8: Audit

The purpose of the audit is the evaluation of results achieved and recording the experience gained in order to refine cost estimates and project procedures for the future.

To audit project procedures and organization (best carried out before the project team breaks up) are tasks of an audit as well as to audit the technical concept and economic projections as soon as the relevant plant section(s) and equipment is in satisfactory continuous operation and the financial magnitude have stabilized.

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4. DIFFERENTIATION BETWEEN MINOR AND LARGE PROJECTS

Since the expenditure of a project can vary significantly a differentiation concerning project size has to be considered in order to dispose the commensurate project implementation procedure.

The most likely differentiation is to differentiate between projects which require a production interruption and projects where it is not needed to stop producing clinker, respectively cement.

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Minor Projects	Large Projects
Main Features	
Do not require a production stoppage Tasks concerning the infrastructure of a cement plant such as water supply system, lighting, heating, cooling etc. Can mostly be carried out by the company's personnel.	Require a production stoppage Tasks concerning certain sections of a production line such as the installation of a separator to an existing mill (incl.) electrical work and commissioning) or the erection of a waste heat recovery system etc. Co-operation with external suppliers (one or more)
People Involved	
The hierarchical plant organization does not change, it is only completed by the existence of a project coordinator (chosen out of the existing plant organization) in order to control the technical, temporal and financial project implementation but without authorities over the plant staff. First priority of the people involved is their daily works, whereas the respective energy optimization project is of second priority (with a preferred completion date, but can be postponed)	A project team can be composed with people out of the existing plant organization (and eventually completed by consultants, suppliers etc.) in regard to subordinate their daily work and to be able to concentrate on the respective energy optimization project as a full-time job of first priority. A project team can be composed out of consultants, suppliers, i.e. an external project team.
Minor Projects	**Large Projects**
Procedure	
Engineering Procurement of material (out of workshop, spare parts storage, local markets or oversee) Installation and commissioning Audit	Preparatory engineering Procurement of material (mainly from foreign suppliers) Final engineering Supervision and control of project execution Start-up and commissioning Audit

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Remarks	
It is important to mention the danger of getting bogged down or falling into oblivion in particular for this type of project since they are less urgent and don't have a determined completion date. Such projects are often carried out by the maintenance team.	Since this type of project requires a production interruption it is of utmost importance to have it well planned and coordinated in order to achieve the shortest possible stoppage. Such projects are often carried out by the maintenance team during a production stoppage for maintenance work. It would for that reason be advantageous to have member(s) of the maintenance department as part of the project team since they are used to carrying out such planning (incl. co-ordination of provision of auxiliary equipment such as ladders, cable reels, emergency lighting, lifting platforms, welding equipment etc.) in respect of avoiding misplanning and achieving a shortest possible production stoppage.

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"HOLDERBANK"

PROJECT MANAGEMENT

Brief Guide and Checklist for Capital Expenditure Projects for Replacement and Modifications of existing Plant Facilities

"Holderbank" Management and Consulting Ltd.
Engineering Switzerland

March 1996
HES/ZKZS/LEU

PROJECT MANAGEMENT

Key Issues

1. GENERAL

1.1 Definition of Project

- Technical idea / concept to be studied and or realized.

Main features:

- Project has to fulfill certain **requirements**, assure certain **performances**.
- Project has **defined start and end,** time required depends on internal and external constraints
- Project has a **limited** budget
- Project requires **special organization,** it is often disturbing factor in permanent organization

1.2 Initiation of Project

- Initiated project should be outflow from masterplan
- Initiation shall be done in written form:
  - setting of objectives
  - establish main tasks and responsibilities
  - name project manager and project team
  - outline stepwise procedure

2. STUDIES

- Set-up organization and procedures
- Review of requirements compared with present situation (same / higher / lower), all parties concerned to be involved
- Study of alternative solutions, at later stage time will not be available (literature, suppliers, HMC)

- Assessment of alternatives based on:
  - technical criteria
  - experience / references
  - economical aspects
- Decision to proceed with project

  -> introduction in investment list or presentation of investment proposal (possibly in Basic Engineering Phase)

3. BASIC ENGINEERING

- Adapt organization and procedures to project phase
- Establish contracting plan
  - taking into account complexity of project, own resources etc.
  - choice between:

    - detail engineering (several contracts)

    - packages (2 to 3 contracts)

    - turn-key (1 contract)

    Detail engineering is being preferred solution for "Holderbank".
- Establish time schedule with key events
- Preparation of Tender Document

  Note: - Tender Document shall only be issued once high probability of project realization is being established.

  Purpose: - Definition of technical criteria, scope of supply and commercial conditions

  - Safeguard interest of Client

  - Provide basis to Tenderers to prepare Tender

  - Provide basis to Client to evaluate Tenders

  Form: - Tender Document preferably issued in future Contract Form
- Preselection of Tenderers
  - at least 3 to 4 Tenderers
- Evaluation of Tenders

  Comparison tables to be prepared containing:
  - technical data

- scope of supply
- warranties
- experience / references
- commercial conditions

The evaluation criteria are laid down in the Tender Document.

Tables can be prepared during Tendering Period, do not await Evaluation Period.

- Preparation of investment request => Approval by Management to proceed with realization of project

4. DETAILED ENGINEERING / PROJECT EXECUTION

- Adjust project organization and procedures to requirements
- Contract negotiations
  - at least 2 potential Contractors to create competitive situation
  - technical aspects and warranties to be clarified first
  - price negotiations to be assured by small team and not carried out under time pressure, apply target price policy.
  - all contractual matters to be solved during negotiations and incorporated into Contract, in execution phase power of Client is being reduced.
- Establish tight cost and time control
- Check and approval of Contractor's drawings, to be done with necessary care, concerned parties to be engaged (production, maintenance)
- Workshop inspection
  - engagement of outside experts to be considered (global purchasing of Contractors)
- Site supervision
  - adequate site supervision to be assured, also required to coordinate site activities with plant operations
  - establish and update list of deficiencies
- Start-up and Commissioning
  - involvement of personnel from operations and maintenance
  - arrange foreseen performance tests
- Project audit
  - to be carried out about 12 months after commissioning by project team and operations / maintenance

- review of following:
  - project execution
  - performance, recalculate savings, IRR and pay-back period and compare with investment request
  - outline of conclusions for present and future projects

Annex 1: Investment Cost Estimate
Annex 2: Investment Proposal / Request
Annex 3: Tender Document / Table of Content

INVESTMENT COST ESTIMATE

Project : .....

Phase : .....

Description	**Cost Mio .....**	**Cost Mio .....**
1. Disassembly and Demolishing		
Disassembly M&E-Equipment	.....	
Removal steel structure	.....	
Demolishing civil structures	.....	
Disposal equipment, steel and civil structures	.....	
Total 1 : Disassembly and Demolishing	-	.....
2. Mechanical Equipment		
Machine No. 1 :	.....	
Machine No. 2 :	.....	
Machine No. 3 :	.....	
Machine No. 4 :	.....	
Machine No. 5 :	.....	
Total Mechanical Equipment (Basis FCA)	.....	
Transport and Insurance	.....	
Taxes and Duties	.....	
Erection	.....	
Commissioning	.....	
Total 2 : Mechanical Equipment		.....
3. Electrical Equipment		
Medium Voltage Distribution & Transformers	.....	
Low Voltage Distribution and Motor Control Centers	.....	
Plant Automation	.....	
Instrumentation	.....	
Drives and Motors	.....	
Cabling, Grounding, Lightning Protection	.....	

Description	Cost Mio	Cost Mio .....
Total Electrical Equipment (Basis FCA)	.....	
Transport and Insurance	.....	
Taxes and Duties	.....	
Erection	.....	
Commissioning	.....	
Total 3 : Electrical Equipment	-	.....
4. Structural Steel and Civil Works		
Structure 1 :.....	.....	
- Excavation	.....	
- Concrete Works	.....	
- Structural Steel Works	.....	
- Various (blockworks, windows, etc.)	.....	
Structure 2	.....	
- Excavation	.....	
- Concrete Works	.....	
- Structural Steel Works	.....	
- Various (blockworks, windows, etc.)	.....	
Structure 3 :	.....	
Excavation	.....	
- Concrete Works	.....	
- Structural Steel Works	.....	
- Various (blockworks, windows, etc.)	.....	
Total 4 : Structural Steel and Civil Works	-	.....
5. Engineering		
Mechanical Engineering	.....	
Electrical Engineering	.....	
Civil Engineering	.....	
Total 5 : Engineering	-	.....

Description	Cost Mio	Cost Mio .....
6. Various Contingencies	.....	
Total 6 : Various	-	.....
GRAND TOTAL 1-6	-	.....

Above cost estimate does not include VAT and cost for spare parts.

Date: ..... Name and Signature: .....

Investment Cost Estimate

Typical Cost Structure in %

1.	Mechanical Equipment	100 %
1.1	Transport and Insurance	5 % of 1.
1.2	Erection and Commissioning	30 % of 1.
2.	Electrical Equipment	30 % of 1.
2.1	Transport and Insurance	5 % of 2.
2.2	Erection and Commissioning	30 - 50 % of 2.

Accuracy of Investment Cost Estimate

Phase:	Studies	± 20 %
	Basic Engineering	± 10 %
	Detailed Engineering	± 5 %

INVESTMENT PROPOSAL / REQUEST

Project : .....

1. PRESENT SITUATION

- Description of problems
- Loss of production
- Extra cost involved (production, maintenance)

2. ALTERNATIVES INVESTIGATED

- Description of alternatives
- Advantages / Disadvantages
- Selection of alternative(s)

3. SOLUTION PROPOSED

- Detailed description
- Proposed suppliers
- References

4. INVESTMENT COST ESTIMATE

- Basis of cost estimate
- Cost estimate (budget)

5. ECONOMICAL EVALUATION

- Cost difference (savings) between present situation and proposed solution
- IRR-Calculation
- Pay-back-calculation (usually not more than 2 to 3 years)

6. TIME SCHEDULE

- Duration of project execution
- Critical path

Annexes : Investment cost estimate

Date: .....

Name and Signature: .....

Approved:

Date: .....

Name and Signature: .....

Comments:

TENDER DOCUMENT

TABLE OF CONTENT

PART A : Tendering Conditions

- **General Information**
  - Purpose
  - Client
  - Brief description of project
  - Scope of tender
  - Proposed time schedule
- **Conditions of Tendering**
  - Commercial part
    - Lumpsum price in ... currency
    - Price breakdown and weights
    - Time schedule
    - Comments on supply contract proposal
  - Technical part
    - Technical specifications
    - Technical specification sheets
    - Schematas and drawings
    - Reference list
    - Warranties
    - Alternative proposals
  - Date of submission of tenders

PART B : Contract Proposal

- **Supply Contract Proposal**
- **Annex 1 : General Information**
  - Information on existing equipment related to project
  - Process data to be observed for proposed project
  - Material data (raw materials, additives, correctives, cement compositions, etc.)
- **Annex 2 : General Requirements on Mechanical Equipment**
  - Standards to be applied
  - Standardization (gears, clutches, etc.)
  - Requirements and design criteria on individual equipment (e.g. filters, conveyors)
  - Painting and colour codes
- **Annex 3 : General Requirements on Electrical Equipment**
  - Standards to be applied
  - Voltage levels
  - Standardization (instruments, safety devices etc.)
  - Sub-control systems
  - Drives
- **Annex 4 : Technical Specifications**
  - Supplier's specifications (from successful Tenderer)
  - Technical specification sheets
- **Annex 5 : Services**
  - Mechanical Engineering
  - Electrical Engineering
  - Civil Engineering
- **Annex 6 : Time Schedule**
  - Detailed time schedule

- **Annex 7 : Price Breakdown**
  - Individual prices and weights (from successful Tenderer)
- **Annex 8 : Performance Warranties and Tests**
  - Procedures for measurements of warranties and conducting of performance tests
- **Annex 9 : Workshop Inspections**
  - Tests to be conducted by contractor at individual manufacturing steps
  - Tests to be carried out by Client / Engineer at individual manufacturing steps
- **Annex 10 : Drawings**
  - Flowsheet
  - Arrangement drawings

SUPPLY CONTRACT

Key Conditions

1. PRICE

- Fixed lumpsum price, preferably in local currency

2. PAYMENT CONDITIONS

- 20 (30) % as down payment, against advance payment bond for amounts above USD 50'000.--
- 50 (60) % at delivery
- 10 (20) % at provisional acceptance against warranty bond

3. TIME SCHEDULE

- Overall time schedule
- Special schedule (shut down period, etc.)

4. WARRANTY PERIOD

- 12 (24) months starting from provisional acceptance

5. WARRANTIES

- Delivery time
- Overall time for project execution (from coming into force up to provisional acceptance)
- Shut-down time
- Performance (capacities, clean gas content of filters)
- Function and quantity
- Lifetime of special parts (e.g. gear boxes up to 5 years)
- Lifetime of wear parts (e.g. filter bags up to 2 years)

6. PENALTIES

- Penalties due in case of non fulfillment of above warranties except for clean gas content, which has to be achieved.
  - Penalties in % of lumpsum price for delays and non fulfillment of performances
  - Replacement (partial) of equipment not reaching lifetime

7. INSURANCE

- Transport insurance by Contractor
- Third party liability insurance by Contractor
- Erection Insurance / All Risk Insurance by Contractor or Client

8. BONDS

- Bonds to be issued by a first class bank, occasionally Concern Guarantees are being accepted.

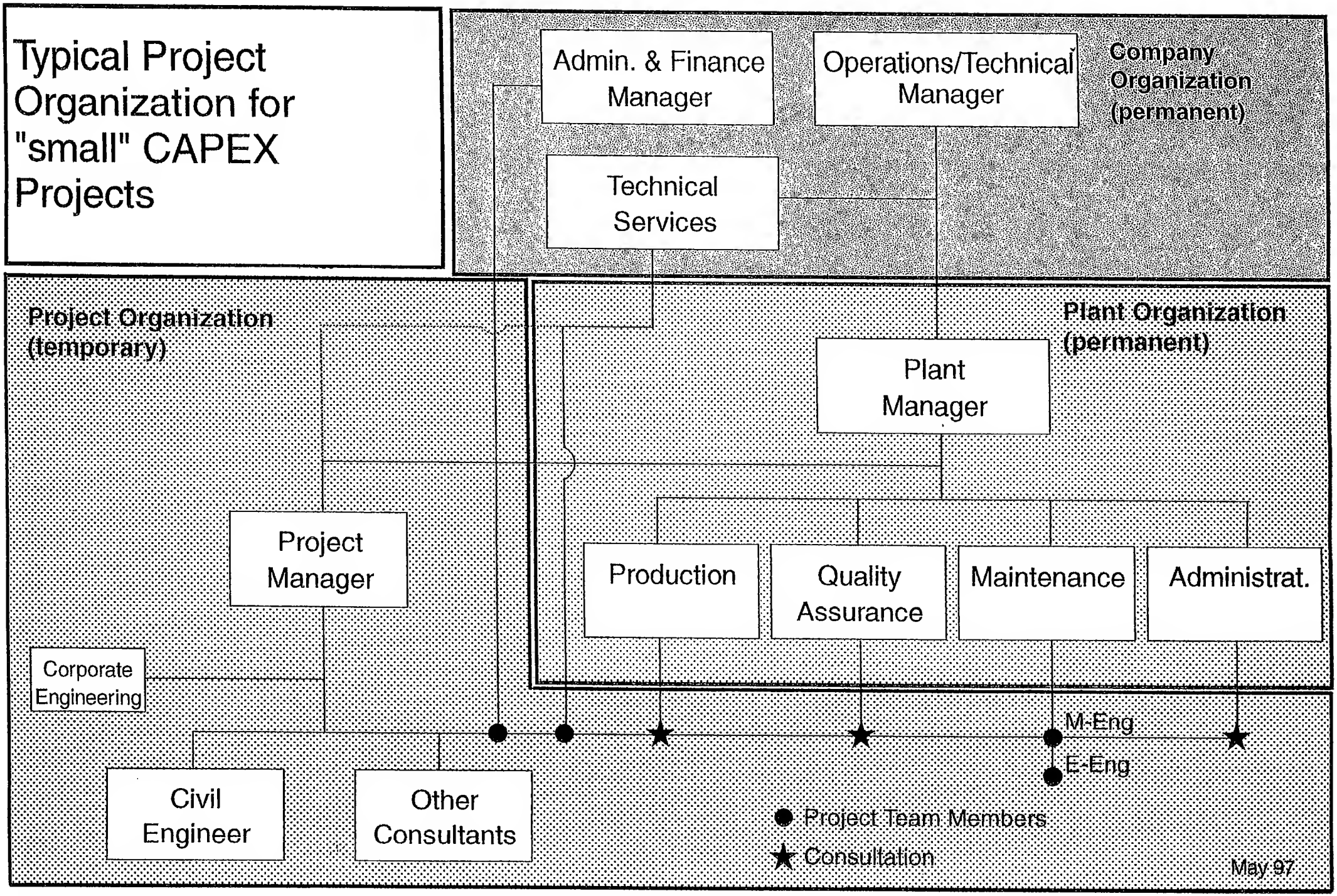
9. MODIFICATION OF CONTRACT

- Modification to be agreed in writing,

10. COMING INTO FORCE

- Conditions to be clearly defined.

Typical Project Organization for "small" CAPEX Projects



Chapter 11

Principles and Guidelines of optimized investment concept

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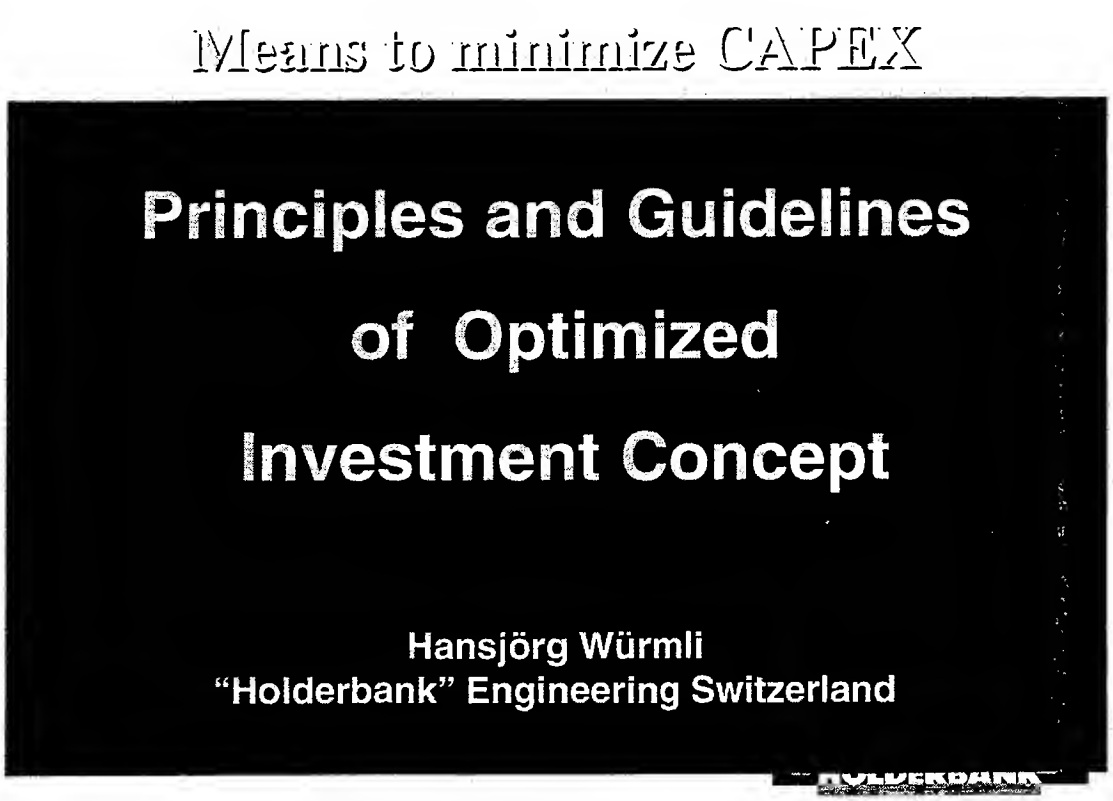
Optimized Capital Expenditure Projects

Slide 1



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Slide 2

Means to minimize CAPEX

Content

- Introduction
- Design philosophy and guidelines
- Application in design and engineering
- Review meeting
- Conclusion

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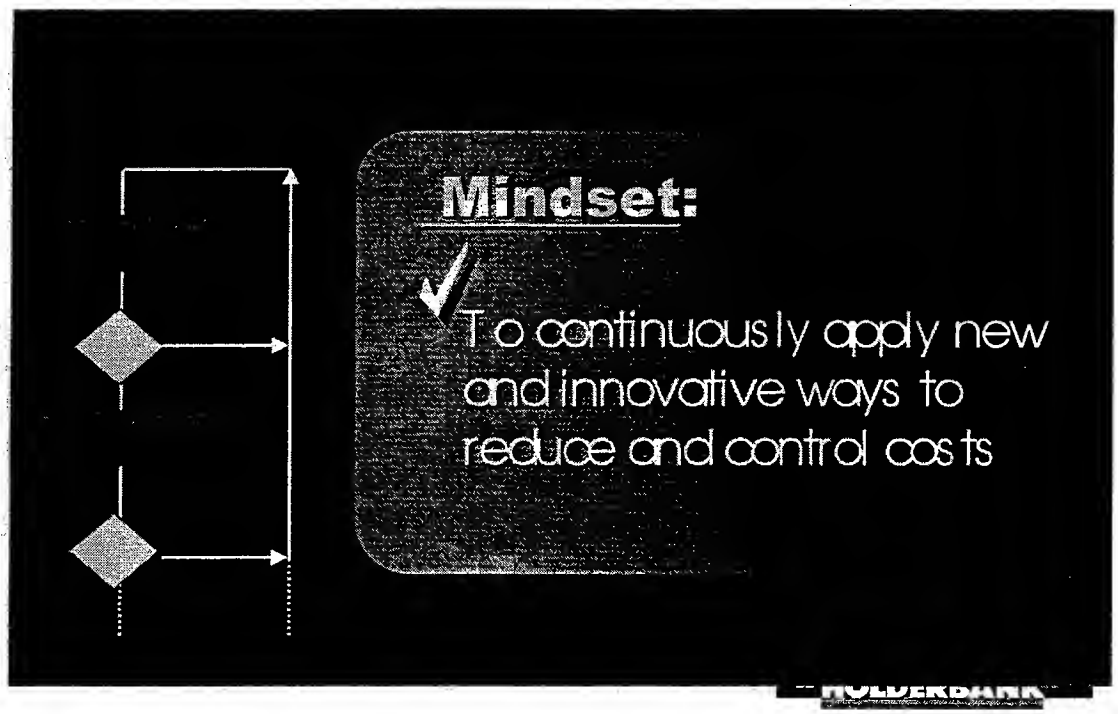
Slide 3

Means to minimize CAPEX



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Means to minimize CAPEX

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Slide 5

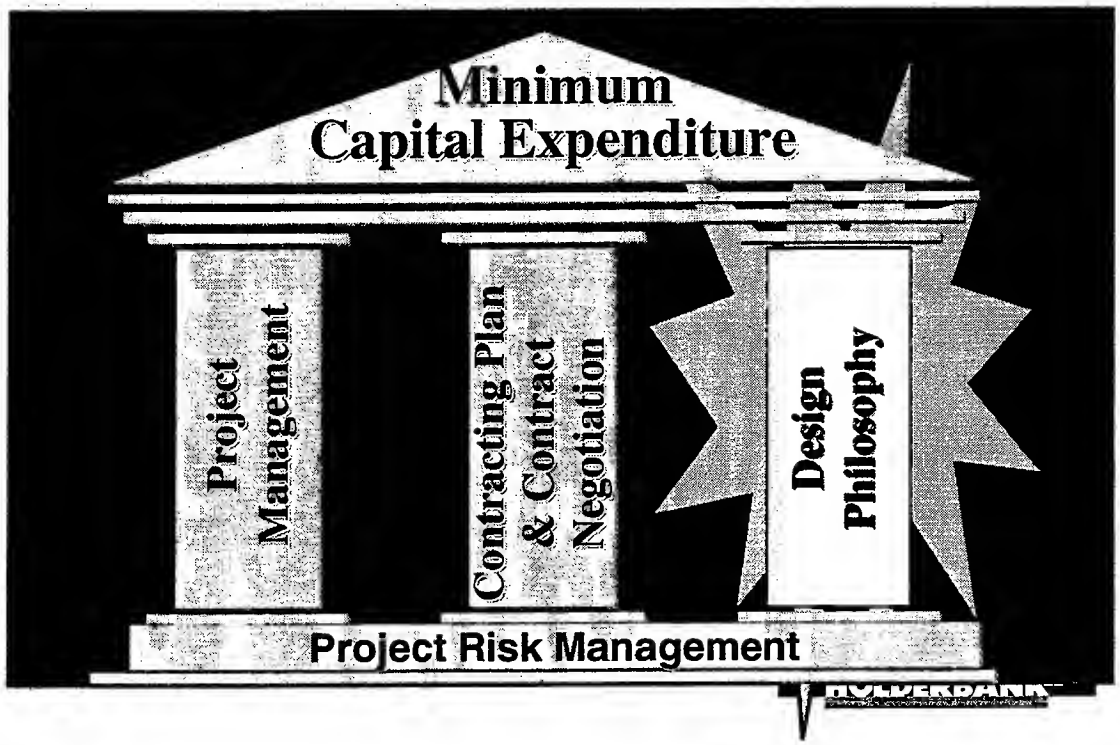
Means to minimize CAPEX

ptimize nvestment :

- Is a design and engineering philosophy
- Is aiming at lowest initial investment at acceptable operating cost.

Slide 6

Means to Minimize CAPEX



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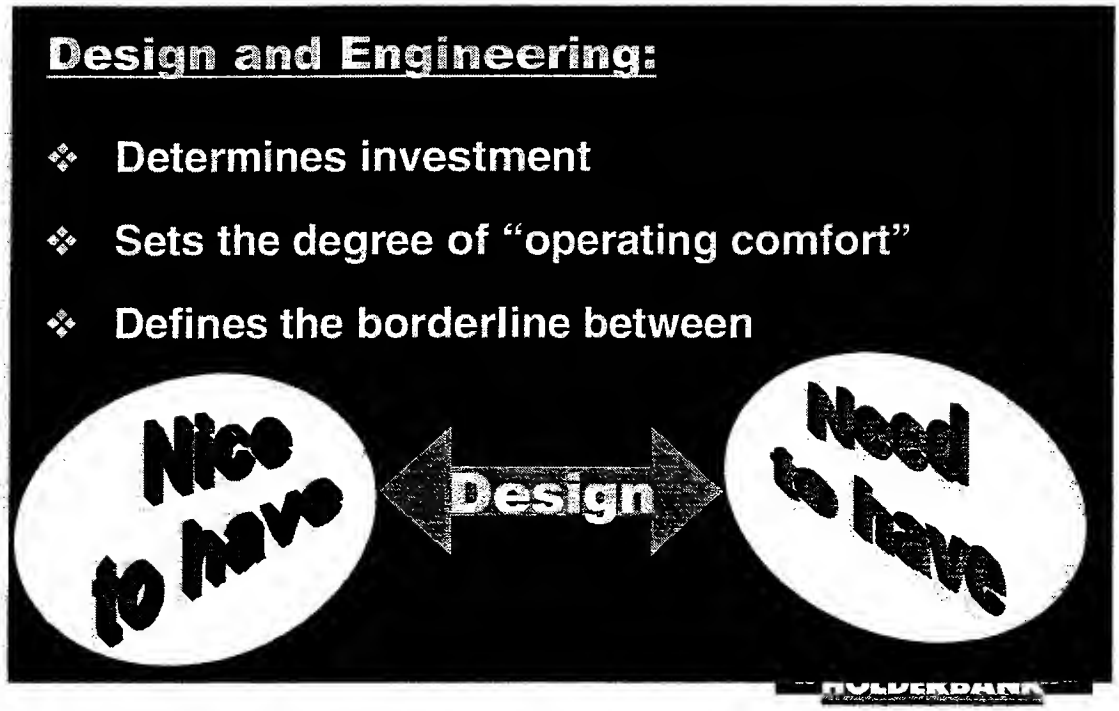
Design Philosophy



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Slide 8

Design Philosophy

Key Influencing Factors:

- Sizing and dimensioning of equipment, buildings and structures
- Arrangement of equipment
- Overall plant layout.

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Design Philosophy

Principles of Economic Design:

- Capacities of equipment within minimal practical limit
- Capacities of storages kept at minimum
- No standby equipment
- Step-wise project implementation
- No "nice to have", only "need to have"
- Buildings kept at minimum
- Value engineering
- Outsourcing
- Rent rather than buy
- (Use of second hand equipment)

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Design Philosophy

Plants built under the optimized Investment Concept are:

- Efficient
- Highly Reliable
- With State-of-the-Art Technology
- With Advanced Emission Control Systems
- AND PRODUCE HIGH QUALITY PRODUCTS.

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Design Philosophy

Economic Design means:

- Taking higher operational risks
- Accepting somewhat higher operating cost
- From "just in case" to "just in time"
  - Need for preventive maintenance
  - Need for better training of operating personnel
  - Need for contingency procedures
- Expect more difficulties during start-up of new installation.

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Design Philosophy

Economic Design does NOT mean:

- Taking uncalculated operational risks
- Cheap equipment
- Compromise on safety
- Compromise on product quality
- Compromise on environmental performance.

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Application of Economic Design

- Application in project concept and design from the beginning
- Design Reviews.

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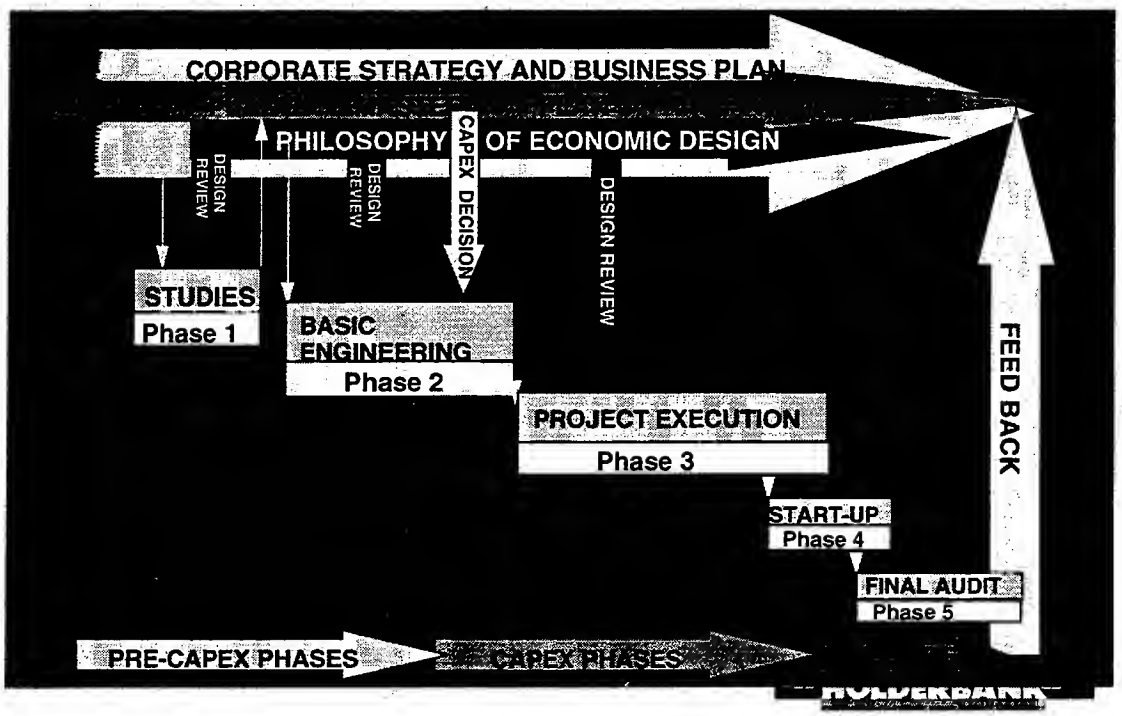

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Checkpoint

To ensure, that minimum investment requirements are followed in every major investment.

Plans and specifications need to be critically reviewed!

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- Critical assessment of proposed concept and technical solutions in consideration of minimal investment aspects.
- Consideration of alternatives resulting in reduced investment cost
- Assessment of consequences and risks
- List actions and proposal for further consideration by the project responsibles.

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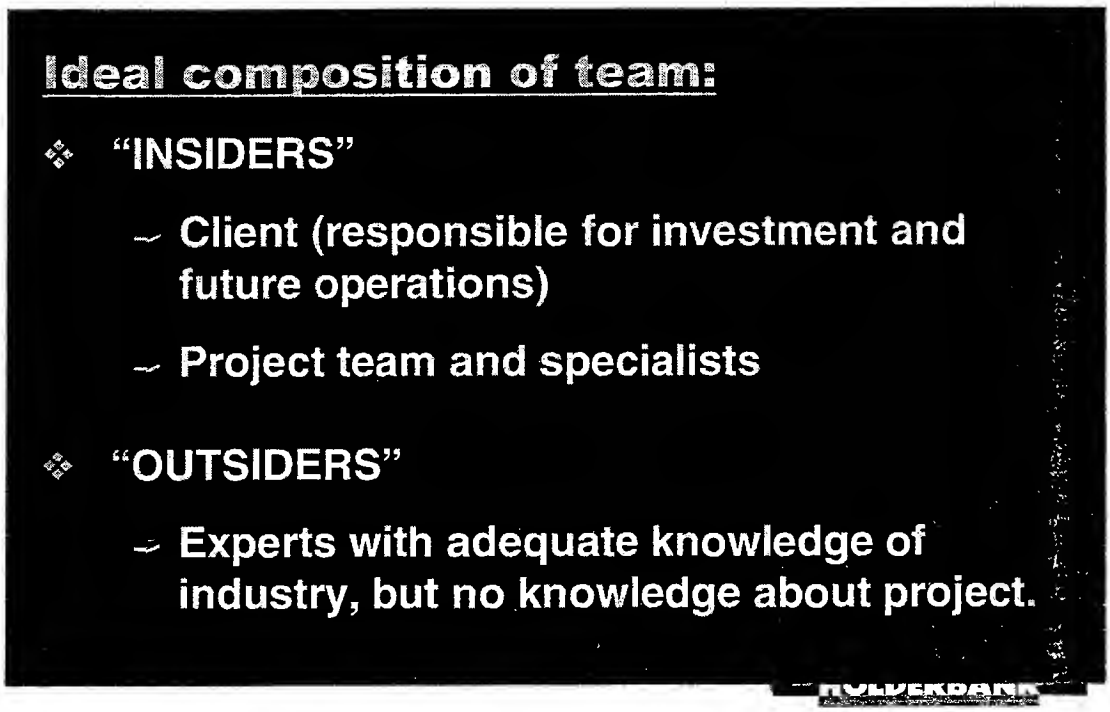

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What:	Who:
❖ Presentation of project General, Target, Market, Environment	Client
❖ Detailed presentation of individual section	Project Manager
– Analyze for conformity with minimal investment targets	
– Questioning of proposed concept	Group
– Alternative proposals.	

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Action	Consequences	Risk
5) No auxiliary furnace for raw mill, as expected use is less than 10% of time.	At feed moisture >9% over extended period (>3 days), nominal kiln output my not be maintained. *Saving: USD 170 k*	Minimal loss of production; < 1%
6) Kiln feed bin: delete stand-by dosing device. Provide only outlet with blind flange.	Kiln must be stopped when dosing device fails. *Saving: USD 18 k*	Acceptable, second dosing system can be added later.
7) Delete roof and OHT crane over raw mill.	hire mobile crane for maintenance on SE/BE; provide sufficient hard standing area. *Savings: USD 425 k*	Non availability of mobile crane on short notice may lead to kiln stoppage.

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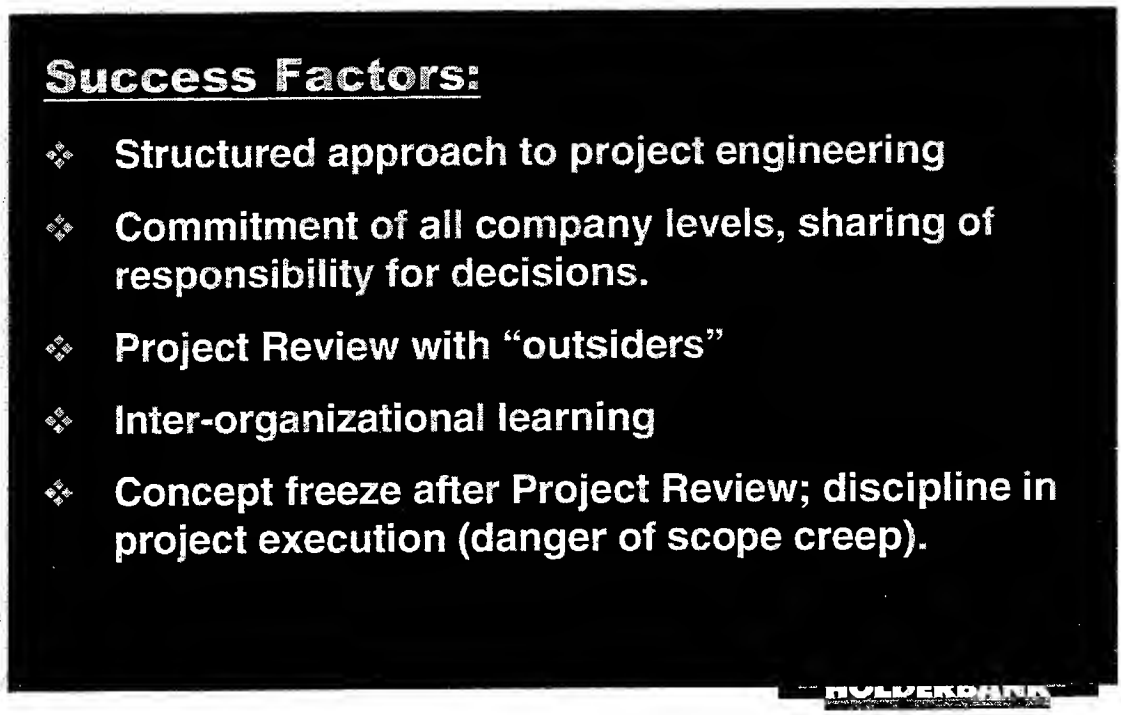

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Concept of Optimized Investment Cost has proven to be a powerful tool in reducing assets for new installations

- It can successfully be applied to <u>every project</u>, provided:
  - it is fully supported by management and all staff involved,
  - set targets are strictly adhered to during entire design and execution phase,
  - all personnel involved accepts strategy as a challenge which, when applied in an innovative, target oriented approach, will lead to cost effective solutions.

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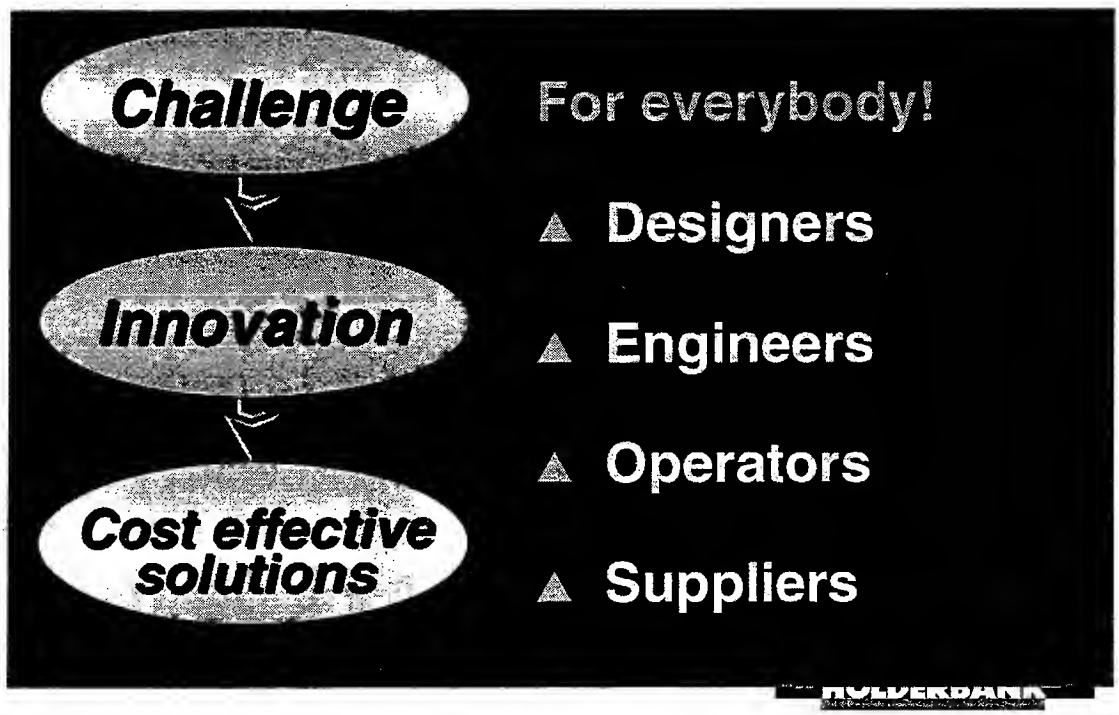
Slide 22



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CAPEX Projects

General Project Procedures

-

Procedures to Effective and Efficient Capital Expenditures

"Holderbank" Engineering Switzerland
Cement Seminar 2000

"HOLDERBANK"

Procedures for Effective and Efficient Capital Expenditures

= Increase of fixed asset base
~ Investments in Property, Plant and Equipment

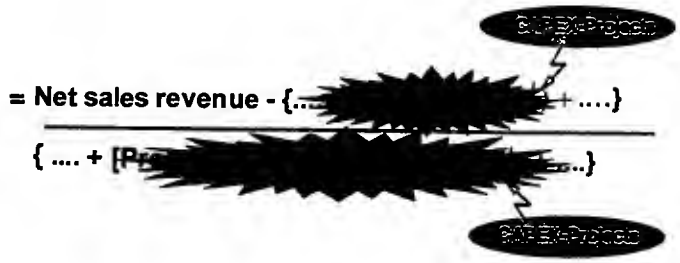
1. Replacement
2. Rationalization
3. Expansion, Diversification

4. Product Quality
5. Social & Safety
6. Environment

"HOLDERBANK"

Impact of CAPEX on RONOA

Return On Net Operating Assets (↑) RONOA	Net sales revenue (↑) - Cost (↓) / Net Operating Assets (↓)	= EBIT / NOA



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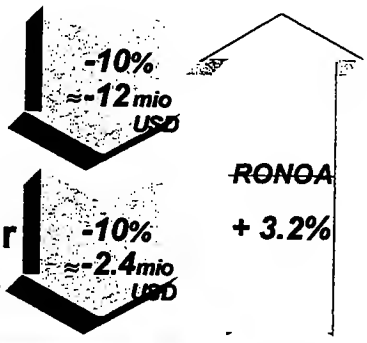
Impact of Asset Reduction on RONOA

❖Fixed assets value

❖Fixed operating cost per year

⇨*interest (10%):* *1.2 mio USD*

⇨*depreciation (10 years):* *1.2 mio USD*



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Impact of Asset Reduction on RONOA

- ❖ Substantial impact of investment cost on RONOA
- ❖ Investment cost are determined by ourselves
- ❖ Only <u>one</u> opportunity to influence investment cost

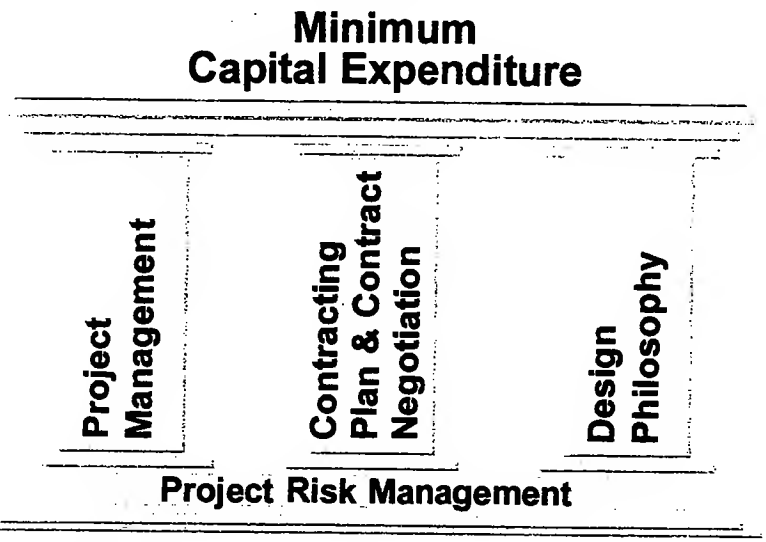
➡ ***Cost reduction is imperative for each investment in property, plant and equipment***

The "Holderbank" Group

Financial Key Figures

		1999	1998	1997
Net sales	*mio. CHF*	12194	11268	11265
Operating profit (EBIT)	*mio. CHF*	1706	1567	1435
Net Operating Assets (NOA)	*mio. CHF*	13648	12680	12173
Investments in Property, Plant and Equipment - Replacement - Expansion	*mio. CHF*	 480 631	 496 471	 577 823
Depreciation	*mio. CHF*	1151	1124	1143

Means to Minimize CAPEX



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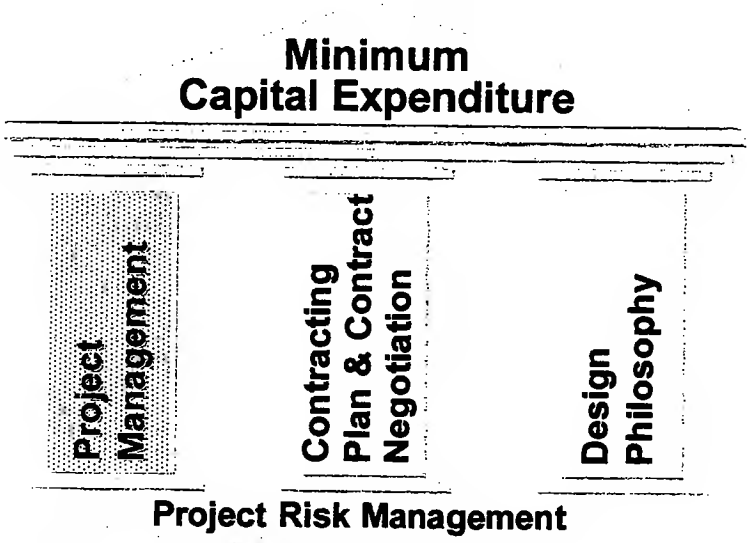
Means to Minimize CAPEX



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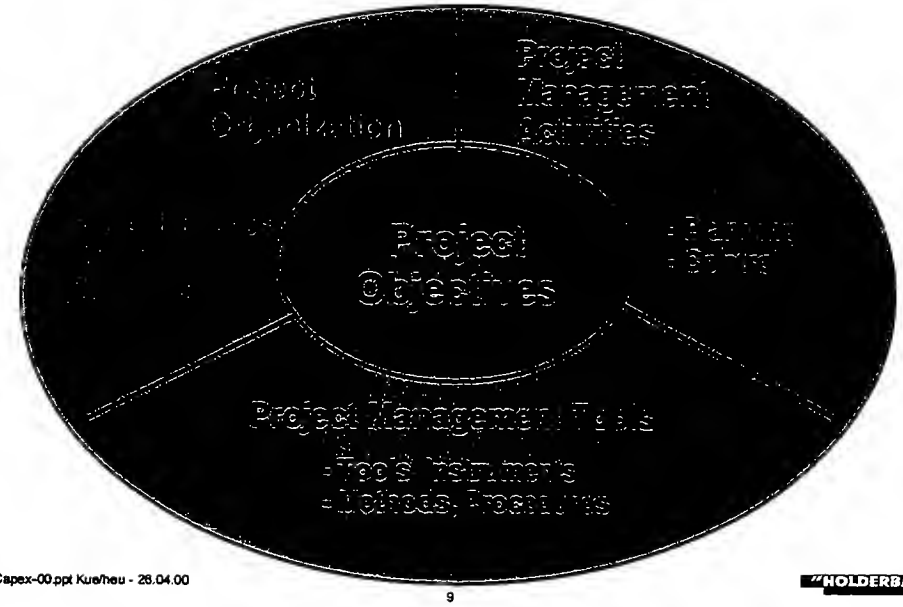
"HOLDERBANK"

Elements of Project Management



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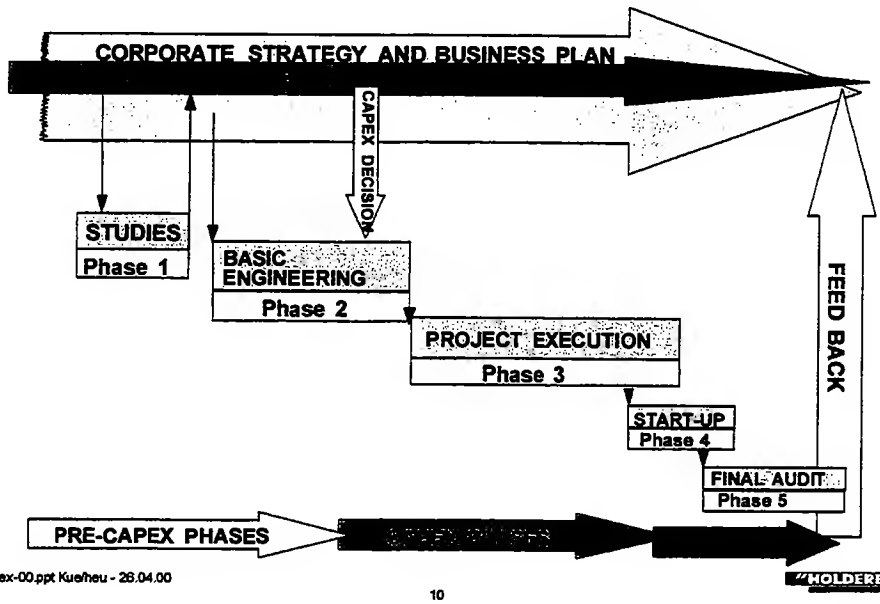
Project Organization : Project Phases



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Project Organization

Project Phases

Phase	Purpose
Study Phase	Establishing the technical, economic, legal environmental impact and often also the political base for an investment decision for a well defined project
Basic Engineering	Completion of engineering work up to the point of placing orders (letter of intent) with the main equipment supplier
Detailed Engineering/ Project Execution	Complete all detailed engineering, procurement and construction. Implement an overall project monitoring and control system for quick identification of problems and deviations from set targets and to enable fast and adequate corrective actions (schedule) (-> quality control).

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Project Organization

Project Phases

Phase	Purpose
cont.	Establish at an early date adequate organization and personnel structure to operate and maintain the plant/facilities. Assure raw materials supplies and utilities for start-up and normal operation.
Start-up/ Commissioning	Ensure that complete installations function properly and continuously under load. Ensure that contractual obligations by suppliers are met and full rated output is achieved on provisional taking-over of plant.
Final Audit	Evaluation of results achieved and recording the experience gained in the first year of operation.

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Project Organization

Set-up / Structure

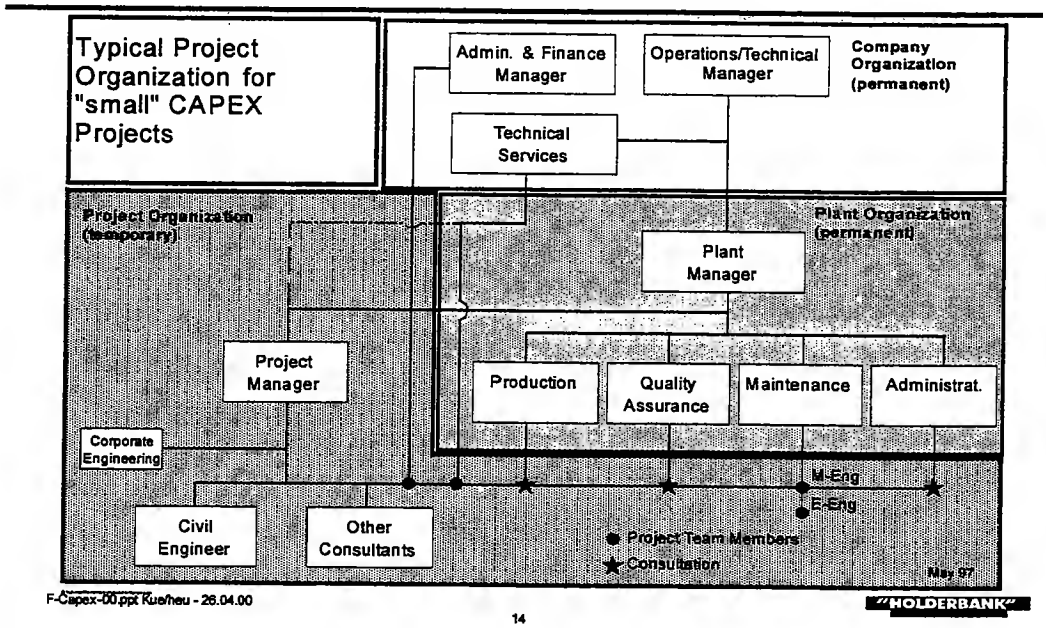
→ Internal organization of project team, assignment of tasks and responsibilities

→ Integration in plant / company / corporate organization

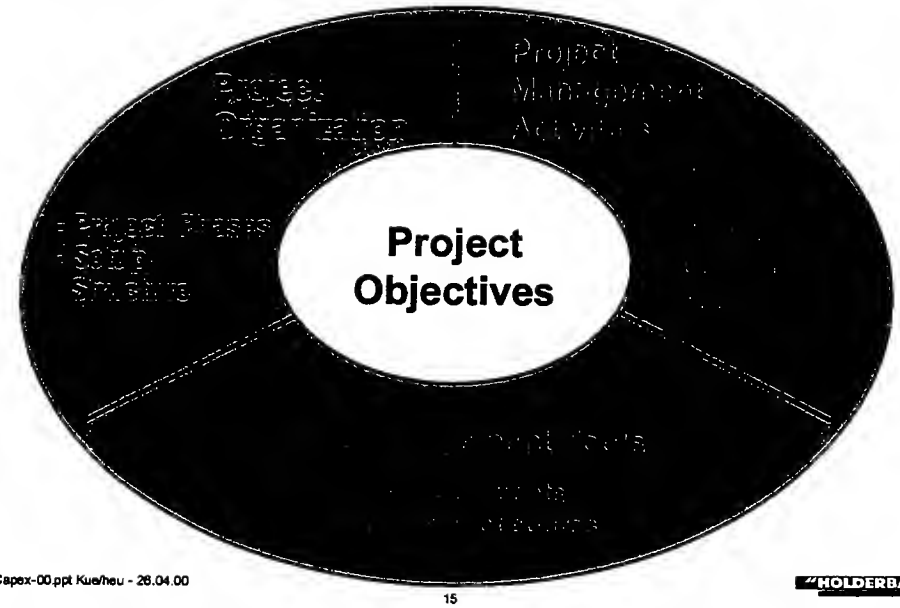
- **Organization Charts**
- **Job Descriptions**
- **Lines of Communications**,

...

Project Organization



Elements of Project Management



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Project Management Activities

Planning

→ **Establish Project Organization**

→ **Plan Project Activities**

- Sub-phases, intermediate objectives
- Individual tasks
- Coordination of individual tasks
- Requirements estimation: personnel, time
- Cost estimation
- Time schedule with milestones
- Project information system

→ **Define Project Management Tools**

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Project Management Activities

Control

→ Monitor time and cost

→ Monitor project progress, quality

→ Take measures where plan deviations occur

→ Coordinate & monitor various teams (Owner - Supplier(s) / Contractor(s))

Project Progress Report

Project Management Activities

Tools

Project Management Approach and Computer Tools
(Standardization, if possible)
- Text, Spreadsheets ...
- Computer Aided Design
- Electronic Mail
- Project Planning

→ Standard Reports

→ Standard Meetings

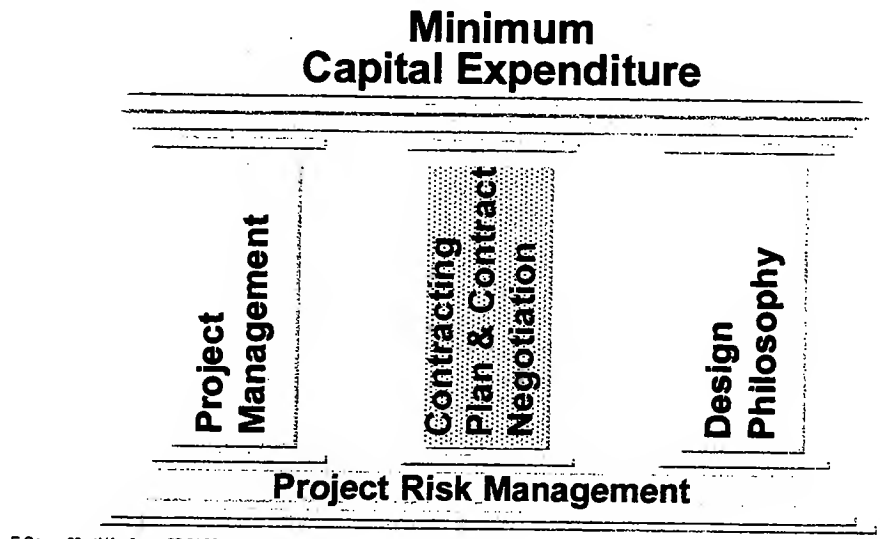
→ Lines of Communication

→ Agreements / Documents on Critical Issues

→ Project Procedure Manual

→ ...

Means to Minimize CAPEX



Contracting Plan and Contract Negotiation

- **Detail - Package - Turnkey**
  - ⇨ *to suit best the project*
- **Contract negotiation**
  - ⇨ *effective negotiation techniques*
- **Tender documents and tender evaluation**
  - ⇨ *quality determines effectivity and efficiency of contract negotiations*

Summary and Conclusion

Successful Project Implementation and Minimization of CAPEX:

→ Projects embedded in company strategy
(→ Plant Masterplan!)

→ Proper project organization : → Phases of CAPEX projects
→ Project team

→ Proper performance of standard project management activities and application of adequate project management tools

→ Appropriate contracting strategy and effective negotiation

Chapter 12

Plant Master Plan

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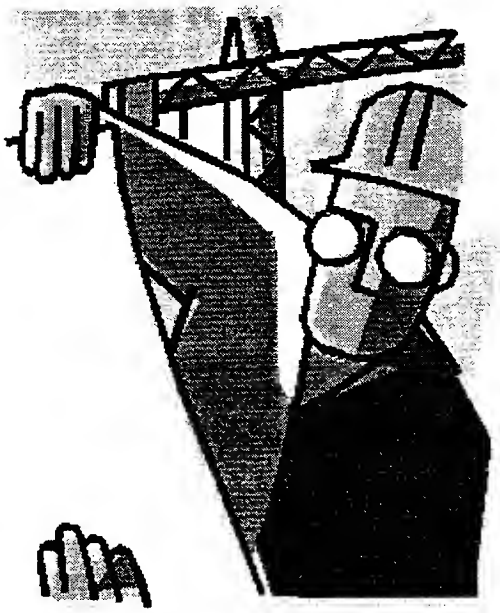
PLANT MASTER PLAN



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6. Procedure for Elaboraton of plant Master plans 297
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1. INTRODUCTION

In periods of growing demand, investors usually take the opportunity to set milestones in direction of improving productivity and expanding capacity as well as controlling cost. New plants, or suitable plant extensions, are the appropriate answers then. In period of harsh competition, shrinking market shares, high interest rates and thus tight investment budgets, it proves to be difficult to even keep up the pace of change in technology.

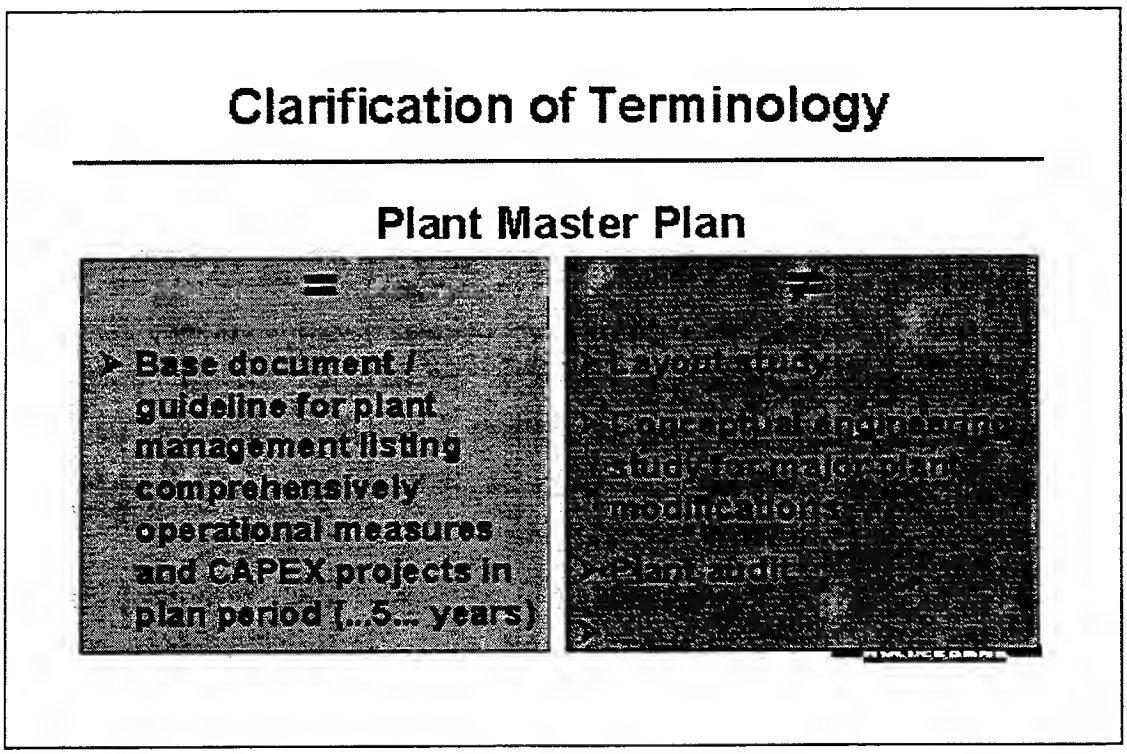
Comprehensive know-how and adequate assistance are always required to adapt the company to the changing requirements and the future needs.

As the immediate and future needs of a company are determined mainly through a dynamic process of changing parameters of the environment, a dynamic planning instrument must be used for the identification and registration of these needs.

This appropriate instrument is called Plant Master Plan.

2. CLARIFICATION OF TERMINOLOGY

Plant Master Plans are, since a long time, a well known planning tool on plant level. However, so far, there was no common understanding on purpose and content of Plant Master Plans and for this reason a vast variety of plans were called Plant Master Plans.



In recent years attempts were made in the "Holderbank" Group to standardize the Company Planning Cycle and in particular the Business Planning Process. This standardization called for a clear definition of the objectives and the contents of Plant Master Plans: The Plant Master Plan shall be the tool for plant management, comprehensively outlining the operational measures and CAPEX projects to be implemented in the plan period.

"The Plant Master Plan" is a strategic paper outlining the development of the plant in all its fields of activity, taking into account the directives of the corporate strategy and the relevant external and internal factors.

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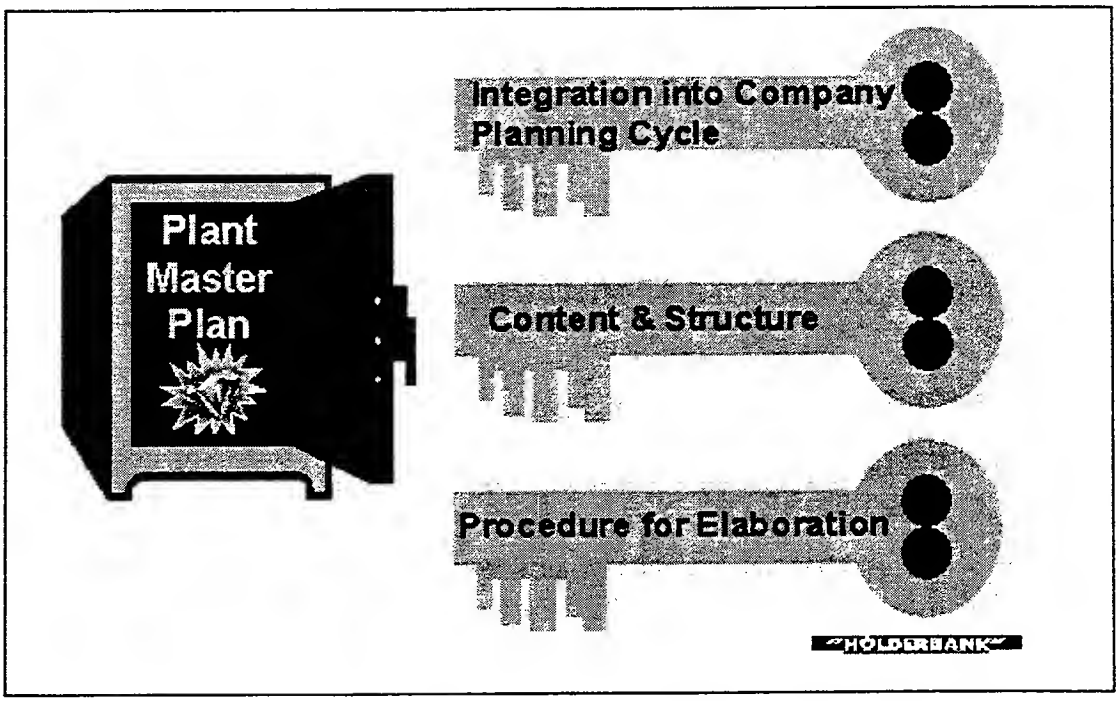
3. PLANT MASTER PLANS: COMMON PROBLEMS

A closer look at the realities of Plant Master Planning reveals following common problems:

Plant Master Plans : Common Problems

- Lack of coordination with Business Plan
- Shortcomings in sustainability
- Wishful thinking
- Strong focus on physical installations and CAPEX
- Lack of economic analysis
- Lack of involvement
- Lack of alternatives
- Lack of longterm thinking
- Problem solving mainly by CAPEX
- ...

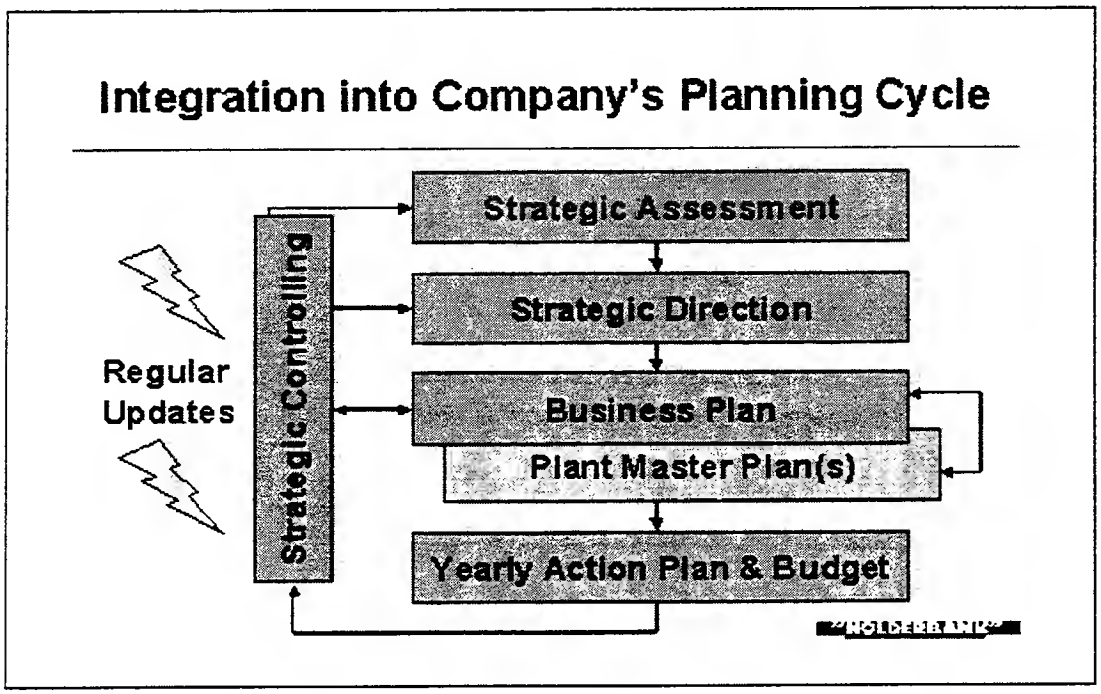
Analysis of these problems shows that they touch 1) the integration of the Plant Master Plan into the company's planning cycle, 2) the content and structure of the Plant Master Plans, and 3) the procedure for elaborating Plant Master Plans.

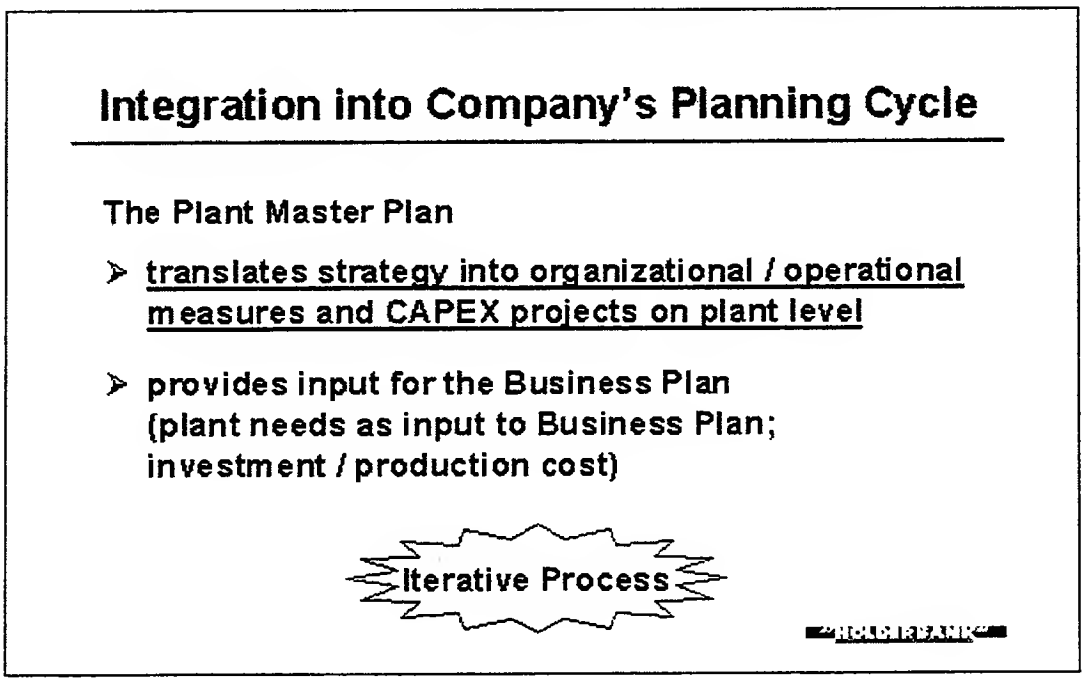


Properly addressing these three problems provides in fact the keys to successful Plant Master Planning. If these three elements are mastered, the result then is a Plant Master Plan which is a helpful and valid tool for plant management to operate and develop their plant in compliance with company strategy and Business Plan objectives.

4. INTEGRATION IN COMPANY'S PLANNING CYCLE

Elaborating a Business Plan and the corresponding Plant Master Plan(s) is an iterative process: On the one hand, the Plant Master Plan translates strategy into operational measures and on the other hand it provides input to the Business Plan, e.g. production and investment costs.





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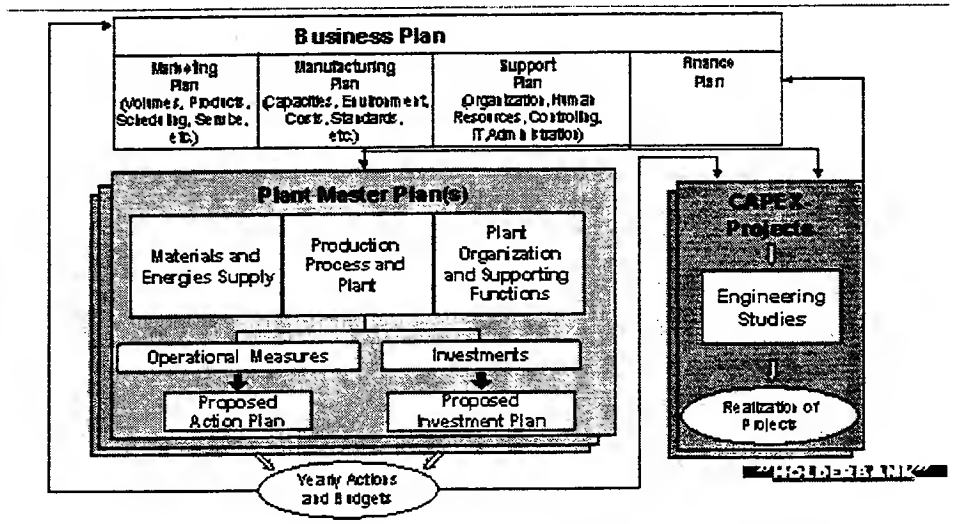
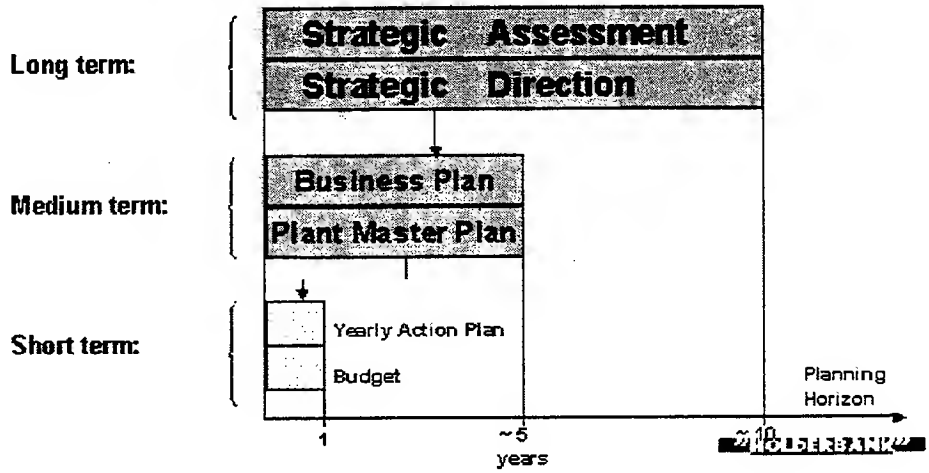
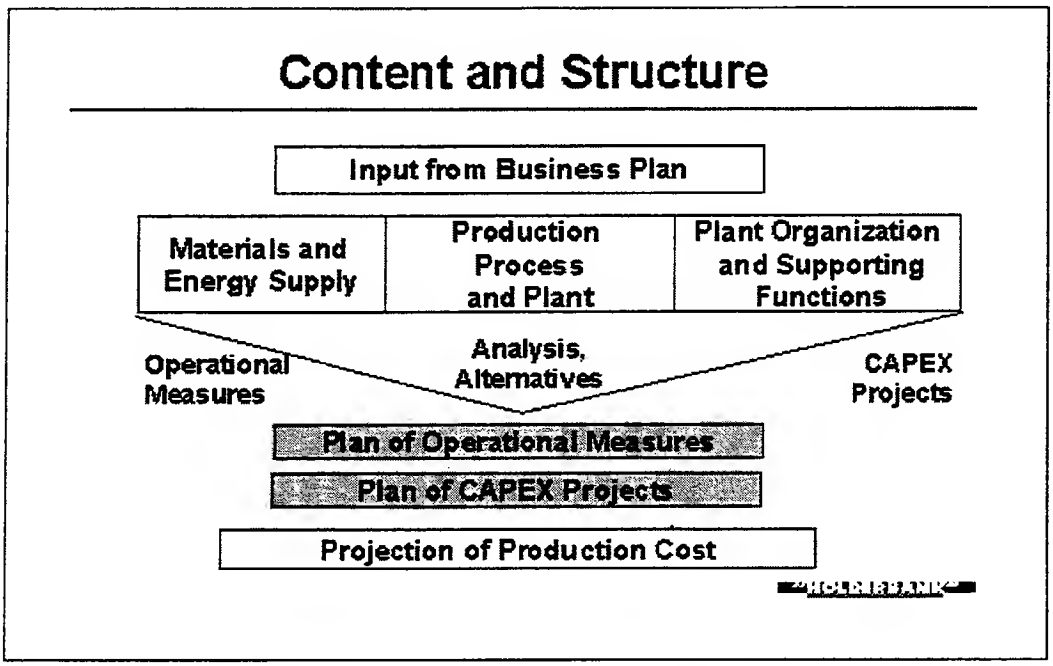
Integration into Company's Planning Cycle



Integration into Company's Planning Cycle

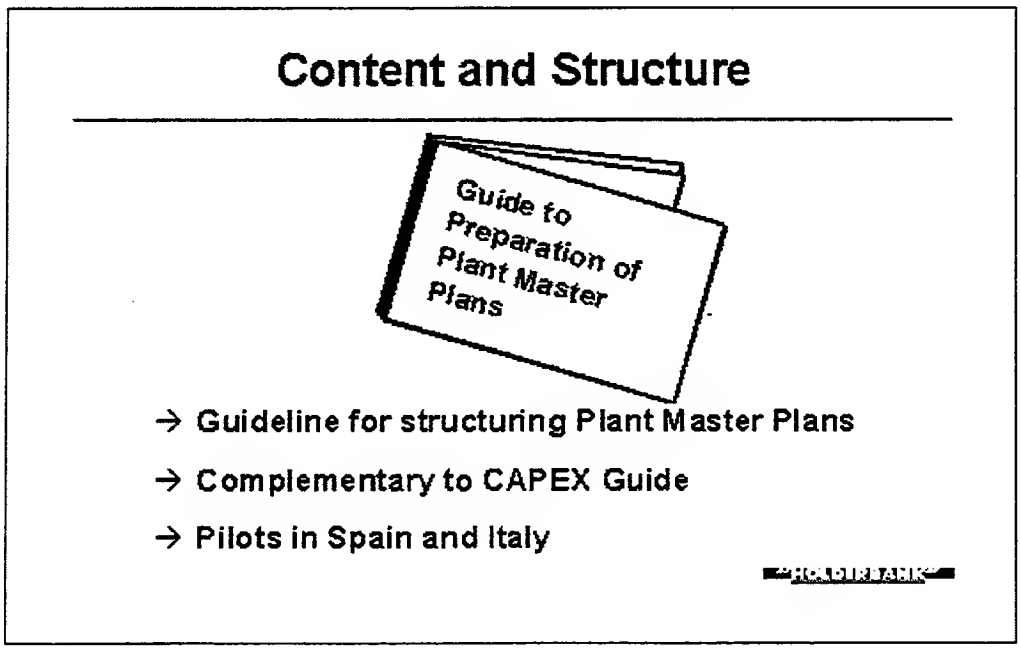


5. CONTENT AND STRUCTURE OF PLANT MASTER PLAN



The Plant Master Plan identifies gaps in the areas of materials and energy supply (from main raw materials and fuels to alternative raw materials and fuels and mineral components etc.), production process and plant (from raw material extraction to dispatch, internal and external infrastructure etc.) and plant organization and supporting functions (from organizational structure to crossfunctional processes, like maintenance, quality assurance etc.).

The Plant Master Plan clearly defines operational measures and CAPEX projects and indicates priorities, cost impacts and financial benefits.



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6. PROCEDURE FOR ELABORATON OF PLANT MASTER PLANS

Procedure for Elaboration of Plant Master Plans

General approach for individual plant sections and functions:

1. Assessment of status quo
   Implications of Business Plan
2. Alternatives: elaboration, evaluation, selection
   (optimum exploitation of existing facilities)
3. Operational / organizational measures
4. CAPEX projects

} Description, Cost / Return, Priority

To assure comprehensiveness of the Plant Master Plan, the team put together to elaborate such a Plant Master Plan must be composed interdisciplinarily, providing knowledge and experience from all relevant areas and disciplines.

On the other hand such a team should include not only plant staff but also representatives of the company management to assure that those who will be responsible for its implementation are really committed to the proposals and priorities of the resulting Plant Master Plan.

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Including external experts, from HMC and/or from other companies and their plant, further helps to challenge the status quo and to consider a variety of alternatives (Faster Learning Organization). Priority focus must be on optimum exploitation of existing installations in order to minimize CAPEX.

Procedure for Elaboration of Plant Master Plans

Targets:

- Comprehensive approach
- Challenge of status quo
- Consideration of a variety of alternatives
- Commitment

Team configuration:

- Interdisciplinary
- Plant
- Company
- HMC
- Other companies, plants

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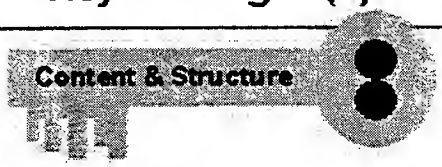
7. CONCLUSION AND KEY MESSAGES

Key Messages (1)

The Plant Master Plan is an integral element of the planning cycle, translating strategy into operational measures and CAPEX projects, on a five year horizon.

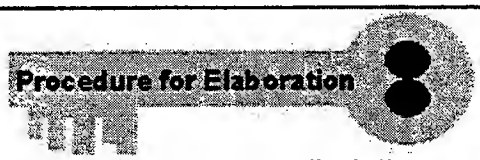
Key Messages (2)



The Plant Master Plan ...

- identifies gaps in all relevant aspects in the areas of materials and energies, production process and organization
- clearly defines operational measures and CAPEX projects and indicates priorities, cost impacts and financial return

Key Messages (3)



- Elaboration requires an interdisciplinary team from the plant, the company, HMC and other companies / plants to assure a comprehensive approach, challenging the status quo, considering a variety of alternatives and get the commitment.
- Priority focus on optimum exploitation of existing installations in order to minimize investments.

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Chapter 13

Project Risk Analyses

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Project Risk Management

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1. **SLIDE 1: CEMENT SEMINAR 2000**

Project Risk Management

D. Bach **Business Risk Management**
H. Würmli **"Holderbank" Engineering Switzerland**

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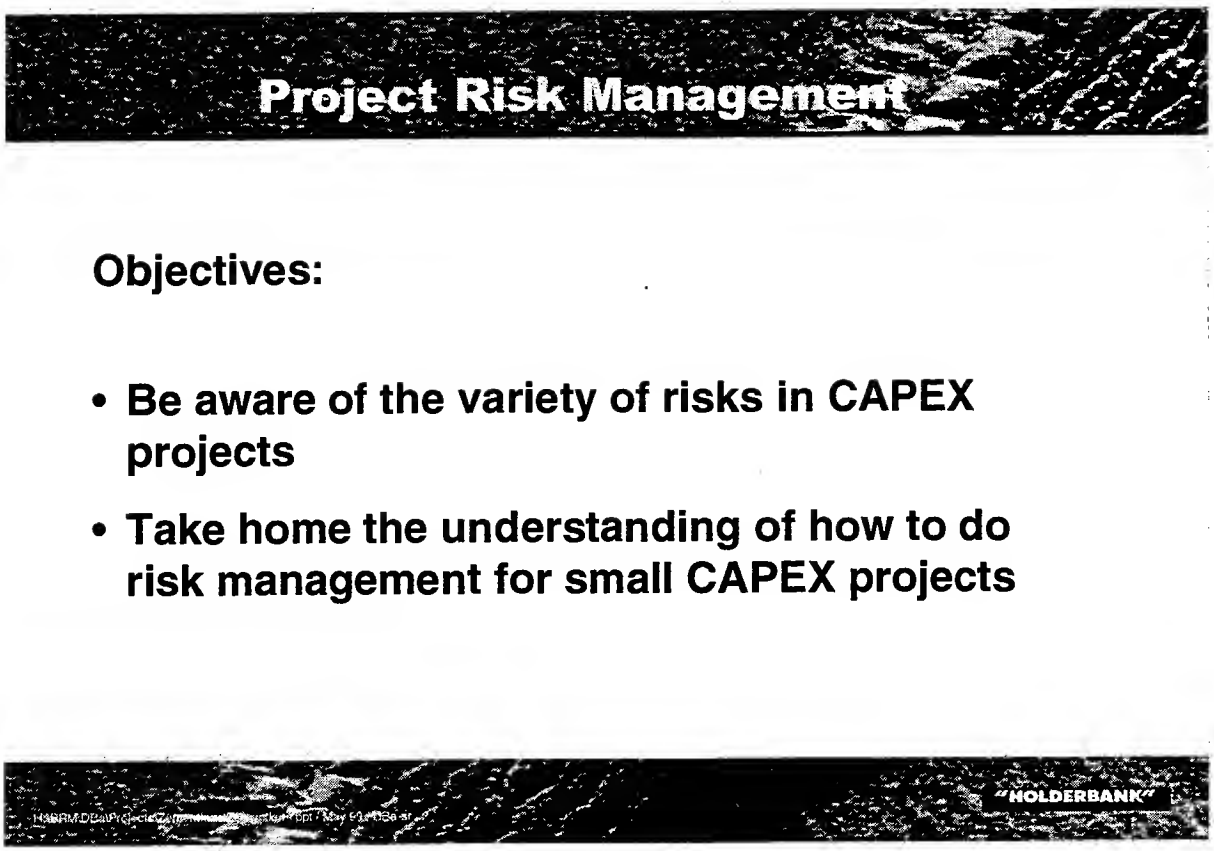
2. SLIDE 2: PROJECT RISK MANAGEMENT



3. SLIDE 3: BUSINESS RISK MANAGEMENT

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4. SLIDE 4: BUSINESS RISK MANAGEMENT

Target Areas:

⇨Group Companies:
- Cement
- Aggregates
- Concrete
- Waste
- .....

⇨Financial Holdings
⇨HMC
⇨"Holderbank" Group
⇨**Projects (incl. CAPEX Projects)**

5. **SLIDE 5: CASE**

Kiln Shell Replacement in a Lime Plant

Scope: To remove two approximate three meter lengths of kiln shell and replace with one six meter length of second hand kiln shell on site, and realign the kiln to remove the "wobble" at the front of the kiln.

- The project shall be executed during an extended kiln shut-down.
- Clients shall not be affected by this project in a manner unacceptable to them.

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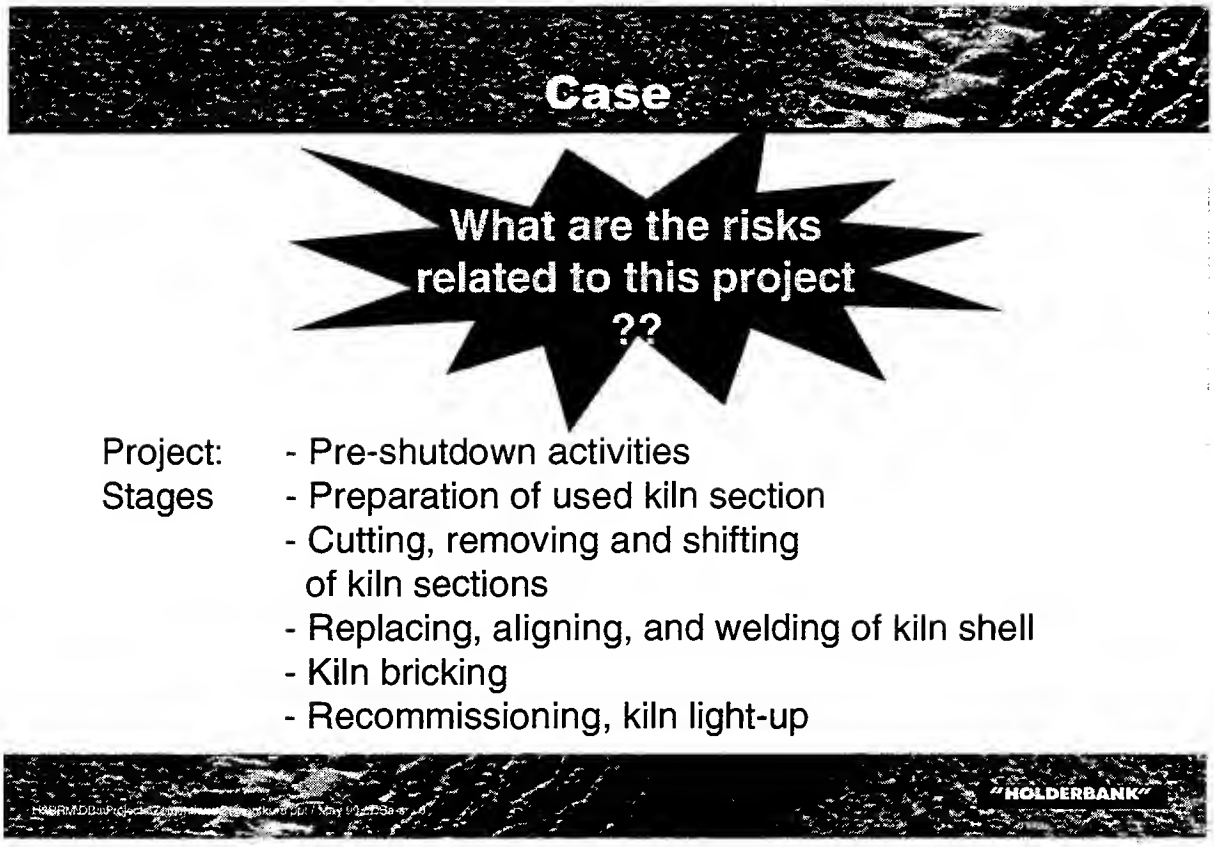
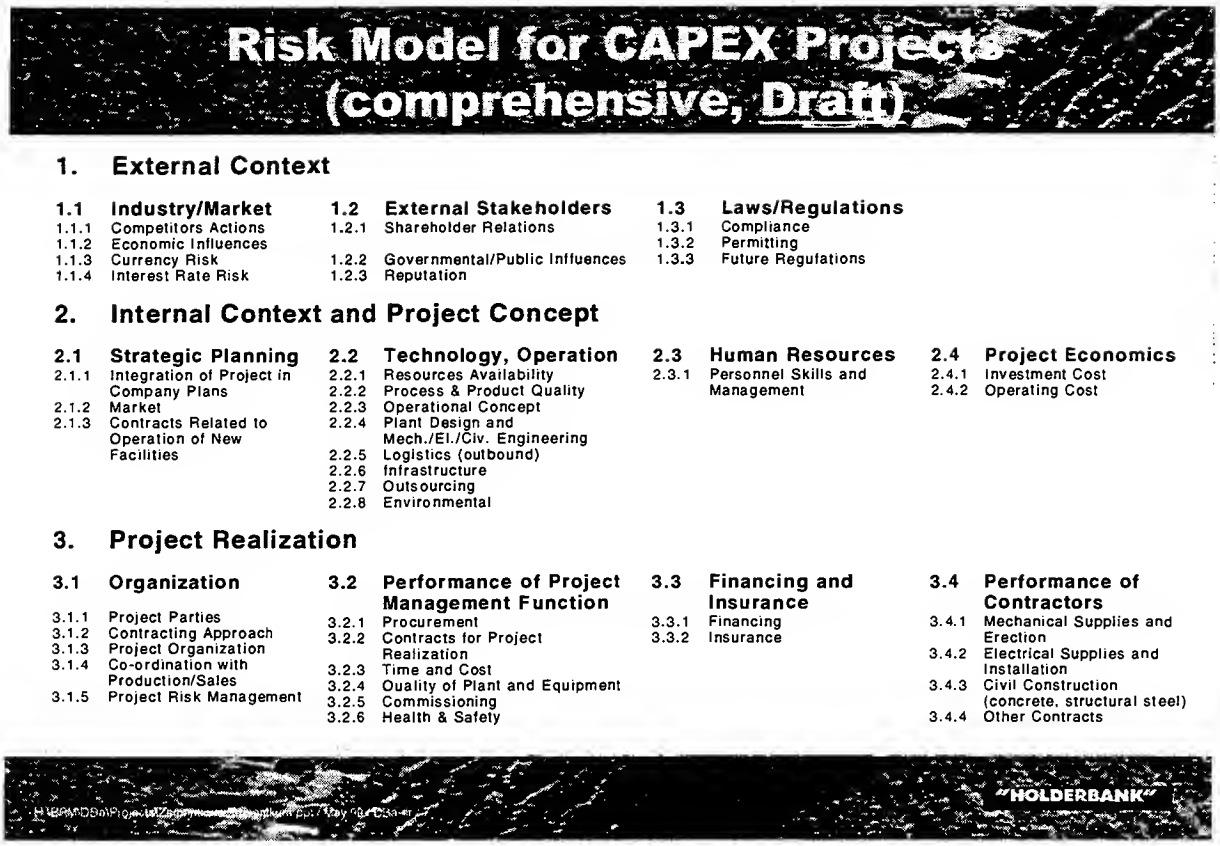
6. SLIDE 6: CASE



7. SLIDE 7: RISK MODEL FOR CAPEX PROJECTS



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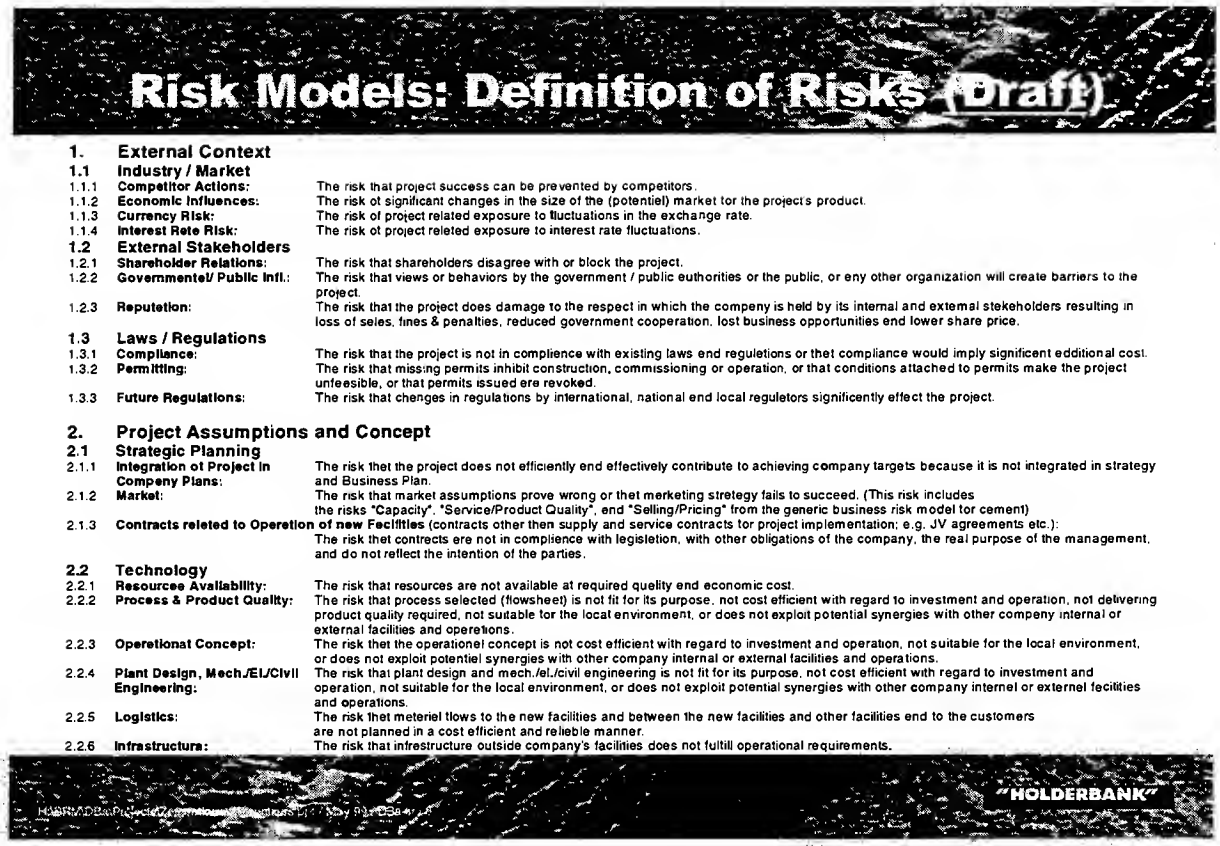
8. SLIDE 8: RISK MODELS – DEFINITION OF RISKS



Risk Models: Definition of Risks (Draft)

1. External Context

1.1 Industry / Market

1.1.1	**Competitor Actions:**	The risk that project success can be prevented by competitors.
1.1.2	**Economic Influences:**	The risk ot significant changes in the size of the (potentiel) market tor the project's product.
1.1.3	**Currency Risk:**	The risk of project related exposure to fluctuations in the exchange rate.
1.1.4	**Interest Rate Risk:**	The risk ot project releted exposure to interest rate fluctuations.

1.2 External Stakeholders

1.2.1	**Shareholder Relations:**	The risk that shareholders disagree with or block the project.
1.2.2	**Governmentel/ Public Infl.:**	The risk that views or behaviors by the government / public euthorities or the public, or eny other organization will create barriers to the project.
1.2.3	**Reputetion:**	The risk that the project does damage to the respect in which the compeny is held by its internal and extemal stekeholders resulting in loss of seles, fines & penalties, reduced government cooperation, lost business opportunities end lower share price.

1.3 Laws / Regulations

1.3.1	**Compliance:**	The risk that the project is not in complience with existing laws end reguletions or thet compliance would imply significent eddition al cost.
1.3.2	**Permitting:**	The risk that miss:ng permits inhibit construction, commissioning or operation, or that conditions attached to permits make the project unfeesible, or that permits issued ere revoked.
1.3.3	**Future Regulations:**	The risk that chenges in regulations by international, national end local reguletors significently effect the project.

2. Project Assumptions and Concept

2.1 Strategic Planning

2.1.1	**Integration ot Project in Compeny Plans:**	The risk thet the project does not efficiently end effectively contribute to achieving company targets because it is not integrated in strategy and Business Plan.
2.1.2	**Market:**	The risk that market assumptions prove wrong or thet merketing stretegy fails to succeed. (This risk includes the risks "Capacity", "Service/Product Quality", end "Selling/Pricing" from the generic business risk model tor cement)
2.1.3	**Contracts releted to Operetion of new Feclfities** (contracts other then supply and service contracts tor project implementation; e.g. JV agreements etc.):	The risk thet contrects ere not in complience with legisletion, with other obligations of the company, the real purpose of the management, and do not reflect the intention of the parties.

2.2 Technology

2.2.1	**Resourcee Availability:**	The risk that resources are not available at required quelity end economic cost.
2.2.2	**Process & Product Quality:**	The risk that process selected (flowsheet) is not fit for its purpose, not cost efficient with regard to investment and operation, not delivering product quality required, not suitable tor the local environment, or does not exploit potential synergies with other compeny internal or external facilities and operetions.
2.2.3	**Operetionat Concept:**	The risk thet the operationel concept is not cost efficient with regard to investment and operation, not suitable for the local environment, or does not exploit potentiel synergies with other company internal or external facilities and operations.
2.2.4	**Plant Design, Mech./El./Civil Engineering:**	The risk that plant design and mech./el./civil engineering is not fit for its purpose, not cost efficient with regard to investment and operation, not suitable for the local environment, or does not exploit potential synergies with other company internel or externel fecilities and operations.
2.2.5	**Logistics:**	The risk thet meteriel tlows to the new facilities and between the new facilities and other facilities end to the customers are not planned in a cost efficient and relieble manner.
2.2.6	**Infrastructura:**	The risk that infrestructure outside company's facilities does not fultill operational requirements.

9. SLIDE 9: RISK MODELS – DEFINITIONS OF RISKS

Risk Models: Definition of Risks (Draft)

2.2.7 **Outsourcing:** The risk that foreseen split between in-house and outsourced resources is not optimum or that this split is difficult to correct later on.
2.2.8 **Environmental:** The risk that the project does not comply with the company's own stated environmental policy and / or with public expectations.

2.3 Human Resources
2.3.1 **Personnel Skills and Management:** The risk that new facilities will not be operated successfully due to HR reasons.

2.4 Economic Analysis
2.4.1 **Investment Cost:** The risk that investment cost exceed estimates and budgets resp.
2.4.2 **Operating Cost:** The risk that operating cost exceed estimates.

3. Project Realization

3.1 Organization
3.1.1 **Project Parties:** The risk that parties involved in a project (company, contractors, consultants, authorities, financial institutions etc.) are not able to carry out the responsibilities assigned to them or taken over by them.
3.1.2 **Contracting Approach:** The risk that the contracting approach does not adequately consider project environment, project charactenstics and the requirements and capabilities of the company.
3.1.3 **Project Organization:** The risk that overall project organization as well as project organizations of parties involved in the project are not adequate to carry out the responsibilities assigned to them.
3.1.4 **Co-ordination with Production/Sales:** The risk that project execution seriously affects the company's medium-/long-term position in the market.
3.1.5 **Project Risk Management:** The risk that (key) risks materialize due to missing or ineffective Project Risk Management procedures.

3.2 Performance of Project Management Function
3.2.1 **Procurement:** The risk that the procurement procedure (from call for tenders to final negotiations and conclusion of contracts) does not lead to optimum cost - benefit conditions of contracts related to equipment supplies and construction/erection.
3.2.2 **Contracts related to Equipment Supplies and Construction/Erection:** That risk that supply and service contracts suffer major legal, commercial or other deficiencies.
3.2.3 **Time and Cost:** The risk that time schedules and cost estimates are not maintained due to missing or ineffective control functions, deficiencies in contracts or "scope creep" during project execution, or due to external influences such as natural disasters, strikes etc.
3.2.4.1 **Quality of Plant and Equip.:** The risk that target quality levels are not achieved due to insuff. contractual specification and inadequate control in workshops and on site.
3.2.5 **Commissioning:** The risk that commissioning is not completed within commissioning period foreseen.
3.2.6 **Health & Safety:** The risk that health hazards, injuries, illnesses, or fatalities occur during project execution.

3.3 Financing and Insurance
3.3.1 **Financing:** The risk that the project cannot be financed at feasible conditions, that the set-up of the financing is delaying the project execution, or that that restrictive conditions (e.g. contractors selection) are imposed.
3.3.2 **Insurance:** The risk that project implementation is not adequately protected by insurances.

3.4 Performance of Contractors
3.4.1 **Mechanical Supplies & Erection:**
3.4.2 **Electrical Supplies & Installation:**
3.4.3 **Civil Construction (concrete, structural steel):**
3.4.4 **Other Contracts:**

Identical definition for all "Performance of Contractors" Risks:
The risk that contractors do not perform in accordance with their contracts and thereby jeopardize project success (time, quality, cost).

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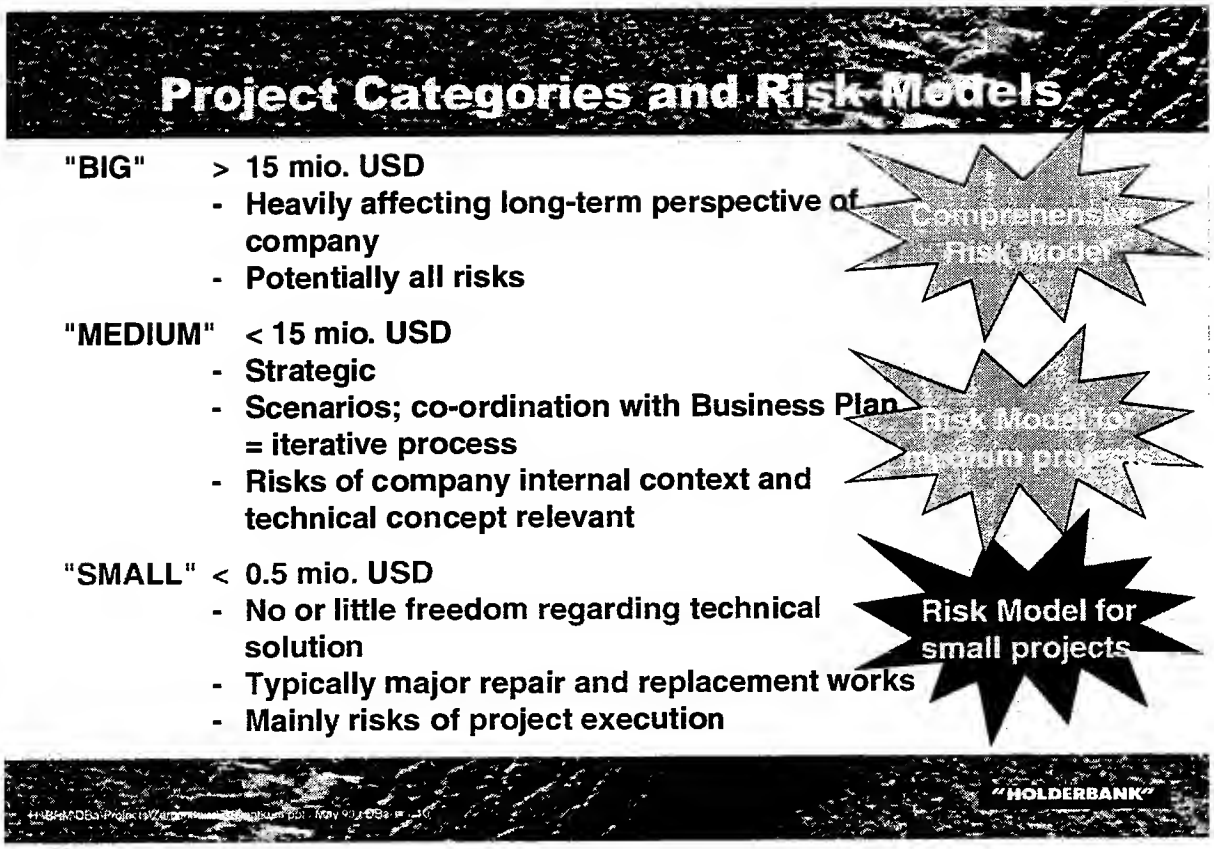
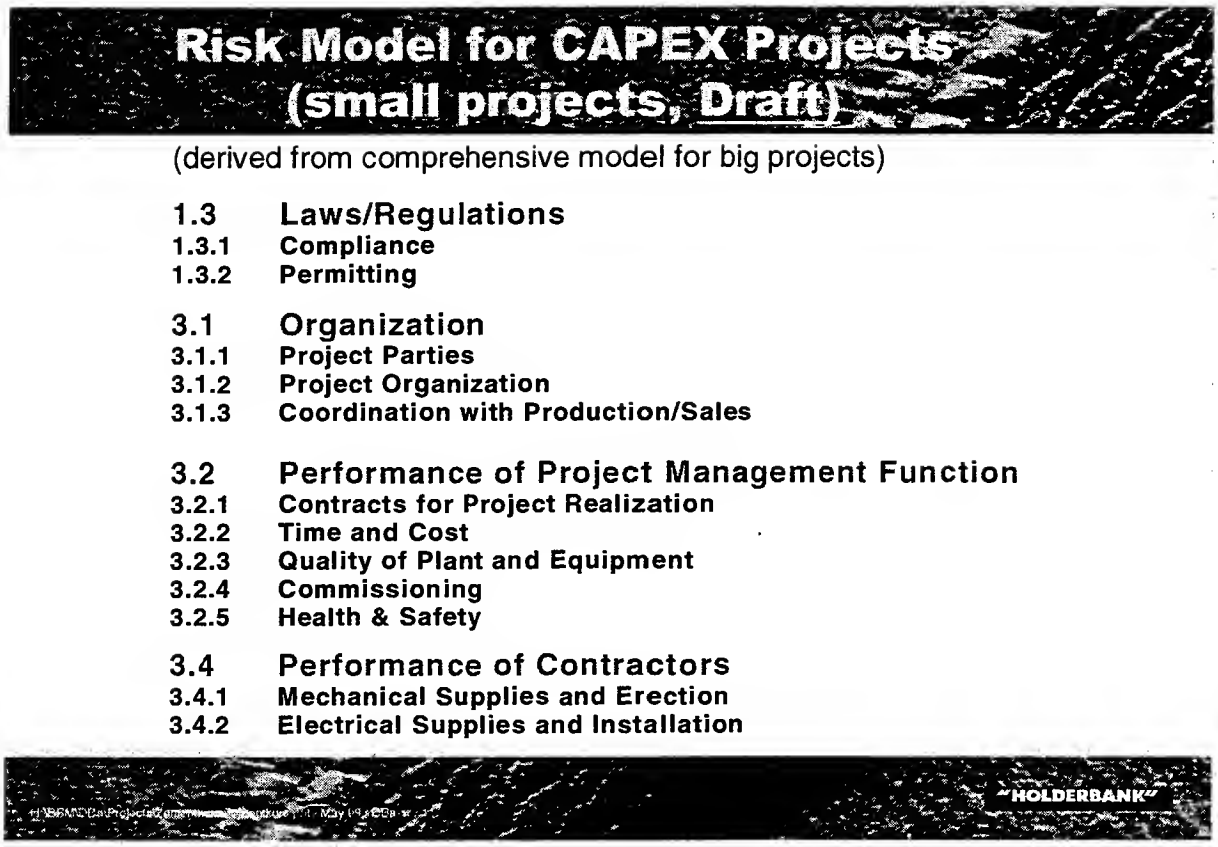
10. SLIDE 10: PROJECT CATEGORIES AND RISK MODELS



11. SLIDE 11: RISK MODEL FOR CAPEX PROJECTS



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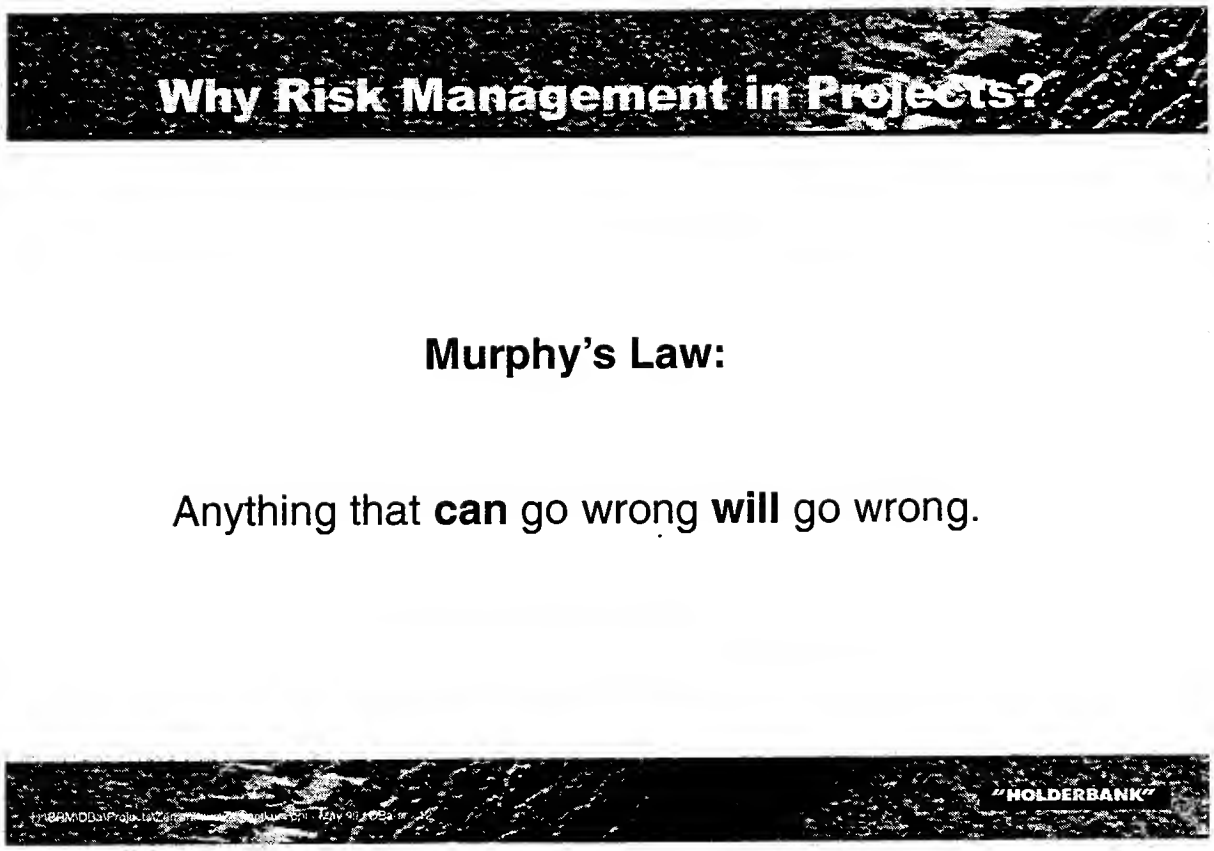
12. SLIDE 12: WHY RISK MANAGEMENT IN PROJECTS



13. SLIDE 13: WHY RISK MANAGEMENT IN PROJECTS

Why does Murphy's Law not always prove right in projects?

➡ **Project Management**

Good project management considers risks and takes measures to control and manage these risks.

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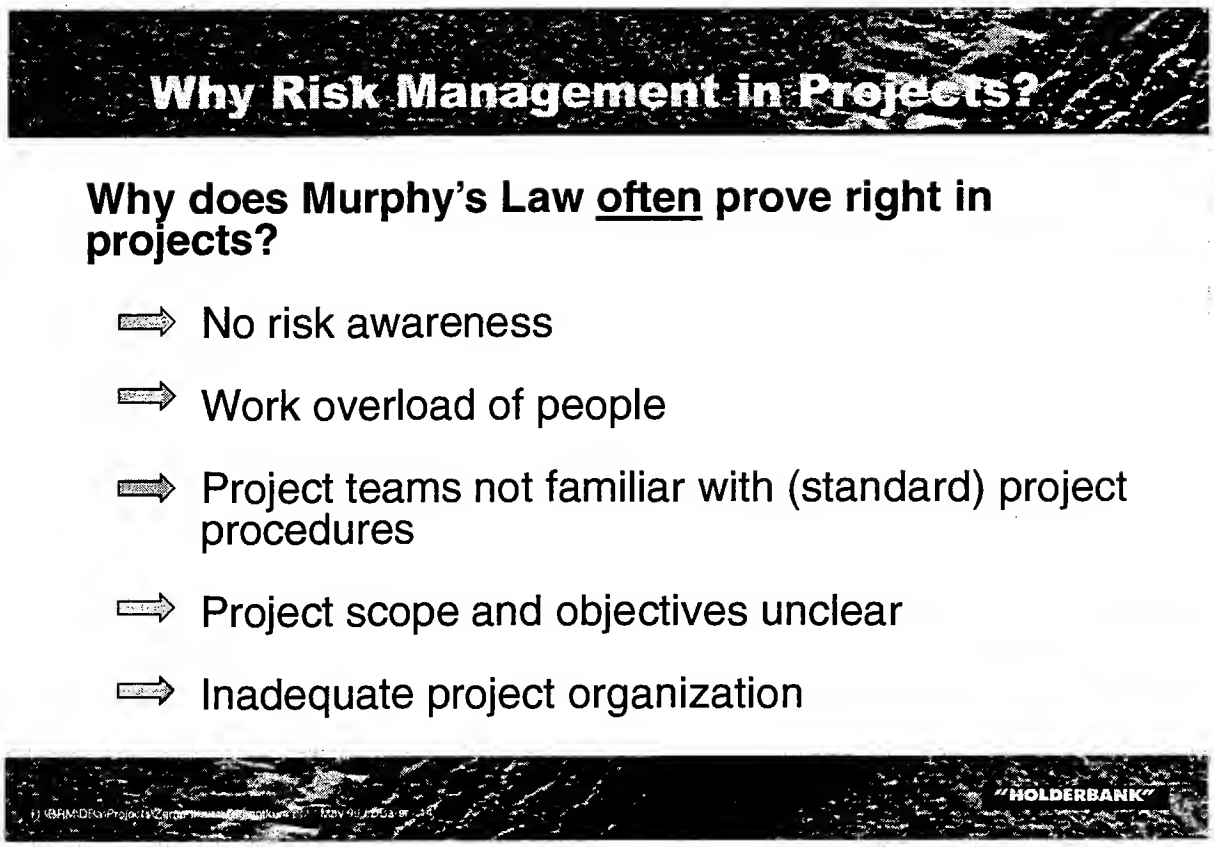
14. SLIDE 14: WHY RISK MANAGEMENT IN PROJECTS



15. SLIDE 15: WHY RISK MANAGEMENTIN PROJECTS

The answer:

Business Risk Management for Projects (BRMP)

BRMP for Project Management

~ Safety rope for mountain climbing

~ Safety net for high-wire walk

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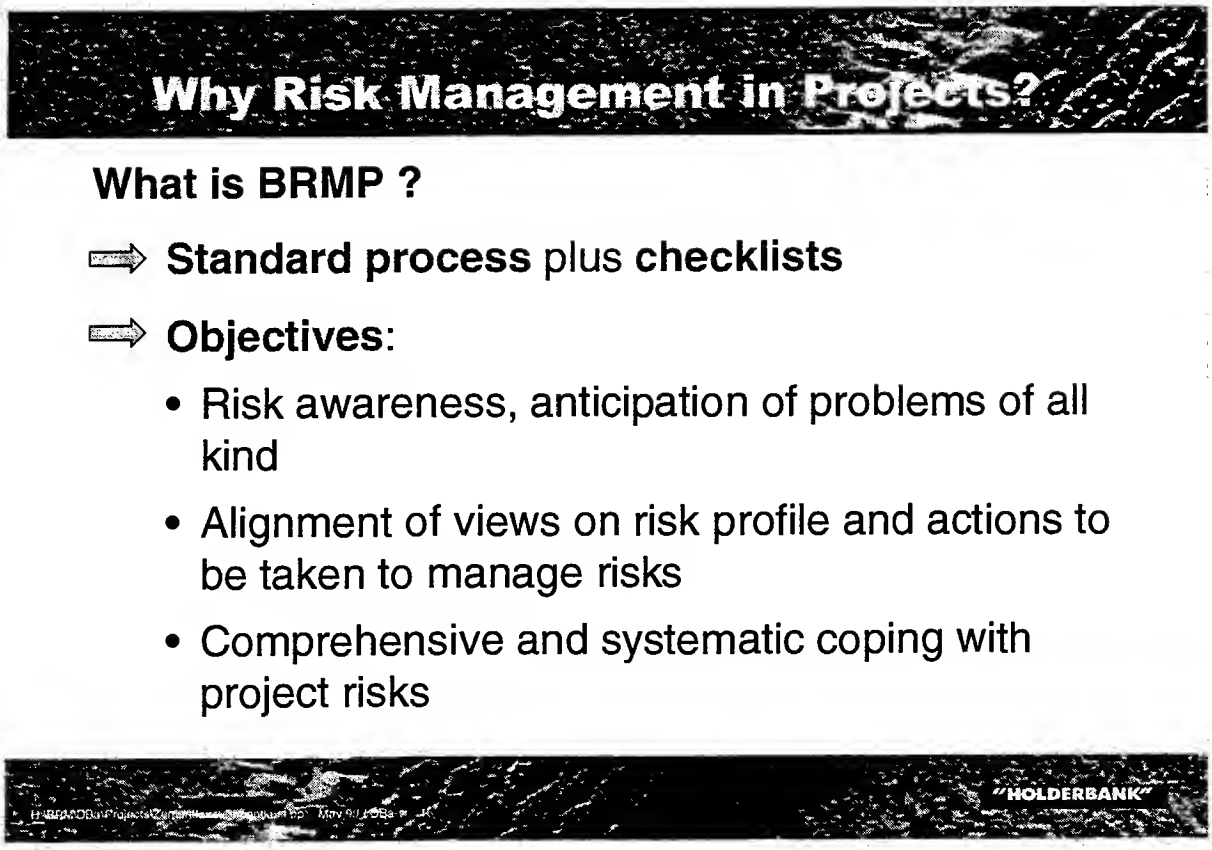
16. SLIDE 16: WHY RISK MANAGEMENT IN PROJECTS



17. **SLIDE 17: BRMP FOR SMALL PROJECTS**

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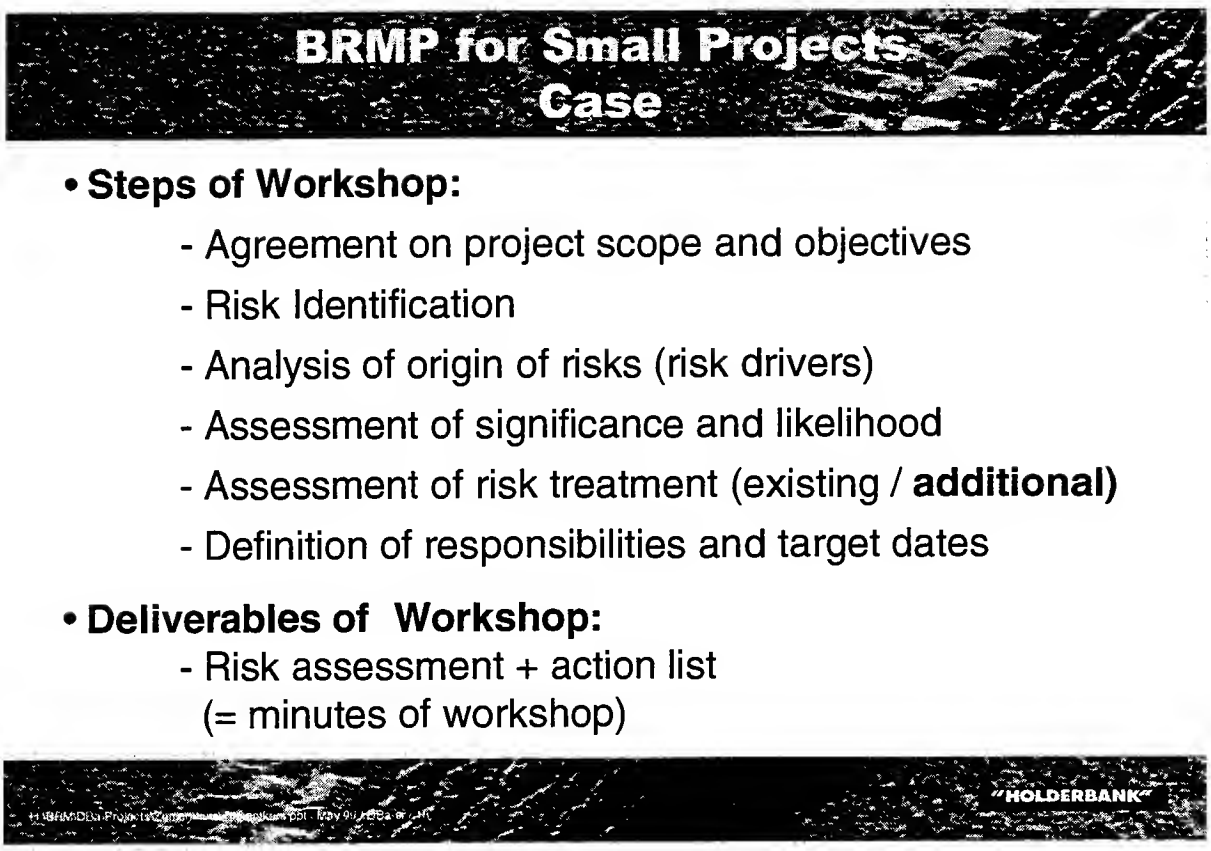
18. SLIDE 18: BRMP FOR SMALL PROJECTS



19. SLIDE 19: BRMP FOR SMALL PROJECTS

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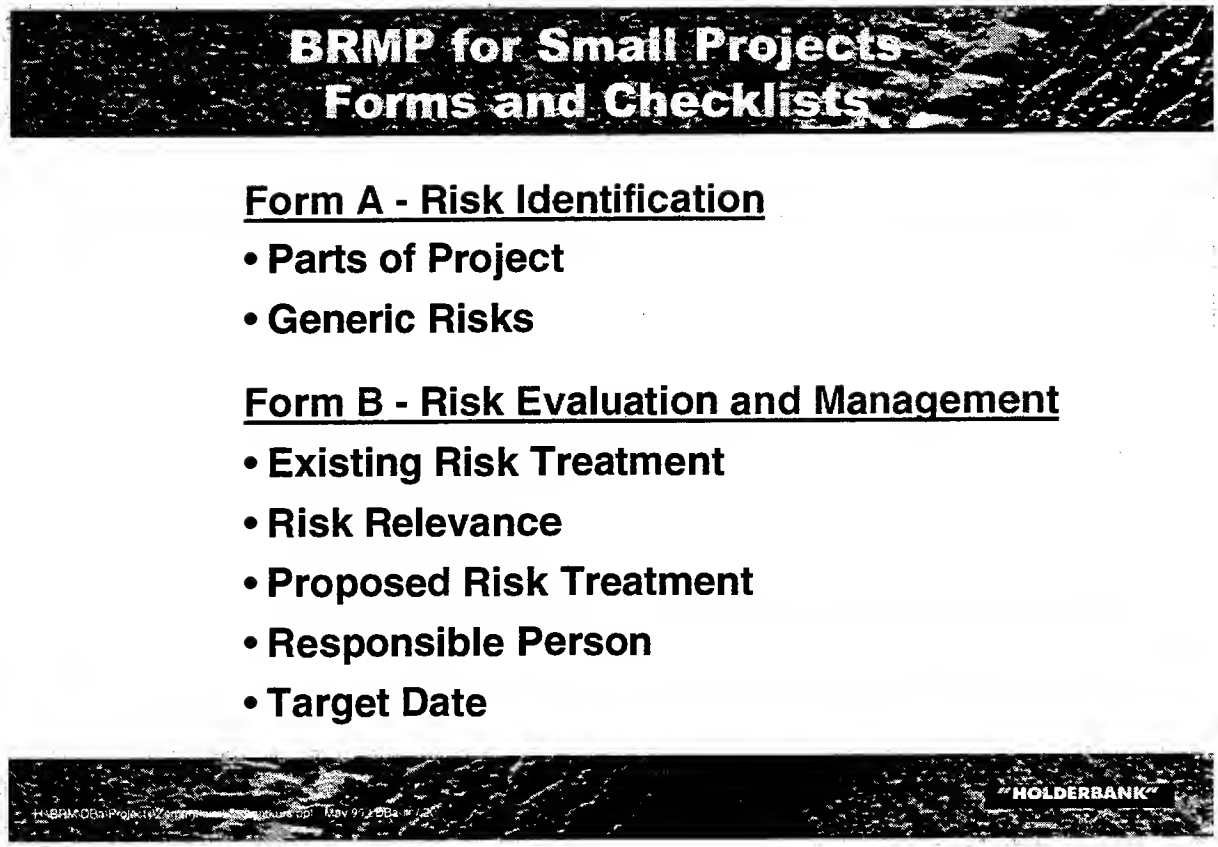
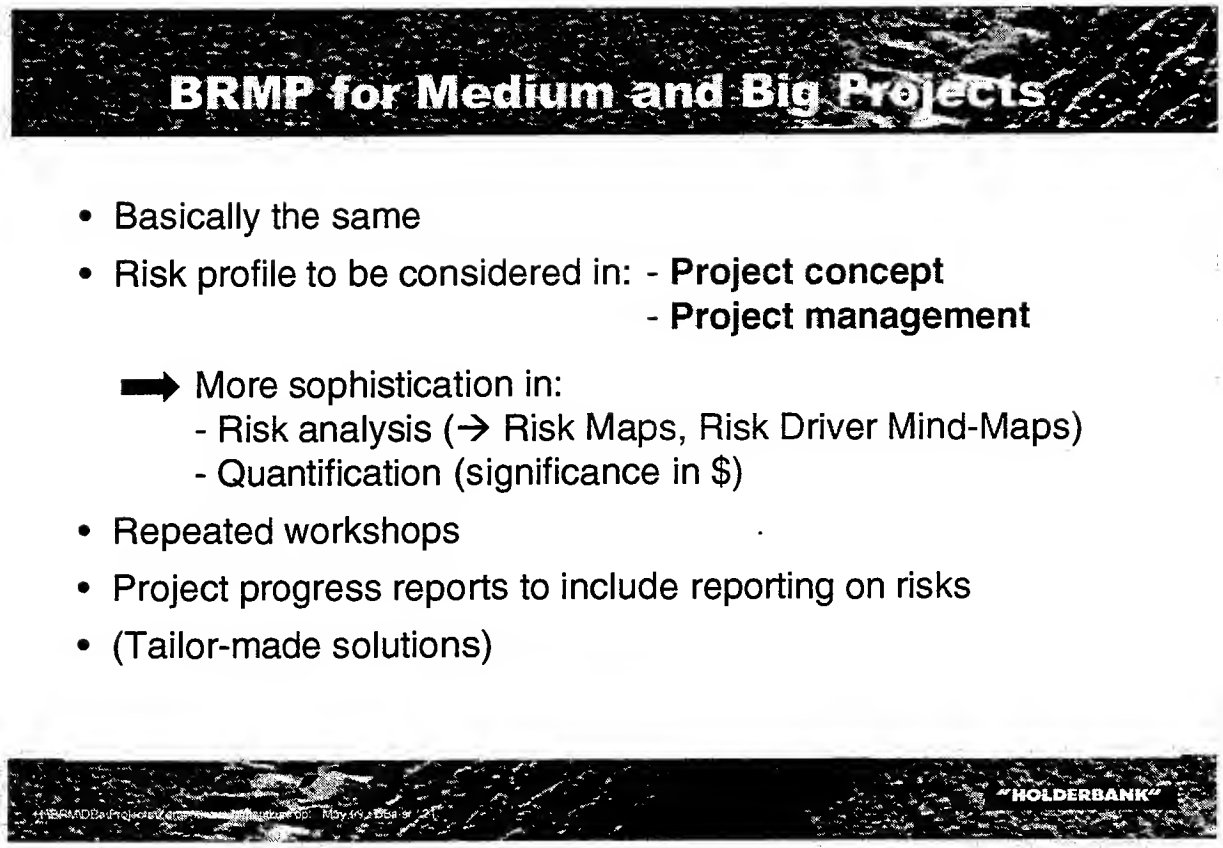
20. SLIDE 20: BRMP FOR SMALL PROJECTS



21. SLIDE 21: BRMP FOR MEDIUM AND BIG PROJECTS



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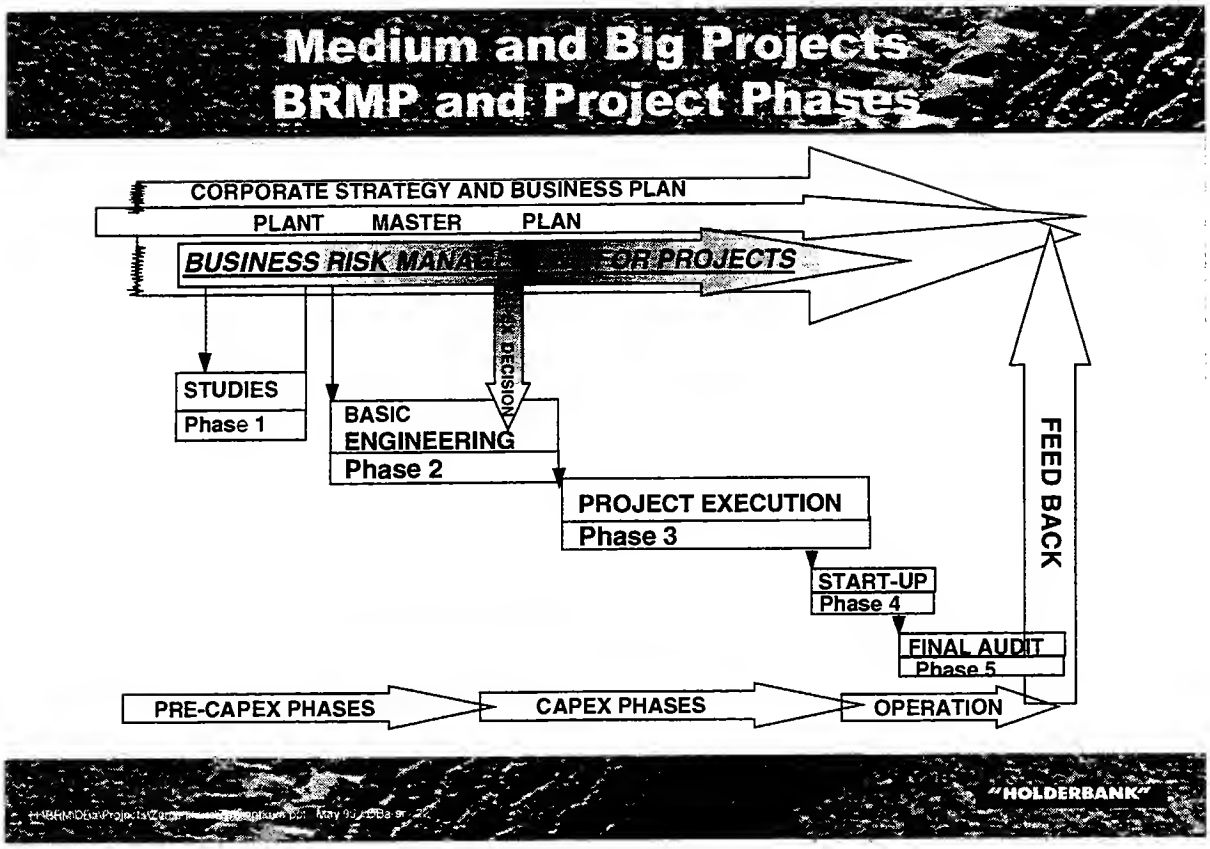
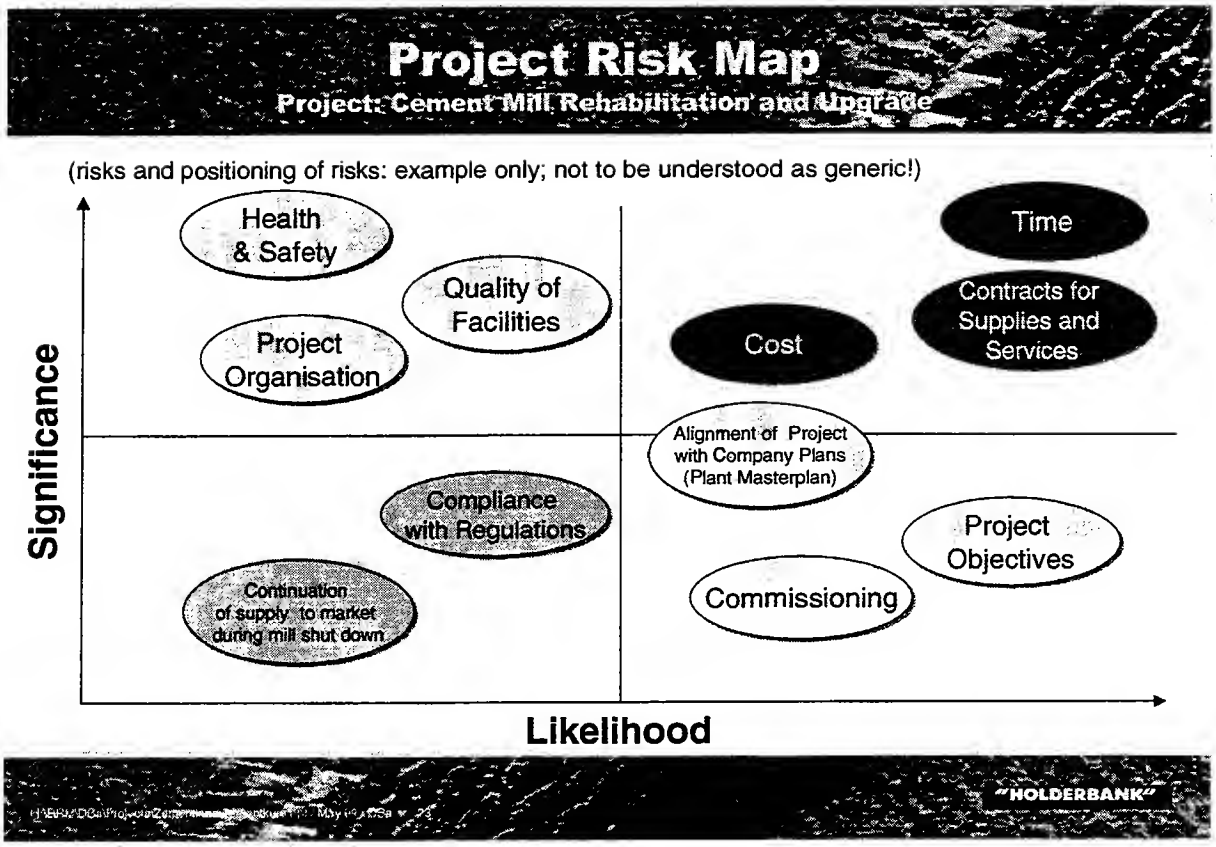
22. SLIDE 22: MEDIUM AND BIG PROJECTS



23. SLIDE 23: PROJECT RISK MAP



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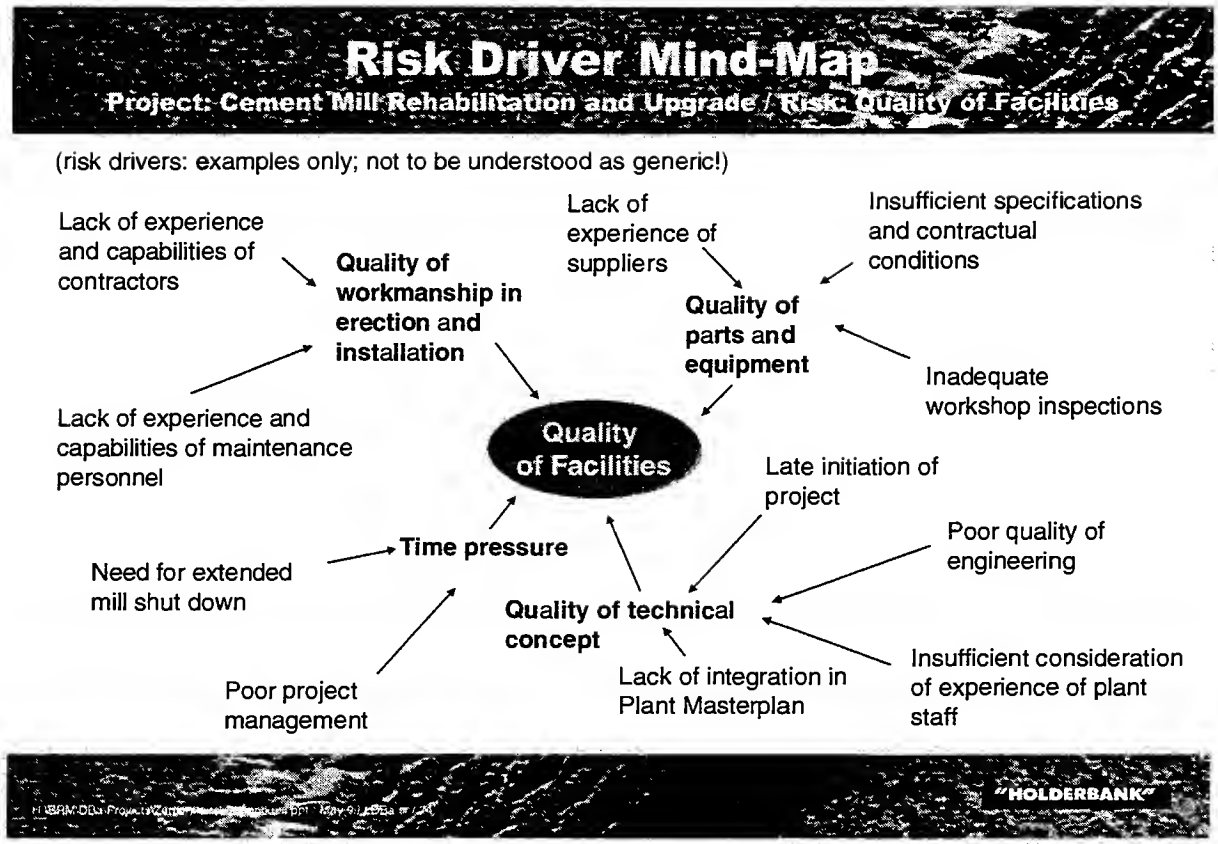
24. SLIDE 24: RISK DRIVER MIND-MAP



Business Risk Management for Projects (BRMP)

Checklists and Forms

for

Small Projects

May 1999
Business Risk Management
"Holderbank" Engineering Switzerland

Project: *Kiln Shell Replacement* **Form A1**
Risk Identification *(examples only)*

Generic Risks ➢ Parts of Scope/ Project Stages ➘	Project Parties	Project Organisation	Co-ordination with Production/ Sales	Compliance	Permitting	Contracts for Project Realisation
Pre-shutdown activities			2) Lime stockpile may be too low at the commencement of the shutdown. 4) The shutdown plan and schedule may be inadequate.			9) The composition of the existing kiln material is not known and this will impact on the welding material to be used.
Preparation of used kiln section						
Cutting, removing and shifting of kiln sections				12) Noise may be generated during shutdown activities.		
Replacing, aligning,and welding of kiln shell				11) Significant welding will be required and this may pose a hazard or health risk to personnel. 12) Noise may be generated during shutdown activities.		
Kiln bricking						
Recommissioning, kiln lightup						

Project: *Kiln Shell Replacement* **Form A2**
Risk Identification *(examples only)*

Generic Risks ➢ **Parts of Scope/ Project Stages** ➘	**Time**	**Cost**	**Quality**	**Commissioning**	**Health and Safety**	**Performance of Contractors**
Pre-shutdown activities						
Preparation of used kiln section					5) No formal safety plan by the contractor and no formal transfer of risk.	
Cutting, removing and shifting of kiln sections	1) The contractor's equipment may not arrive in suitable time.				5) No formal safety plan by the contractor and no formal transfer of risk. 6) Some of the work, e.g. debricking, may require work in a confined space. 11)Significant welding will be required and this may pose a hazard or health risk to personnel.	7) The moving piece of the kiln may fall and be damaged.

Project: *Kiln Shell Replacement* **Form A2**
Risk Identification *(examples only)*

Generic Risks ➢ **Parts of Scope/ Project Stages** ↓	**Time**	**Cost**	**Quality**	**Commissioning**	**Health and Safety**	**Performance of Contractors**
Replacing, aligning,and welding of kiln shell	1) The contractor's equipment may not arrive in suitable time.		8) The kiln may not be cleaned prior to welding leading to problems with the weld. 9) The composition of the existing kiln material is not known and this will impact on the welding material to be used. 10) Incorrect alignment of the kiln may occur leading to ongoing operational problems.		5) No formal safety plan by the contractor and no formal transfer of risk.	7) The moving piece of the kiln may fall and be damaged. 10)Incorrect alignment of the kiln may occur leading to ongoing operational problems.
Kiln bricking	3) Late delivery or non-delivery of refractory material.					
Recommissioning, kiln lightup						

Project: *Kiln Shell Replacement* **Form B**
Risk Evaluation and Management *(examples only)*

Part of Scope / Project Stage: *Pre-shutdown Activities*

Risk and Risk Description	Existing Risk Treatment	Risk Relevance*		Proposed Risk Treatment	Respons. Person	Tar-get Date
		Signifi-cance L,M,H	Likeli-hood L,M,H			
1) The contractor's equipment may not arrive in suitable time.		M	M	Confirm that the equipment has been shipped and will arrive at site prior to the scheduled shutdown commencement time.		
2) Lime stockpile may be too low at the commencement of the shutdown.	Current kiln output is considered to be sufficient to meet this target stockpile by the commencement of the shutdown. A customer survey has been undertaken to gauge expected customer demands during the shutdown period.	M	H	Identify and agree an alternative supply and develop a supply agreement. Continue customer surveys. Ensure that customer silos are filled. Ensure that ISO containers are available and loaded prior to the shutdown. Consult with specialist in this regard. Develop and implement a silo management strategy to ensure that off-specification material in silos at the time of the shutdown is minimized. Schedule production to maximize lime stockpiles at the start of the shutdown.		
3) Late delivery or non-delivery of refractory material.	It is known that the overseas manufactured refractory has been shipped and is currently on route and delivery details confirmed. Local manufacturer representative has been asked to confirm delivery date.	M	L	Do not start the shutdown unless sufficient refractory is available.		

* CONSIDERING EXISTING RISK TREATMENT!

Project: *Kiln Shell Replacement* **Form B**
Risk Evaluation and Management *(examples only)*

Risk and Risk Description	Existing Risk Treatment	Risk Relevance*		Proposed Risk Treatment	Respons. Person	Tar-get Date
		Signifi-cance L,M,H	Likeli-hood L,M,H			
4) The shutdown plan and schedule may be inadequate.	An experienced contractor who has undertaken other kiln shell replacements elsewhere in the vicinity has been engaged. This contractor has contributed to the development of the plan. Owner also has experience with this contractor.	M	L	Ongoing review of the shutdown plan and schedule.		
5) No formal safety plan by the contractor and no formal transfer or risk.	The contractor employs experienced crews and has safely undertaken similar jobs in the past.	M	L	Discuss the option of development of a safety plan and work method statements for key work practices with the contractor. Follow through insurance and workers' compensation situation for the contractor who is a foreigner. Investigate owner's exposure to a major claim if an incident should occur.		
6) Some of the work, e.g. debricking, may require work in a confined space.	Site confined space procedure. Site lockout procedure.	M	L	Confirm whether the kiln interior will be required. Develop a specific procedure for kiln entry during the shutdown.		
7) The moving piece of the kiln may fall and be damaged.	The strategy for moving the kiln piece has been designed by the contractor to be suitable for movement without failure.	H	M	Check if crane support will be used throughout the process. If not, obtain further information from contractor on how the moving piece will be supported.		
8) The kiln may not be cleaned prior to welding leading to problems with the weld.		M	L	Ensure that clear instructions are given during debricking to ensure kiln is clean enough prior to welding.		

* CONSIDERING EXISTING RISK TREATMENT!

Project: *Kiln Shell Replacement* **Form B**
Risk Evaluation and Management *(examples only)*

Risk and Risk Description	Existing Risk Treatment	Risk Relevance*		Proposed Risk Treatment	Respons. Person	Tar-get Date
		Signifi-cance L,M,H	Likeli-hood L,M,H			
9) The composition of the existing kiln material is not known and this will impact on the welding material to be used.	All likely materials required to be used in welding are known to be available.	M	L	Undertake sampling and analysis as soon as possible once the kiln material is available for sampling.		
10)Incorrect alignment of the kiln may occur leading to ongoing operational problems.	The contractor is expert in undertaking kiln shell replacements. Owner's personnel will confirm alignment prior to commencement of welding (included in work schedule).	H	L	No additional risk treatments are recommended.		
11)Significant welding will be required and this may pose a hazard or health risk to personnel.	Contractor expertise and existing welding procedures.	L	L	Ensure that contractors are briefed regarding owner's requirements in relation to welding. Ensure that ID fan is used to ventilate the interior of the kiln.		
12)Noise ma y be generated during shutdown activities.	There is a restriction of night time operations at site. Owner's personnel is well aware of noise issues and takes control action to reduce noise. A complaint procedure is in place for dealing with environmental complaints, e.g. noise complaint.	L	L	No additional risk treatments are recommended.		

* CONSIDERING EXISTING RISK TREATMENT!

Project: ... ...
Risk Identification

Form A1

Generic Risks ➢ Parts of Scope/ Project Stages ➘	Project Parties	Project Organisation	Co-ordination with Production/ Sales	Compliance	Permitting	Contracts for Project Realisation

Project: ... ... **Form A2**

Risk Identification

Generic Risks ➢ Parts of Scope/ Project Stages ➘	Time	Cost	Quality	Commissioning	Health and Safety	Performance of Contractors

Project: ... ... **Form B**

Risk Evaluation and Management

Part of Scope / Project Stage: ... ...

Risk and Risk Description	Existing Risk Treatment	Risk Relevance*		Proposed Risk Treatment	Respons. Person	Tar-get Date
		Signifi-cance L,M,H	Likeli-hood L,M,H			
1) ...						
2) ...						
3) ..						
4) ...						
5) ...						

* CONSIDERING EXISTING RISK TREATMENT! L, M, H = LOW, MEDIUIM, HIGH